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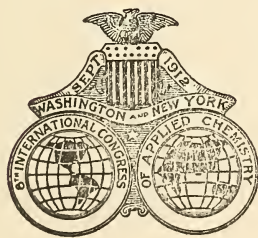# CONGRESS
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SECTION IIIa: METALLURGY AND MINING

VOL. III

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VOLUME III

SECTION IIIa: METALLURGY AND MINING

CONTENTS

DEVELOPMENT OF THE AMERICAN WATER JACKET, LEAD BLAST FURNACE

By R. C. Canby

New York, N. Y.

The American water jacket blast furnace may properly be said to be the outgrowth of the lead smelting practice which originated at Eureka, Nevada. The development is distinctly traceable from these beginnings into Utah and Colorado, where were still further developed and established those details of construction and practice which have so universally stamped themselves upon the art of lead smelting.[1]

It is not necessary in a paper of this scope to enter into a discussion of the actual "first use" of the "water jacket,"[2] nor to

[1] An article in Mines, Mills & Furnaces makes special mention of great smelting losses (at Eureka, Nevada) running up to 30–40%. This was possibly true in the very early days, but it is only fair to say that the old slags were finally resmelted, and at the time I (Frank Robbins) left Eureka, slags carrying over 1 oz. silver per ton, or over 1.5% lead were considered bad.

[2] During the discussion, pro and con, as to the priority of the water jacket which was carried on in the technical journals in about 1885, Mr. Frank Robbins of Los Angeles called attention (in a private letter) to water jacket furnace Stetefeldt erected, as constructor. The furnace was smelt galena with a silicious gangue. The attempt to do this without roasting and without proper fluxing resulted in disaster. Of course the wiseacres at once condemned the ores of the district as "rebellious," "refractory," etc.

My father who was the first to smelt successfully, was wise enough to select the carbonates and oxidized ores for his experiment; and, in consequence, was at once successful. His first stack (See Fig. No. 1) was a draft furnace and the ore ran through it like butter, and his judgment was vindicated.

At the beginning of an operation the slag was run off into ordinary iron wheelbarrows, and the bullion moulds were made of sheet-iron, folded and reinforced by a heavy wire rim. Primitive as were these expedients they served in the campaign of education. His success encouraged others, and the work advanced and improved.

When I went to Eureka in 1870 the original draft stack of my father had given way to two small blast furnaces.

Within a period of four or five years there were twelve or more smelting plants erected. It is worthy of note as showing the economy of concentration,

arrange chronologically a list of the various smelting plants. Neither does it seem necessary to examine into historical data as to the earliest smelting in Virginia and the New England Colonies, nor to include Missouri-Kansas-Illinois district since the practice of that district was peculiar to itself, and does not seem to have had any direct influence upon the development of the furnace and practice which one has in mind when considering the American water jacket furnace.

We will see also that it has not been the growth of the blast furnace itself that has been wholly, or perhaps mostly, responsible for the advancement made in the practice of lead smelting.

It is convenient, and appears reasonably correct, to consider this advancement in periods of a decade, and ignoring one or two previous furnaces, to commence with the first attempts at smelt-

that when Eureka had reached the zenith of ore production, two plants were alone treating the whole output of the mines. These were, for the time, large and well-managed institutions (The Eureka Consolidated and The Richmond Consolidated). It did not take the mine owners long to learn that a smelting charge (cost) at a well constructed large plant was less than the operating cost in a small individual one. It is also worth noting that eventually the two large plants made an agreement and fixed the smelting charge which was all the miner could stand, and just low enough to prevent him from selling his ore at Salt Lake.

In 1870, with the exception of Stetefeldt, no skilled metallurgists were in evidence. There were two or three itinerant assayers whose apparatus, and perhaps knowledge, did not extend beyond the range of the crucible and cupel. Careful analyses of the slags were, however, made with hourly frequency on the point of a long handled shovel.

I think it was Hahn who dubbed it, "the Age of Muscular Metallurgy."

My father had a smelter foreman named Pritchard who was the son of a Welshman and a Mexican woman, racially a metallurgical marvel. He had a Mexican and Indian crew, and "feast days" were vigorously, if not religiously observed. During one of these shut-downs, Pritchard whitewashed the interior of the furnace with a mixture of bone-ash, showing the difficulty there was in getting good refractory material for furnace lining. Pritchard had got his idea from the resisting property of the cupel.

About this time three bright young German metallurgists came to camp, Karl von Leibenau, Albert Arents, and Otto H. Hahn.

When Bateman & Buel sold their holdings to The Eureka Consolidated Company, that Company engaged Arents to remodel and increase the plant. Hahn came about the same time but I am uncertain about his early work.

These men were the first to introduce dust chambers in the camp. This

ing in Nevada in 1870, where the precourser of the blast furnace, the "draft furnace," was used coincidently with the first crude sandstone blast furnaces. These furnaces were usually two or three feet square, and those which had blast were supplied by Sturtevant fans, or small Root blowers. They had sheet-iron or water-cooled tuyeres. When the Richmond Consolidated works were rebuilt, Edward Probert introduced his hydrocycle or round water jacket furnace, with intruding water jacketed tuyeres.[3]

The advent of Arents and Hahn in Eureka may be truly said to mark the inception of lead blast furnace smelting. In Arents' siphon tap, first adopted in Eureka, and in the reestablishment of the furnace bosh, we have the most important details of purely lead furnace practice, and we may consider that these were the accomplishments of our first period.

With the early years of the second decade, 1880, we find a decided advance has been made, not only in the furnace con-

was at the Consolidated Eureka. Arents invented and patented the siphon tap leadwell.

When the Richmond Company was formed in 1871, Hahn, with Leibenau as his assistant I think, planned and constructed the smelting works, and for many years this plant was regarded as a model, and metallurgical pilgrims came from Utah and Colorado to get inspiration or to steal ideas.

I believe Hahn and Arents brought the plans of the rectangular Raschette furnaces with them from Germany. At any rate their construction was ever on this design.

In rebuilding the Richmond Plant which was destroyed by fire in 1878, round furnaces were introduced. (These were undoubtedly the patented hydrocycle of Edward Probert.)

Eureka was so at the mercy of the Central Pacific Railway that the bullion was allowed to accumulate, and finally the Richmond determined not to submit to extortion, and built a refinery of its own at the works, shipped dore bars to the Coast, and accumulated the lead. The refining process was a modification of the Pattison process, called the Luce & Rozan described in a "Treatise on Metallurgy" by Frederick Overman, published in 1852.

(Extract from personal letter from Frank Robbins of Los Angeles, California.)

Vol. XIV "Mineral Industry," page 409, states that E. Daggett in 1872 had cast water jackets for round furnace of Winnamuck Mining Company, Bingham Canon, Utah.

[3] See Note 1, also "Metallurgy of Argentiferous Lead," M. Eissler.

struction and arrangement of plants, but an important change had also occurred in business methods by the establishment of centralized commercial smelting works. In Eureka, almost immediately after the first success, there sprang up a great number of small furnaces, each mine having its own local furnace, and even after the centralization of the smelting into the two larger plants, the Richmond and the Eureka Consolidated, these works were still essentially adjuncts to their respective mines, treating largely their own ores. But about the commencement of our second decade, one finds the local or individual furnaces replaced by more complete centralized plants, such as those at Salt Lake City, Leadville, Denver, and Pueblo.

At Salt Lake City, from the beginnings at Eureka, were developed to a very full degree the technical and business details, so that with the beginning of the decade, 1880, most all of the features of our modern practice were already discernible.

The small square sandstone furnaces had been replaced by the rectangular water jacket furnace, mostly 36″ wide, and from 60 to 80″ long, at the tuyeres. The jackets were of cast iron with tuyeres entering at the line of junction, between the jackets. Somewhat later, when the Holden works were built at Denver, Mr. Franz Cazin assumed the responsibility of placing the tuyeres in the middle of the jacket, thus reducing to three the number of different kinds of jackets necessary for a complete set. With this change of the position of the tuyeres, and the dovetailing of the cast lugs for connecting bolts, and the uniform thickness of the cast iron and other improvements of detail, many of which were also devised by Cazin, the cast iron water jacket became the typical jacket until possibilities of cheaper manufacture made the steel plate jacket attractive.[1]

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[1] I (Franz Cazin) proposed to Dr. Iles, about 1884, to put the tuyeres through the center of the jackets, which reduced the number of patterns to three: viz., end right, end left and side jacket.

The first cast iron water jackets gave much trouble on account of breaking, which was principally caused by not having uniform thickness of shell. This was remedied by having cast iron supports for the cores while drying, insuring thereby an exact shape of the core and consequently a uniform thickness of the shell throughout.

The credit for the tap jackets belongs to the German engineer, Mr. Luer-

With the commencement of the second decade, however, it is not yet time to announce any marked advance in hearth areas, or increased furnace capacity.

When, as at Eureka, the smelting works were simply an adjunct to the mining, and the metal contents of the ores were not actually purchased, it was possible to discard matte and speiss, but later when the smelting companies purchased the ores outright, pay-

mann. I wrote to him for Dr. Iles in about 1883, and with his consent Iles used the tap jackets first in the Grant Smelter furnaces.

The tap jackets were first only jacketed at the sides and top of the tap-hole, later the water space was carried all around.

Troubles by breaking off the bolt-lugs of the jackets were remedied by the Nesmith loose lug, introduced about 1884.

Top jackets were first used in the shape of cast iron corner blocks at the Grant Smelter, about 1885.

Steel top jackets, 5' high, resting on the cast iron jackets were first used at the Globe Smelter by Iles in July, 1887.

This, it seems, went too far and later furnaces f. i. at the Philadelphia Smelter in Pueblo, had again only cast iron corner blocks.

I used top beam jackets supported on wrought iron bars stuck through the columns in 1895, and the same suspended from the deck beams in San Luis Potosi, in 1898.

Iles proposed eliptical tuyere seats in order to blow in different directions. Bailey also had tuyeres blowing at an angle.

The first furnaces of the Philadelphia Smelter in Pueblo were built in 1883 with a flat bottom for outside separation, like copper furnaces. The standard cast iron jackets rested on a brick bottom of 14″ thickness, enclosed in a cast iron frame 5' x 12'. Immediate failure.

The slag spouts were first solid and were bolted to the caisson, which was inconvenient for changing the burned spouts. Then pockets were cast on the caisson plate, the spout was dropped in and held by wedges. Then the pockets were made as separate castings and bolted to the caisson (1888), and the slag spouts water cooled.

Troubles in the cast iron deck beams and the cast iron crucible caissons by shearing the bolts and spreading, I overcame by the heavy diagonal bolts in the corners.

In 1895 the Globe Smelter erected a large furnace 66″ x 160″ section at the tuyeres. The jackets were of cast iron, 20″ wide and 4' 6″ high, bosh 10″, 8 tuyeres on each side. No top jackets. Distance from furnace floor to charge floor 41', ore column about 32'. Floor of basement (for crucible) 7' below furnace floor. Total height 48'. Gases taken off in center of furnace by a flue 7' in diameter. On each side had a round charge hopper, as used with iron furnaces. The furnace was a failure.

(Extract from personal letter Franz Cazin, Denver, Colo.).

ing cash for their metal contents, such wasteful methods were no longer permissible.

In 1852 Frederick Overman's "A Treatise on Metallurgy" had set forth quite clearly the importance of so combining the ores and fluxes as to produce suitable slag composition, of the importance of which he had the fullest realization. It was, however, at the Germania, near Salt Lake City, that the first systematic attempt was made, principally through the efforts of Eilers and Hahn to obtain properly proportioned slag combinations, so that to the word of Overman was added the testimony of that school of metallurgists who, combining theory and practice at the Germania works, founded a distinct science of lead smelting.

With the larger furnaces and increased blast pressures of the more recent practice, together with the preliminary preparation of the ores of the charge, there seems to be a tendency to again be less particular as to the exactness of slag composition, an exactness which was always one more of theory than of actual attainment, and to be content with a suitable "kind" of slag than to demand a rigid adherence to an absolute "type" of slag.

The demonstration of slag formation by solution as made in Europe by Barth, and in this country by Hofman, would also indicate that the adherence to slag types, which, for a time, were so pronounced in American literature and practice, was undoubtedly extreme.

With the smaller furnaces and "carbonate ores" the slag was handled in hemispherical slag pots or buggies, pots mounted on two wheels. The entire contents of the pot, as drawn from the furnace, were thrown over the slag dump.

The necessity of collecting and saving the matte resulted early in the 80's in systematic efforts to separate the matte and speiss from the molten slag.

The conical "overflow-pot" was adopted, which, when partly filled with matte, was exchanged for a fresh overflow-pot. The slag overflowed from the overflow-pot into the slag pot. There came also into use the continuous-flow fore-hearths working somewhat upon the principle of the siphon-tap, such as those of Eurich, Mathewson, Iles, and Huber. This was also the period

of the Devereaux pot, with a tap hole in the side, allowing the retention of a slag shell for resmelting.

This led to the use of the large double bowl slag car operated on tram tracks by mule power. At the Omaha and Grant works at Denver the Devereaux pot idea was developed into a system of large settling pots with the Devereaux tap holes from which slag was tapped.[1]

Previous to this a system of centrifugal separation had been tried at the Omaha and Grant works. Mechanical difficulties and power cost were apparently too great, but this centrifugal separator was interesting in that it apparently demonstrated that the silver in the slag was very largely mechanically disseminated sulphide, since the slag after this mechanical separation was quite free from silver.

At this period, especially in Leadville, the mattes which were becoming so troublesome were practically iron mattes, with little or no copper. Iron matte in such small quantities in slags with increasing zinc, barium and alumina contents was difficult to separate from the slag, so that it was just at this period, the latter half of the second decade, during the transition stage from carbonate to sulphide ores, that probably the greatest silver losses were experienced.

It was a time also of great competition for supplies of carbonate lead ores for fluxing, so that the obtainable percentage of lead upon the furnace charge was forced to the lowest limit, much lower than was compatible with good metallurgical results. This aggravated the loss due to the disseminated iron sulphide through the slag. It was still a period of high iron slags, the specific gravity of which was by no means lessened by the increased percentage of barium coming from newly developed mining districts. This was probably the most trying period, both metallurgically and financially, through which the custom smelting companies passed.

The metallurgists soon recognized the cleansing effect upon the slag of a sufficient fall of actual copper matte as opposed to the evil effect of the smaller quantities of almost pure iron matte, so that with the advent of our third decade, we find all the lead

[1] See "Mineral Industry," XXII.

furnaces with a "matte fall" of from five to ten per cent of first matte ranging from 2% to 8%, or for short periods reaching 12 or 15% of copper.

During the period just referred to when there was such difficulty owing to the slags being rich in silver, the lead assay of both the slags and mattes was reasonably low, but now a change had come. While the actual silver assay of the slag had greatly improved, and was very much lower, the direct production of silver, that is the proportion of the silver contained in the charge which was delivered from the furnace in the lead bullion produced was too small. Where previously the slags had assayed from 0.2 to 0.5% lead, and the mattes 5 to 8% lead, the slags were now ranging about 1.5% lead, and the mattes from 15 to 20% lead for considerable periods of time at practically all of the plants.

Furnaces had now increased in size to about $42''$ x $120''$, and the blast pressure was correspondingly increased. Somewhat later, that is, 1895–1905, a still greater tendency to force the blast pressure to a maximum was developed. To "tighten the charge" and "increase the blast pressure" was the characteristic feature of the time. At two important plants, believing that the demands for pressure had passed the point of efficiency for the rotary pressure blower, piston blowing engines were installed.

But this high blast developed furnace conditions and difficulties which were very serious, such as channeling of the charge with slips, funneling, etc., which allowed not only unreduced ore to pass through the smelting zone, but the zone of fusion was driven up so high in the shaft, regardless of the most approved interior lines, that the furnaces burned out over their jackets and water-cooled liners, and top jackets were resorted to. Lead reduced in this upper zone was reoxidized at the tuyeres, thus further enriched the slag and matte in lead.

The theories of lead smelting and its literature had unduly emphasized the importance of the slag composition, with descriptions of the advantages of this or that "type" of slag. But now the difficulty was not met by any theory of slag composition, and the lead metallurgist turned his attention to other features, such

as furnace lines, angle and position of bosh,[1] etc. As to the lines of the furnace, it does not seem to have been shown that the actual angle of bosh was of so great importance; but it was demonstrated beyond doubt that the position of the bosh was, that is, if the bosh was too high above the tuyeres the smelting zone was driven up so that lead which had been reduced was again oxidized at the tuyeres and entered the slag and matte.

It was apparently a question of improper reduction and the lead metallurgist turned to the iron furnace practice for assistance, and this rather confirmed the high blast pressure.

It is a noticeable fact, if one glances over the American technical literature which naturally reflects the point of view of the two branches of metallurgy, that one finds in the lead smelting such a preponderance of consideration given to the slag composition, or slag types, whereas regarding the iron blast furnace cinder, scarcely a reference is made to its analysis. This, of course, is perfectly natural, since the conditions are so entirely different. The lead furnace produces a great preponderance of slag, while in the iron furnace, the product pig iron is relatively so much greater, and the iron has so high a specific heat and temperature that it renders quite negligible the question of cinder composition. It is, therefore, the consideration of the gases, the relative proportions of carbon monoxide and carbon dioxide which have more largely occupied the attention of the iron furnace metallurgist. The lead metallurgist, at least in America, had given but little attention to the analysis of the gas issuing from the furnace. But naturally at this period of manifestly imperfect reduction in the lead furnace, attempts were made, more or less spasmodically, to systematize the observations of gas analyses, in order to arrive at some theory, the application of which would be of actual assistance in effectively controlling the lead in the slag, and especially in the matte. But there seems to have been nothing of

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[1] At about this time a comparative diagram of the different lead furnaces then in use showed a wide divergence in the amount or angle of bosh. Sixteen degrees was for some reason denoted as correct, and at two large smelting works jackets differing from this were discarded, but with no appreciable change in results.

general benefit developed from the study of the gas, although some very interesting results were obtained.[1]

It became evident that the unsatisfactory furnace conditions were very largely due to the fineness of the material of the furnace charge. Attention was now directed to the beneficial effects, both as to coarseness of charge and other favorable qualities of "pot-roasted" ore or ore sintered by other methods of blast-roasting. The very general use of these forms of predigested furnace feed has resulted more beneficially than possibly any other one step in the advance of lead smelting, an advance quite apart from the lead blast furnace itself.

The preliminary roasting of ores in the hand reverberatory furnaces had about reached its greatest perfection with the beginning of our third decade (1890), and the "fusion box" as an adjunct to the hand roaster had practically run its somewhat costly course. Early in this decade came an era of large mechanically operated roasters of a reverberatory type.

Since a special paper before this Congress by Mr. A. S. Dwight upon Sintering effectually covers the entire question of roasting, the progress of roasting will be omitted from this paper,[2] except further on in reference to the increased capacity due to blast-roasted and sintered furnace-charges.

Likewise, since there is also to be presented to this Congress a paper by Mr. Anton Eilers upon the "bag house," the question of flue dust collection and fume condensation will be omitted.

The commencement of our fourth decade (1900) finds substantial progress toward larger furnaces and increased furnace capacity, with corresponding introduction of mechanical devices for the handling of material to and from the furnaces.

The first movement toward an increased size of furnace in the shape of a wider rather than a longer furnace was made at the

[1] Mr. A. S. Dwight by a series of interesting figures based upon gas analyses showed that not much over one-half of the air which should have passed through the furnace, as figured from blower revolutions, was accounted for, and that the blower efficiency was apparently less than had been assumed. (Personal conversation.)

[2] See also Efficiency of Ore Roasting, by A. S. Dwight, Eng. & Mg. Journal, December 30, '12.

The first successful furnace at Eureka, Nevada.—Operated by Mr. Robbins.
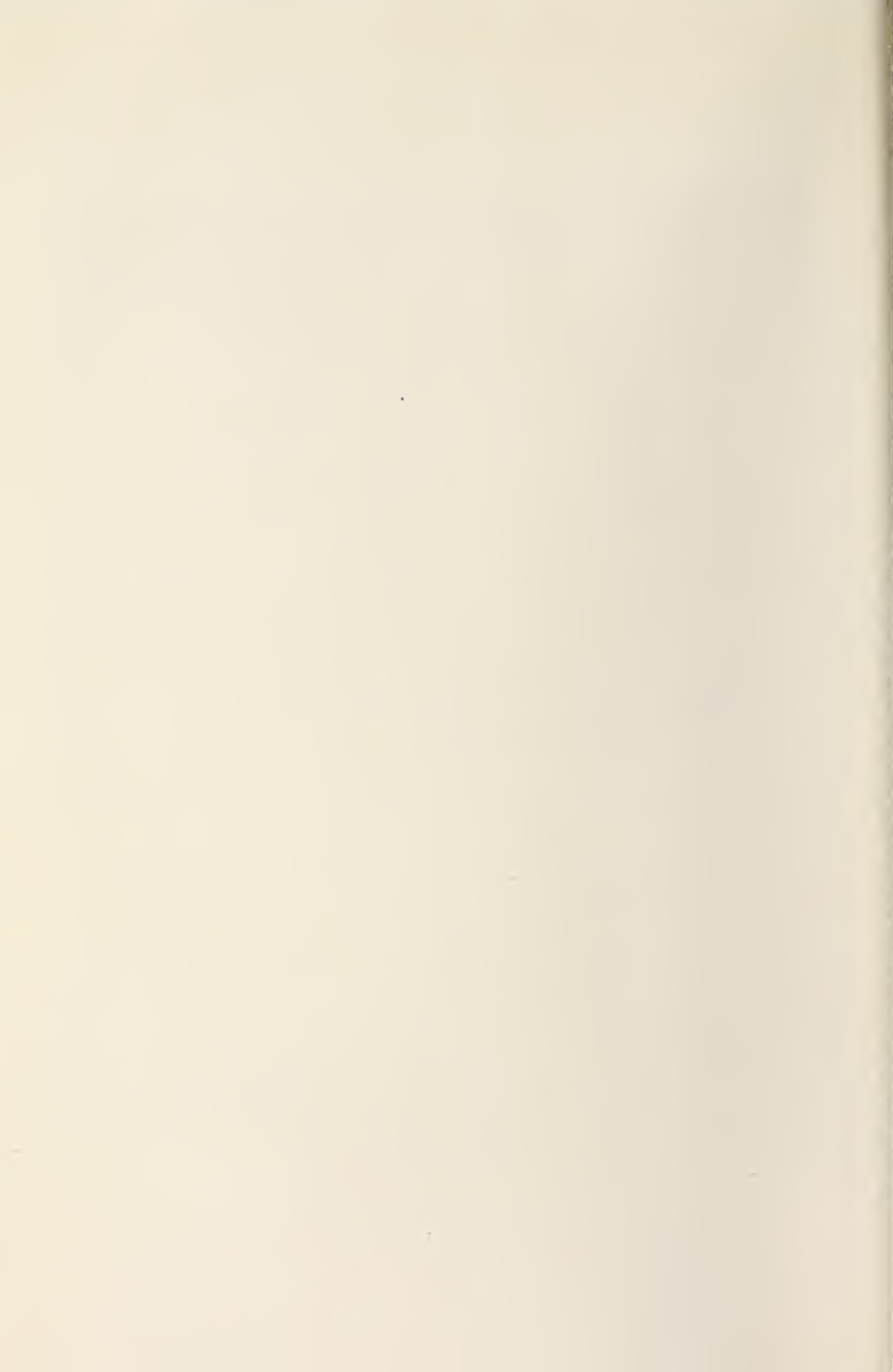
Typical matte handling at modern lead smelting plant.
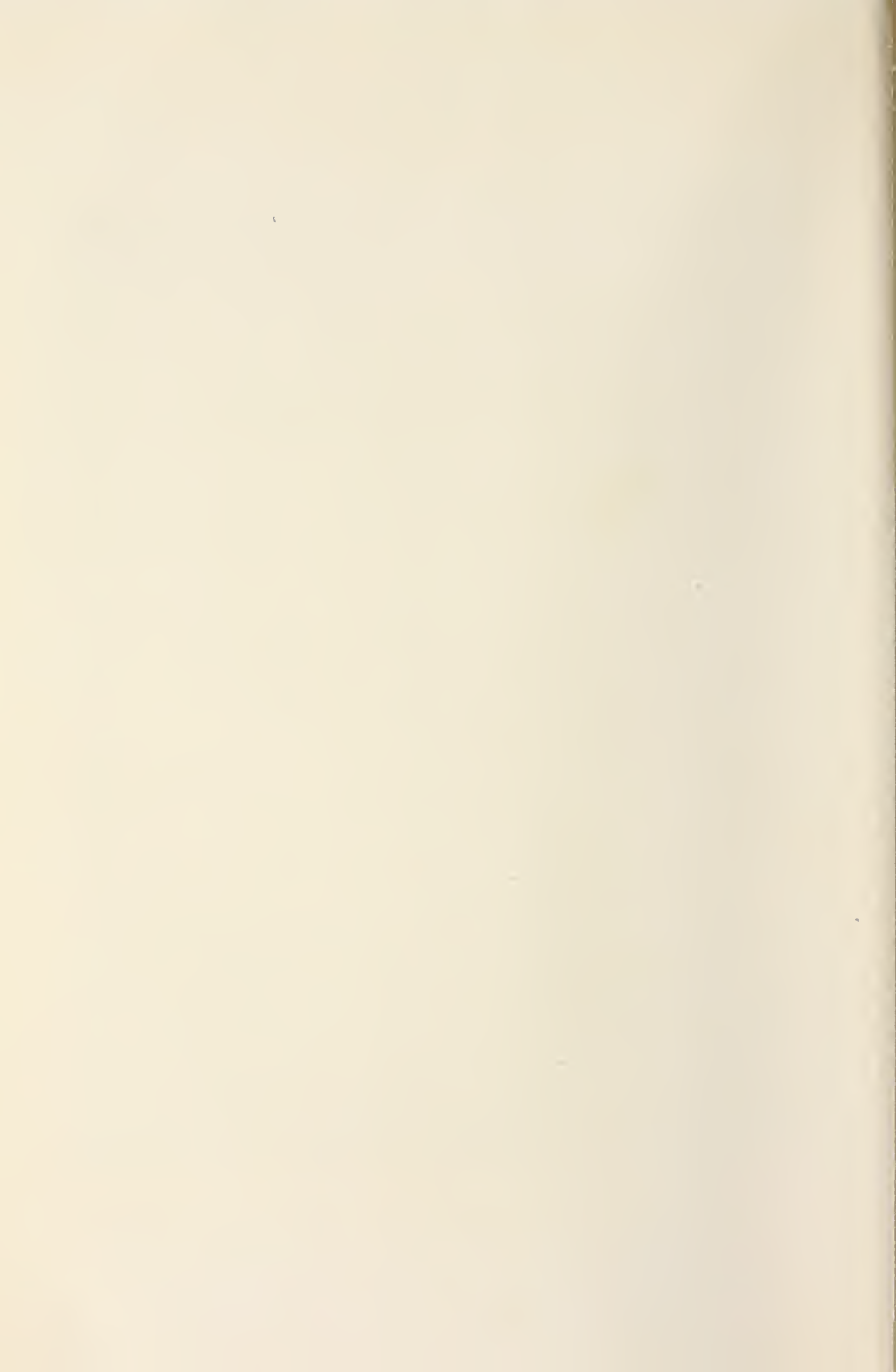
works of the Pueblo S. & R. Company in 1885.   But as we shall
see, the general practice for the increase of hearth area has been
to lengthen the furnace, with only a slight addition beyond 36″
to the width.

It has been characteristic of the lead blast furnace practice
that the units have been comparatively small, seven to twelve
furnaces to handle the ore capacity of a plant.   The present idea
is, however, that three, or at most, four units, should be of
such capacity that they should handle the total desired capacity
of the works.   Every adjunct of the furnace has had to be in-
creased correspondingly.

The little hemispherical slag pot mounted upon two wheels
in which one man laboriously wheeled 250 to 280 pounds of slag
has been replaced by slag cars conveying 3,000 to 8,000 pounds,
operated quite generally by a steam locomotive or an electric
trolley motor.

The first locomotives for handling lead blast furnace slag were
used, by the author, in 1888, at the Arkansas Valley Smelter in
Leadville, and were possibly, then, apparently ahead of their
time, but the system proved an economy.[1]

Coincidently with the problem of handling the larger tonnages
of slag came the necessity of more perfect separation of the
matte from the slag.   Large forehearths in the form of actual
reverberatory furnaces, with capacities of from thirty to forty
tons of the Rhodes-Van Cleve slag separator type were intro-
duced at several plants.   At El Paso, with this type of separator,
the writer designed an arrangement for granulation of the matte
direct from the separator.   Also at these works were used in
connection with the separator a continuous-flow forehearth of the
types usually designated as Orford.

The reverberatory slag separator, however, had the disad-
vantage, that frequently a slag which was perfectly satisfactory
in the blast furnaces proved troublesome in the separator.   The

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[1] The decision to adopt the locomotive for handling the slag at the Arkansas
Valley works in '88, was no doubt somewhat influenced by the fact that there
were available several Denver & Rio Grande Railroad narrow gauge engines,
which were too light for the traffic at that time.   In purchasing the second
engine, however, a new Baldwin locomotive was procured.

separator also decidedly limited the percentage of zinc, which could be carried on the charge.

Matte settlers or forehearths constructed with sectional sides and ends which can be readily taken apart for removing crusts, and mounted on trucks to fit the matte car tracks are now generally adopted.   (See Plate No. 00.)

Mechanical feeding of the lead blast furnace was one more step in advance.[1]   For round furnaces the bell and hopper had been used more or less successfully, although tending to give too hot a furnace top.

At the Pueblo S. & R. Co. plant, in 1895, the first success was made of mechanical feeding.

Later, by Dwight's modification of Hixon's East Helena automatic feeding device, was obtained the baffle plate device for throwing coarser material to the middle and finer toward the side-wall of the furnace, which has since become the essential feature of automatic feeding.   Dwight in his work upon automatic feeding was the first to sufficently emphasize the importance of the mechanical element of the charge in blast furnace work, and to point out that the operations would be immensely benefited by proper preparation of the material.

We have thus followed lead smelting from its first crude "rule-of-thumb" beginnings in Nevada, through what may be termed a scientific period into a highly developed art.   The technical questions which were of particular interest during any one period of development were different from those of another period, changing from time to time with the changing requirements or points of view.   It has not seemed reasonable, therefore, to enter too minutely into the technical detail of the practice of each succeeding decade.

But even at this time if we attempt to state what is the present efficiency, we find that we are again in a transition stage.   We find that the capacity of no one set of furnaces can be comprehensively compared with the capacity of any other set without considering also certain modifying conditions.

[1] See Mechanical Feeding, Silver Lead Furnaces, A. S. Dwight, Vol. XXXII, Transactions, A. I. M. E.

It has always been known that the capacity of a furnace would be greatly retarded by an increase in the proportion of fines in the charge, and yet, as noted above, there was one period, not much over a decade past, when there was quite a prevalent belief that the evils due to imperfect reduction would be overcome by packing the charge to prevent too rapid passing of the reducing gases and by correspondingly increasing the blast pressure. We have seen also the troubles of this attempt to drive a high-pressure blast through a fine charge.   At just about this time the Huntington & Heberlein "pot roasting" process made its appearance, the furnace charges were improved not only by the character of the pot roasted material itself, but the removal of the fine material which would otherwise have gone directly on the blast furnace charge.

At one plant with furnaces 48″ x 160″, with average charge, the daily capacities range from 140 to 160 tons.   The advent of the H. & H. Process with a fair proportion of the pot roasted product, raised the tonnage to about 200 tons.   The conditions of the charge were still further improved by the installation of the later form of blast-roasting and sintering device of Dwight & Lloyd, when the capacity of these furnaces were still further increased to 250 or more tons.   These same furnaces, when on matte concentration, have a capacity of 350 to 375 tons.

Such a progressive record as this most plainly indicates how important has been the effect of these processes for a preparatory treatment of the ore, and this is why, as I stated in the beginning, the recent progress cannot be considered most largely due to improvements of the blast furnace itself.

And so we find the modern lead furnace, generally speaking, 45 to 48″ wide by 160 to 180″ long, with capacities ranging about 200 tons.   Apparently between 25 and 30% of the increase in capacity has so far been obtained by preliminary preparation of the charge.

At one plant the entire furnace charge including silicous ore and some oxidized ore, that is the entire charge except the limestone, (and of course coke) is "blast roasted" and sintered. The greatly increased furnace capacity fully warrants this preliminary treatment even of ores, which not being sulphide are

not so treated for the purpose of desulphurization, but simply because of the advantage of such predigested furnace feed. On these furnaces the capacity is practically doubled.

Not only has there been the increase in tonnage which has correspondingly reduced labor and general cost but when material is sintered, by the more recent methods, requiring only a few minutes, instead of as many hours as in the pot roasting, it has been found that the percentage of fuel can also be appreciably reduced, for illustration, from 11% to 9% fixed carbon.   This is possibly due to the fact that the material subjected to this shorter time of blast roast has the iron contents as a lower form of oxide.

The tonnage of the 48″ x 160″ furnaces above referred to, while large in the aggregate, is equalled or surpassed in tons per square foot of area at the tuyeres by other plants.   For ore charges $5\frac{1}{2}$ tons per square foot at the tuyeres is about the best operating record at the present time.

The largest lead blast furnace in operation at present is 42″ x 220″, which is small when compared with Mathewson's well-known Anaconda copper furnaces, one of which is five times as long, and about one foot wider.

A furnace 45″ x 215″ was considered for Trail, B.C., but with the character of the charge, unique for America as carring from 50 to 55% lead, and with about eighty per cent of the charge pot-roasted or sintered material, their small furnaces had ample capacity for the ore available.

It is observable that increased hearth areas have been attained, as was done by Mathewson, by increasing the length of the furnace, since American furnaces are universally of rectangular section.

It would not be possible to obtain for publication actual figures as to the present cost of smelting, nor as to metal savings, or as it is expressed, metal losses.

It is very evident that so great an increase in furnace capacity has correspondingly reduced labor and general expense cost per ton.   Besides this, we have also noted a decrease in fuel when smelting blast-roasted material.

In the second decade, with furnaces 30″ x 100″ in Denver, Iles gives $4.65 per ton, and toward the close of the third decade

$2.26 in 42″ x 140″ furnaces. Both of these figures are apparently lower than was the average cost in furnaces of the sizes indicated at the periods given. It is safe to assume, however, that present costs are, generally speaking, somewhat within the latter of the above figures.

If one had before him a tabulated statement of the average metal losses, gold, silver, lead and copper, for each of the decades which we have been considering, made up from the metallurgical statements of those periods, one would find the figures misleading unless due consideration is given to certain features which govern the figures of loss or apparent gain. The actual improvement in metal recoveries has been greater than would be indicated by a simple comparison of the percentage figures of such a tabulated statement. For example, when the Leadville ores were smelted practically by themselves, the gold content of the ore smelted was so low that the increment due to "traces" of gold (that is ores assaying under $\frac{1}{10}$ of an ounce which was then not paid for nor charged on the metallurgical statement) bore such a proportion to the total gold purchased, that the metallurgical statements of the period showed an apparent gold gain of from 50 to 150%. But the actual money value represented in these figures was comparatively unimportant, since the gold assay of all the ore smelted was low. In the Denver and Pueblo plants this gold "gain" was not so marked, especially after ores from Cripple Creek came into the market. It was found also, that, in addition to the increment due to traces of gold in the ore, the smelting works had considerable gains in gold which were due to the fact that in the sampling of bullion assays too high in gold were obtained, so that it was necessary to remelt the bullion in large kettles, holding an entire car lot, and take the sample from the molten bullion in order to obtain correct samples.

Then again during the period of great competition for ores, when only 7 or 8% of lead was carried on the charge the apparent lead loss was correspondingly high. It is evident that under these conditions even with the same actual lead loss by slagging and volatilization, the apparent percentage loss was nearly double of what it would have been with 12% of lead on the charge. Further "wet lead" or chemical analysis has practically replaced

the fire assay, so that in these days apparent losses, based upon wet lead will be greater than those based upon fire assay.

So in the case of copper; at one time there was invariably an apparent gain, due to a large proportion of the copper coming from ores assaying below the limit at which copper was paid for. In these days, however, all metal content is figured whether paid for or not.

The comparisons in silver loss were undoubtedly less influenced by such conditions so that a comparison of silver losses more nearly indicates the actual improvement which has been attained. This improvement is due largely to the increased matte fall of a first matte, carrying a sufficient percentage of copper to very materially assist in the cleansing of the slag. Then, too, there is very much less loss in the most recent methods of blast roasting than in the older reverberatory roasting, especially when the furnace was coupled with the fusion box which at one period was so generally adopted

We still have, however, differences due to accuracy of assay arising from character of the ore. At one of the Colorado plants which smelted for a time rather high-grade silver ores, principally zincy sulphides instead of showing an unfavorable metallurgical comparison during this period, actually showed either an apparent silver gain, or practically no loss, while other plants operating upon clean ozidixed carbonate ores showed the maximum loss of the period. This gives the paradox that refractory ores were the more readily smelted. The difference was largely one of correctness of assay.

Metallurgical statements of metal losses are more or less relative.

It is indeed a far cry from the little sandstone and adobe furnaces of Nevada, with their capacities of from eight to ten tons per twenty-four hours, and fuel consumptions of forty or fifty per cent, and no telling what losses in flue dust and fume and uncollected products in the slag, up to the present high perfection of the art.

And yet one cannot but be impressed by the note of conservatism which has marked the technical progress of lead, silver smelting, and to feel that henceforth such improvements as may yet be possible will be much more promptly achieved, until possibly the ultimate art will in no wise resemble present practice.

(Abstract from paper)

"RECENT TENDENCIES AND DEVELOPMENTS IN OPEN–HEARTH PRACTICE"

By R. B. Carnahan, Jr., D. Sc.
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In this paper the author reports on the progress made in open-hearth practice in the United States, especially in the last three years.

He states that the improvements consist largely of metallurgical and chemical refinements, and that but little attention has been devoted to the mechanical side of the question.

He summarizes the developments under three heads: (a) improvements in open-hearth rails, (b) developments in special alloy steels, and (c) the making of pure iron.

The paper further states that the conservation movement that has been sweeping over the country has modified the viewpoint of the American manufacturer, and that in a comparatively short time the United States will be as progressive as Europe in research and development work.

MANGANESE BRONZE TESTS

By Wm. M. Corse and V. Skillman

Of all the modern non-ferrous alloys, Manganese bronze is probably the strongest and most moderately priced of those readily obtained. It is therefore industrially important and of great value in a large number of cases where strength and price are the important factors. Its use is extending rapidly and for that reason its characteristics and properties should be generally well understood. As a result of its being used where strength is necessary, the desired physical properties are often given and specifications have to be met by the manufacturers. It is these specifications and physical properties with which this paper has to deal.

Much has been written about the correct formula and method of making this alloy, but very little has been said about the way to test its physical properties, or what such tests may represent. As the data obtained from the physical tests are the usual criterion by which we judge the results of our efforts to make this bronze, it seems that a clear understanding of them is of as much importance as a knowledge of the composition of the alloy itself. That there is not this understanding is evident in part by the fact that there is such a discrepancy in the specifications which the foundryman is asked to meet.

It seems to be taken for granted by many engineers that the physical properties of the material in every part of any given casting are the same, and that any kind of a test coupon cast in the same mold as the casting from the same metal under identical conditions would have like physical properties. It can readily be shown that such is not the case. This does not apply to manganese bronze alone, but to other alloys as well. A test coupon can do little more than represent the properties of the metal entering the mold when such metal is allowed to solidify in the shape and according to the conditions under which the coupon is made. To illustrate, an example may be cited.

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Suppose we take two identical cylindrical patterns from 1 to 2 inches in diameter and about 5 inches long. We will make up a mold of these in the horizontal position, gating each at one end to the same spruc and placing a larger riser at the other end to take care of the shrinkage. In addition we will place a V-shaped riser on top of one of the patterns, extending its full length. If a good grade of Manganese bronze is used and test bars turned from the resulting castings, we will find, on breaking them, that the casting with the riser running its full length has from 60 to 70 per cent more strength and about 400 per cent greater elongation than the other casting. In giving figures, we will refrain, as far as possible, from quoting the exact results of any one test, but will give the rounded average of a number of similar determinations, as we wish to deal with general principles.

In the example given above, either one of the castings may be considered as a test coupon cast attached to the other casting whose physical properties it is to represent. It very evidently fails in this as must every test bar coupon which is not very similar in shape and size to the casting in whose place it is used. If a test coupon then is to represent only the physical properties of the metal when cast under certain conditions and is not to represent the strength and other properties of every portion of the casting, which it evidently cannot do, it may be concluded that a very important question is: "What pattern of test bar coupon will best represent the character of the metal used?" That this has been realized, is shown by the fact that the American Society for Testing Materials for one, has undertaken to settle this question, and decide upon a standard test coupon for Manganese bronze.

It makes little difference if the standard coupon gives the highest possible values or not, as long as it can be depended upon to give the same general results with the same metal, handled in the same way. For this reason, the coupon cast in the small metal ingot mold has many advantages. Many metallurgists believe that a chilled test bar gives erroneous results. Unless the subject is considered carefully, this is an easy conclusion to reach, especially if one is laboring under the

delusion that every integral part of a casting should have the same physical properties as the test bar representing it. As no pattern of test bar can come anywhere near fulfilling this condition, if the casting is at all complicated, most any kind of sand cast coupon is likely to be as far away from the true values as the chilled bar, with the added disadvantage of having more variables to control, if comparable results are desired. With sand cast bars we have many patterns to choose from, and after a certain one is decided upon, we still have many variables to contend with, in the sand used, in the method of gating, and in the possibility of improper feeding.

The difficulty of getting a representative test coupon is always accentuated in alloys with relatively high shrinkage, such as manganese bronze, so that more importance must be attached to the design of the test coupon here than with the low shrinkage alloys.

The difference in the properties of a chill and a properly designed sand cast coupon are not as great as a number of metallurgists are led to believe, as a result of their use of incorrect casting conditions. Much good Manganese bronze is condemned because the results, on testing, do not come up to the maker's guarantee, the reason being that an improper method of casting the test coupon has been used, which would lead to low results with every Manganese bronze of first quality.

While the ultimate tensile strength of chilled test coupons is from 5 to 10% higher than sand cast, nevertheless the points in favor of the use of the former are well taken and if the results are used with understanding of the conditions, no hardship should be imposed. However, if specifications based upon results which might be obtained with chilled coupons are applied to the usual sand castings, conditions are imposed which are practically impossible to fulfill.

The designing engineer should remember that the strength of his casting depends to a great extent upon himself as well as upon the metal which he specifies. No complicated casting has the same strength per square inch of metal throughout. For this reason, no test bar, either chill or sand cast, can have the same properties as every part of the casting. The strength of

a good test coupon bears a certain relation to the strength of the various parts of the casting, depending upon local conditions, but it does not give the absolute strength of every square inch of metal in the casting.

Test coupons, in general, may be roughly divided into three classes. First: sand cast coupons of about the size of the test bar desired; second: some modification of the first class having a riser to take care of shrinkage as part of the pattern; third: chill cast coupons.

The first class are not suitable for Manganese bronze on account of the practical impossibility of obtaining proper feeding.

Of the second class, the authors have found the patterns suggested by various members of the American Society for Testing Materials, very satisfactory. This consists of a block 3″ wide, 4″ high and $4\frac{1}{2}$″ long with 1″ square coupons extending lengthwise along each side at the bottom, and along the center of the bottom itself, making three test coupons, in all. This pattern automatically takes care of the necessary feeding of the coupons. Further, it is not greatly influenced by differences in pouring temperature, but it is susceptible to changes of formula. Coupons of the first class decrease rapidly in strength and elongation with decrease in pouring temperature.

Chill cast coupons will give from 5 to 10 per cent greater tensile strength and very slightly more elongation than the sand cast coupon recommended above.

A few words may be said in regard to the physical tests which are usually made upon the standard threaded test bar which has a test section two inches in length and one-half inch in diameter. Of the four determinations most often made, *i.e.*, tensile strength, elastic limit, elongation, and reduction of area, the tensile strength and elongation are the only two of much value.

If the testing machine is not run at too great a speed and sufficient time is allowed before the break takes place, the figure for tensile strength is quite reliable. The elongation can also be depended upon if the fracture is such as to allow proper fitting together of the ends of the test bar.

As to the so-called elastic limit, much could be written. At

least two papers have recently appeared, calling attention to the absolute fallacy of the figures usually given for the elastic limit as ordinarily obtained in commercial practice by the drop of beam of common dividers. Both Capp (American Society of Mech, Engrg., June 1910) and Sperry (Brass World), Vol. VII, No. 8) are quite well agreed that Manganese bronze does not have anything like the high elastic limit usually credited to it. The value reported as such, is not, in fact the elastic limit at all, but that more or less vague value, the yield point. The elastic limit of a material is the stressing value beyond which Hook's law ceases to hold. That is, it is the limit of proportionality of stress to strain. The yield point is a higher value and is where the crystals begin to slip and the elongation increases rapidly. The value assigned to this yield point depends solely upon the "personal equation" of the operator and the facilities at hand. The stress-strain diagram for manganese bronze presents a smooth curve throughout without any sharp break at the yield point as is found with the average steel. For this reason even the yield point cannot be determined by the drop of beam, to say nothing of the elastic limit. With increasingly accurate means of observing a definite stretch and more skill on the part of the operator, lower results are reported for the yield point. At best, these are over 25 per cent above the true elastic limit and may be as much as 50 per cent if ordinary commercial methods are employed. The results are therefore of little value, and in view of their uncertainty, it would be well if the value of the so-called elastic limit was excluded from specifications.

The reduction of area is another determination in which the "personal element" enters largely. The cross section of the break is practically never uniform in diameter, and the figure given for its measurement depends greatly upon the operator. It seems to be customary to take the smallest reading, which is evidently not correct, and it would be well if this determination was likewise not required.

By the exclusion of the elastic limit and reduction of area requirements from specifications, little if anything would be lost as they depend very greatly upon the tensile strength and elongation which may be determined quite accurately.

If we can convey the importance of the use of the proper test coupon, coupled with intelligent testing, to the designer and use of Manganese bronze and similar alloys, the purpose of this paper will have been accomplished. The facts stated serve to emphasize the growing interest in all non-ferrous alloy subjects, and as such, are an addition to the rapidly growing literature of these materials which are beginning to receive their proper share of scientific investigation and thought.

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THE DWIGHT & LLOYD PROCESS OF ROASTING AND SINTERING

By Arthur S. Dwight

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In a paper* presented to the Seventh International Congress of Applied Chemistry in London, May 1909, Professor H. O. Hofman, of Boston, Mass., described a then new development in blast-roasting processes and some of the interesting results which it had already attained. It is the purpose of the present paper to briefly describe some of the progress that has been made in this process since that paper was written, the measure of success that has been achieved in meeting some of the serious problems which have long been connected with the metallurgical treatment of fine ores and concentrates, and finally to submit certain facts which may be pertinent to such a discussion. No attempt will here be made at this time to describe the process in detail or to discuss the fundamental principles. Reference can be had to articles previously published.†

For the requirements of this paper it is sufficient to say that the process is essentially a mechanical improvement on the so-called blast-roasting or "pot-roasting" processes for agglomerating fine ores by the heat generated by internal com-

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* "Some Developments in Blast Roasting" by H. O. Hofman. See also, "Recent Progress in Blast Roasting" by H. O. Hofman, Trans. Am. Inst. Mng. Engrs. Vol. XLI (1910) p. 739.

† See "Dwight & Lloyd Sintering Process" by A. S. Dwight, Engineering & Mining Journal, March 28, 1908.

"The Sintering of Fine Iron Bearing Materials" by James Gayley, Am. Inst. Mng. Engrs. Bulletin No. 56, August 1911, p. 631.

"The Sintering of Fine Iron Bearing Materials" by James Gayley, the "Iron Age," Jan. 4, 1912.

"The Sintering of Fine Iron Bearing Materials by the Dwight & Lloyd Process" by B. G. Klugh, with discussion, Am. Inst. Mng. Engrs. Bulletin No. 65, May 1912.

Also U. S. Patents 882, 517; 882, 518; 916,391; 916,392; 916,393; 916,394; 916,395; 916,396; 916,397.

bustion, such processes being typified by the well-known Huntington-Heberlein, Savelsberg, etc. processes. By means of this improvement the following broad results are obtained.

1. Instead of the hugh cakes of pot-product, dense and slaglike, requiring much labor to handle and break up to the proper size for smelting, and accompanied by a considerable proportion of fines, a thin slab of crisp, cellular "biscuit" is produced which, while easy to break into pieces of the right size for smelting, is yet strong enough to withstand ordinary handling and to support the weight of the charge column in the blast furnace. When the operating conditions are intelligently adjusted, the amount of fines made in the Dwight & Lloyd product should be practically *nil*. Much of the success of the process has been due to the extraordinarily favorable properties of this peculiar and distinctly novel metallurgical product.* The apparatus used for manufacturing this thin cake may be either *intermittent* or *continuous* in action, the latter being preferable for many important reasons which will be hereafter discussed.

2. By feeding the fine materials to be sintered in a thin layer of uniform thickness on the perforated bottom of a travelling conveyor, and after automatically igniting the charge on one surface, passing it through a section of the apparatus where air currents are caused to permeate the thin layer, the original ignition is propagated through the mass in such a way that it progresses from the surface of ignition to the opposite surface, and thereby effects a complete sintering of the mass from one surface to the other at the expense of the sulphur, carbon or other combustible element present which supplies the necessary fuel. The conveyor is meanwhile moved at such a speed that the agglomerated product is ready to automatically discharge as soon as the operation is complete.

At the time Prof. Hofman described the process in 1909 it was still in an experimental stage, though already operating on a practical every-day scale, and yielding results that could not be regarded otherwise than as most encouraging and of far-reaching promise. There remained many problems to solve, both

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*U. S. Patent 1,020,345, issued March 12, 1912 to Dwight & Lloyd for "Metalliferous Ore Product."

chemical and mechanical. It can now be said that a point has been reached where the apparatus and practice have been standardized, the chief points of weakness corrected and the effect of variations in chemical compositions quite fully understood. Results can now be forecast as confidently as with an ordinary blast-furnace problem.

It was natural to expect that a few of the first small experimental plants that were installed should yield inconclusive results or fail to exactly meet the severe though rather vague demands made upon them. This, indeed, happened with several of those plants that had been installed at the time Professor Hofman wrote. Among these, however, the plant at Salida, Colorado, which afforded the first example of the "straight line" machine, has been in continuous daily operation ever since, and its capacity has been increased by adding to the original small unit described by Professor Hofman (30″ x 150″ effective sintering area, rated at 50 tons per day) 3 large size units (42″ x 264″ effective area, 100 tons per day rated capacity), the entire battery now treating in the aggregate an average of 250 tons per day.

Every plant which has been installed in the United States since the spring of 1909 has been in practically continuous operation up to the present time, and the results have been so favorable that in all but one case additional units have been installed after the first unit had been tested for a reasonable period of time. In the United States, Canada and Mexico, there are at present some 17 plants in steady operation, the largest of which consists of 6–100 ton units. These plants are variously treating lead, copper and iron ores, the majority being in lead works.

The promptness of the lead smelters to adopt the process is explainable by the fact that these already had had experience with the advantages of the pot-roasting processes and were, therefore, quicker to appreciate the advantages offered by the new improvement. In 7 plants in America the D & L machines have been operating side by side with the old-style blast-roasting-pots for periods ranging up to three years, thus affording an excellent basis for comparison of costs, quality of product and general efficiency. On all these counts the machines have excelled, and

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while the pots have generally been kept in commission, at least in part, it is significant that all increase of capacity in these plants has, without exception, been supplied by D & L machines. In one smelting works after a long competitive trial, a plant of nearly 30 pots was put out of commission and scrapped, its place being taken by 4 large D & L units with regular daily capacity of 425 tons of charge (averaging 45% Pb), which plant has since been increased to 6 units. Under exactly the same local conditions, the average output per man-day was, for the pots, 2.3 tons, and for the D & L machines 16 tons. As a result of this change, the force of operatives was cut from 185 men on the pots to 27 men on the four machines, doing the same work.

In copper metallurgy the advantages of sintering as a preliminary to smelting have not been as self-evident s in a lead smeltings, but they are none the less a fact. The process is as perfectly applicable to copper ores as to lead ores with no essential change in plant or procedure. In fact, the process was originally worked out on copper ores. Four plants are now in regular operation on copper ores, two of which are at present increasing their capacity by adding more units. Most of the interest of the copper metallurgists has lately been directed toward the improvement of the large reverberatory. It is interesting to note in passing that on the basis of present smelting costs in certain large copper smelteries having both blast- and reverberatory-furnaces, combined with average sintering costs elsewhere prevailing, *sintering plus blast-furnace smelting* already rivals in cheapness *mechanical roasting plus reverberatory smelting*, with large potential advantages remaining in favor of the blast-furnace when treating this peculiarly favorable material.

The agglomerating of fine iron ores, pyrites cinder, flue dust, etc., has been the most recent of the practical applications of this process, the only difference in procedure being that the internal combustion necessary for the sintering is derived from intermixed carbonaceous ingredients instead of sulphur.

As concerns type of apparatus now being used, the "straight line" type of machine has been found most effective and convenient and has been standardized in America and Australia to the exclusion of the other available types. In Europe, the "round

table" type seems to be preferred and, in fact, is being used exclusively. No example of the "drum type" originally described* is now in active operation, though it actually possesses many points of advantage all its own. The fact is, a very wide choice of apparatus is permissible, provided certain essentials are kept in view, and that particular type which has been gradually improved and standardized has naturally been given the preference.

The Dwight & Lloyd *intermittent down-draft pans* have been given practical trial in a number of cases in America, Australia and elsewhere, though usually in an experimental way as a preliminary to the adoption of the continuous process. As elsewhere pointed out** the efficiency per square foot of grate surface of the intermittent apparatus is much greater than obtains with the old-style up-draft pots, but for the very same reasons is distinctly inferior to the continuous apparatus. In addition to overwhelming theoretical reasons, this has been practically demonstrated beyond question in every one of the several plants where the intermittent apparatus has been operated for long periods by the side of the continuous sintering machines, both styles treating the same class of material under the same local conditions of labor, etc. The continuous process has invariably proved its superiority in the essentials, viz., lower operating cost per ton and better quality of product with less percentage of fines, while the first cost of the continuous installation is no higher, for the same tonnage capacity.

One of the most interesting of these comparative tests took place at the Port Pirie Smelting Works of the Broken Hill Proprietary Company in Australia. A plant of Dwight & Lloyd intermittent down-draft pans of the tilting type was installed there in 1908 to operate in conjunction with the Carmichael-Bradford process of blast roasting, using calcium sulphate as diluant. The combination yielded much more satisfactory results than had previously been obtained with the Carmichael-Bradford process and the old-style up-draft pots, but these results were entirely eclipsed after a competitive trial had been made with

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* A. S. Dwight, Eng. & Mng. Journal, March 28, 1908, Ob. Cit.

** A. S. Dwight, " Efficiency in Ore Roasting" School of Mines Quarterly Vol. XXXIII, No. 1, Nov. 1911. See also Eng. & Mng. Journal Dec. 30, 1911.

a small Dwight & Lloyd sintering machine, even though the latter was not so arranged as to make its best showing and was in fact handling a smaller tonnage per day.   The final outcome of a long test was that a plant of 6 large size "straight line" Dwight & Lloyd machines was ordered, for the double purpose of replacing the battery of 6 tilting pans for sintering and also to provide for pre-roasting the galena concentrates, which work had previously been done by mechanically rabbled furnaces.   This application of the Dwight & Lloyd apparatus is particularly interesting, and suggests that this method of roasting may be advantageously adopted in other cases and the old-style mechanically rabbled furnaces replaced.   The present practice at Port Pirie is to pass the material over the sintering machine at comparatively high speed without any effort to produce a good sinter, but merely to eliminate sulphur.   The product is then passed through a disintegrating machine and retreated on a second machine, with the result that a sinter of excellent quality and low residual sulphur is obtained.

The reasons for the superiority of the continuous system must be apparent to any one familiar with the practical details of sintering.   Success depends to a pre-eminent degree on the maintenance of absolutely uniform conditions of adjustment as to mixture, permeability, thickness of layer, size of material, compression, moisture, air blast and heat-treatment.   With the intermittent plan, all these conditions have to be re-established for each small batch, thereby demanding a maximum of attendance, and even then the desirable conditions cannot be obtained with anything like the degree of perfection that is easily and automatically secured by the continuous mechanism.   In the effort to develop the continuous system now used, the desideratum was constantly kept in mind of resolving the essential conditions into their primary factors, and this has been so successfully accomplished that with the present system each factor is represented by a simple mechanical control, which can be adjusted independently of all the others.   These adjustments can instantly modify chemical composition, thickness of layer, degree of compression, percentage of moisture, pressure and volume of blast, degree of ignition, time of sintering, or its equivalent, speed of

travel.   Thus it is possible to keep every element of the system up to its maximum capacity, and the changing of one adjustment need not affect any of the others.

Furthermore, in the intermittent process, it is difficult to judge when the layer is completely sintered, because it finishes on the under side next to the grate and, therefore, the finish is out of sight of the attendant.   If discharged prematurely, an undue proportion of raw fines will result.   The sintering time of the preceding charge cannot be taken as an accurate index, because the successive charges necessarily vary widely in degree of compression, and consequently in time of sintering.   It is to be expected therefore that the attendant will allow ample time for the entire charge to finish which, of course, means some waste time, thus diminishing the possible output of the apparatus.   On the continuous machine, a cross section of the finished cake is always visible at the point of discharge and the speed of travel of the machine can be instantly varied to meet the best requirements. The natural result of these facts is to make the efficiency of the continuous apparatus $1\frac{1}{2}$ to 3 times as great as that of the continuous apparatus, which efficiency factor duly affects all cost items both of investment and operating.

On account of its apparent simplicity, the intermittent system has been strongly advocated by a recent writer* who, however, frankly admits that he has had no practical experience with sintering operations.   The idea of simplicity is always attractive and if combined with a high degree of efficiency the experienced metallurgist recognizes the combination as an unconquerable one; but in the present case, the simplicity of the intermittent apparatus is more apparent than real, and, indeed involves disadvantages in the way of complexities of regulation and operation that far out-weigh the supposed advantages, as has been practically demonstrated in the numerous instances, above mentioned.

In the complete operating system as now regularly installed, the Dwight & Lloyd machine is merely a link, though indeed a very important link, in a chain of continuous operations constituting a process complete in itself   The various ingredients of

* " The Important Factors in Blast Roasting" H. B. Pulsifer, Met. & Chem. Eng'g., March and April, 1912.

the charge are automatically measured out from their respective bins by continuously operating feeders whose output can be very accurately adjusted, and delivered upon a main conveyor belt which collects and transports the stream of stratified ingredients to the sintering plant. Before reaching the machines the charge passes through a continuous mixer which affects an intimate commingling of the ingredients and adds exactly the right amount of moisture required for the sintering operation. From the hopper of the sintering machine a thin layer of charge, perfectly uniform in character and of even thickness, is fed to the grates, ignited and sintered as it passes over the suction box where it is exposed to the downward moving air currents, finally arriving at the point of discharge, coincident with the completion of the sintering operation. It is discharged at intervals of a minute or two in the form of slabs of crisp, porous sinter with practically no fines. A suitable conveyor or cars of any description may be used to receive it and transport it to the smelting furnaces. All of these successive operations are subject the same speed control as the sintering machine, so that when the automatic feeding apparatus has once been adjusted to give the right amount of material to supply the machines, the output can be varied without the necessity of changing the quantity-adjustments. Labor of handling is entirely eliminated, attendance being needed merely to watch and maintain adjustments. Two men per shift can easily attend to three machines, but the same force will be required for a single machine. Hence, the labor cost per ton diminishes rapidly as the daily tonnage increases.

With a well-designed plant of one unit treating 90 to 100 tons per day, a sintering cost of 50 cents per dry ton (2000 lbs.) of charge treated can be reasonably expected under average conditions of labor, power and fuel for ignition. This includes royalty and repairs. This figure has been equalled and excelled in several plants as an average for considerable periods of time. Labor will represent about $\frac{1}{2}$ this cost.

The best forms of accessory machines such as suction fans, mixers, igniting devices, automatic feeders, motors, and conveyors have been worked out along safe lines, and may now be regarded as sufficiently standardized for all practical purposes.

The elements of the design of a plant are so flexible that the automatic system can be adapted to almost any kind of a site or building.

An efficient plant embodying the continuous system can be built at a cost of from $80 to $200 per ton day capacity, depending upon locality and freights, labor, whether wood or steel construction, etc. This includes conveyors, motors, and everything but shed. The average cost of plants installed to date is less than $150 per ton day capacity. In fact, some of the large operators who have installed both pots and sintering machines declare that the installation of the latter is very much more economical than any other form of roasting appliance now known to the art. A close study of the facts indicates that this is the logical result of the high efficiency factor previously referred to.

In the matter of fuel for igniting the charge, many different means are employed; gas, gasoline, fuel oil, coal-burning furnaces and wood-burning down-draft braziers, but gasoline has found the widest use because of its convenience and simplicity. On an average $\frac{1}{2}$ gallon of gasoline is used per ton of ore treated.

The items of repairs has been found to average very low. The machine moves so slowly (pallet speed of 10 to 30 inches per minute) that the wear on the machine should be very small. The principal item is grate bars, and these if made of proper quality of metal should yield long service. Recently malleable iron grate bars have been used with highly satisfactory results. In some plants that have been operating for three years, the total cost of repairs, lubricants and other supplies has not exceeded an average of 5 cents per ton.

The superior physical quality of the Dwight & Lloyd sinter has already been referred to. The material is unquestionably a unique metallurgical product and possesses many valuable properties heretofore not combined in any one material. The metallurgists who have had occasion to smelt large quantities of it unanimously agree that its beneficial effect on the blast furnace is very marked. Furnace speeds have been enormously increased (from 20 to 60%) accompanied by a diminished flue-dust production and almost a total absence of the furnace troubles that have heretofore been considered as unavoidable. Import-

ant savings have also been made in the amount of fuel required for the smelting (10 to 30% reduction) which is undoubtedly due to the fusibility of the D. & L. sinter, the ease with which its metals can be reduced on account of the porous, cellular structure offering an enormous surface of contact for the furnace gases, and finally its intimate mixture of ingredients already prepared in a most efficacious manner for smelting.   As food for the blast-furnace, it may well be termed "pre-digested."   Both theory and practice agree that this is so.

The Dwight & Lloyd apparatus is now being used in Germany in connection with the manufacture of sulphuric acid.*   No such application has yet been made in this country, though it is probable that it will be before long.   Gas tests made under ordinary conditions of roasting and sintering sulphides show a range of SO_2 varying from 2 to 7%.   By taking a few precautions, the richness of the gas can be maintained at a suitable point for acid making.

An interesting possibility here presents itself in the tendency of a portion of the sulphur in a sulphide charge to retreat downward in advance of the slowly descending zone of fire, and liquate from the cake in the form of drops of molten metallic sulphur. Under favorable conditions as much as $\frac{1}{4}$ part of the sulphur contents has been extracted in this way, but where rapid sintering is desired, this is purposely avoided.   No effort has yet been made to utilize the principle, but perhaps in connection with some phase of the ever-troublesome smoke problem, some useful industrial application may yet be developed.

* " Manufacture of Sulphuric Acid," M. Hasenclever, Eng. & Mng. Journal, Jan. 20, 1912.

NOTES ON BAG FILTRATION PLANTS

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Notes on the Baghouse

In connection with lead blast furnaces and leady copper matte converters

The use of the baghouse for filtering out fumes produced in certain metallurgical operations is not new in this country. There are no patents in force at this time to my knowledge, which might hamper such use.

Up to about 25 years ago the baghouse was only employed in connection with either zinc works, which produced the oxides, to be sold for paint, or as an auxiliary in extracting lead from high grade oxidized ores and from galena concentrates in the Missouri and neighboring lead regions; this work being then done in the Scotch or American hearths and in very low slag furnaces, in all of which was produced an excessive amount of oxides which were also finally sold as paints. The literature[1] on the above is abundant in text books and periodical publications.

The first large baghouse to be used permanently in connection with lead blast furnaces was erected, as far as I am aware, about 1887 at the Globe Smelting Company's Works in Denver, Colo., under the Lewis-Bartlett patents. The brick flue, running from the blast furnaces to the baghouse was only a few hundred feet long. As their furnaces at that time were run with hotter tops than should have been the case, the gases could not be cooled in the flues before reaching the baghouse to a sufficient degree, to allow most of the H_2SO_4 to drop out before reaching the bags

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[1] Ingalls, Production and Properties of Zinc pp. 13 and 14. Ingalls, Lead and Zinc in the U. S. pp. 67, 68, 69. Ingalls, The Metallurgy of Zinc pp. 669, 673. Hofman, Metallurgy of Lead, pp. 131, 144, 145, 393.

and to prevent the latter catching fire rather too often.   In short, the experiment was not counted an unqualified success among metallurgists and not imitated for many years.   About 1902 and even earlier the lead and other ores in the West had mostly changed to sulphides, so that much roasting was necessary preceding blast furnace smelting and lead losses became greater in consequence, than before.   At that time and in subsequent years I had many experiments made at various Works of the American Smelting & Refining Company with a view of reducing the temperature of the gases below the danger point and of dropping out most of the H_2SO_4 and As_2O_3 in the flues, before the gases reached the fan and the bags.   As the length of the flues in nearly all the Works had been much increased before this time with a view of settling flue dust and part of the fumes by means of Freudenberg plates, or interposed dust chambers with many partitions, it remained to replace parts of the brick flues in the proper places with sections of iron or of concrete flues for the purpose of increased radiation.   It was finally found, that in flues of the proper large size and of a length of 1,000 to 1,800 feet (depending on the outside average temperature in the locality) there was no difficulty in accomplishing the end sought.   It was also found, that with the above length of flues and the devices put in them to give large surface contact for the dust and fumes, very nearly all the silver and gold were precipitated in the flues, hardly any reaching the bags, while a large amount of lead in the shape of fumes travelled on and would have been lost without a filtering apparatus or some spraying arrangement, such as is in use in Tarnowitz.   The baghouse was considered preferable in this country and gradually quite a large number of these were attached to various works.

The object of the present paper is to discuss the employment of the baghouse, in connection with lead blast furnaces and with such copper converting plants, as treat mainly leady copper mattes, derived from lead blast furnace smelting.

The writer has never seen the economical side of such employment in print, and it will therefore perhaps be of interest to give some figures in that connection in addition to a brief description of certain establishments.

Out of a considerable number of smelting plants which employ the baghouse for the collection of fumes, two have been selected for illustration, each of which is representative of its class.

One of these treats the ordinary ores of the West, as they come into the market and are bought by the custom smelters. The furnace charges made from these ores are low in lead (10–12%) and in the precious metals. They also contain much moisture and up to 4% S.

The other treats the slagged oxides, coming from a very large lead refinery, leady copper mattes for concentration, and some ores necessary for slag making. The furnace charges in this case are high in lead, often 40% and over, and rather low in the precious metals. The same Plant also converts concentrated, leady, copper matte to blister copper. A separate baghouse is provided for this operation.

I am indebted to Mr. A. H. Richards, Assistant Superintendent of the Murray Plant of the American Smelting & Refining Company for a detailed description of its baghouse and the operation of same, together with a number of drawings, all of which is given here with very few changes.

The Murray Plant Bag House

General construction of Bag House and Accessories

The Murray Plant bag house was constructed in the year 1906–1907, and was put into actual use July 7th, 1907. The total cost for construction of same was $127,194.89 and included in this amount was cost of 4032 cotton bags, of distributing flue the entire length of Bag House, one 15′ x 6′ Buffalo Forge Fan, one 100 HP motor, fan house, small bath house, and about 100 feet of 17′ x 16′ brick flue to connect with blast furnace flue.

Construction and size of Bag House Building

The outside dimensions of Bag House are 216′6″ x 90′6″, height to roof trusses being 51′6″. Construction is of brick, outside walls being 21″ thick for a height of 16′, then reduced to 17″ for 18′, and to 13″ for the remaining 17′6″. 24″ brick

buttresses projecting 4 feet at floor line and tapering to about 1 foot at roof truss line, are built into the walls to stiffen them. 15 buttresses are built on each side of the building and 5 on each end.

Roof Construction

Drawing No. 2075 will show in detail the evolution in the construction of the Bag House roof. Originally the roof consisted of 2 x 6 rafters spaced 3 feet apart and fastened to 6″ channels by means of an angle clip which was bolted to the rafters and to the channels, 1″ Oregon fir common boards nailed to the rafters and 3 ply elaterite roofing nailed on these boards with vertical seams, the seams being laid in tar paint. This roof commenced to leak after a few months due to the corrosion of the brads holding elaterite roofing in place. These brads were eaten to such an extent that when a compartment was open and a slight wind was blowing, the elaterite roofing would be lifted off from the boards in places. Considerable patching was done during the first two or three years the Bag House was in operation. After two years the nails holding the 1″ board sheeting were eaten to such an extent that a heavy wind blew about 1-3 of the roof off from one compartment. The sheeting itself showed dry rot, but would have lasted some time if the nails had not been partially eaten up. It was evident that nails could not be depended upon for holding the roof down. In the early part of 1910 a new roof was put on top of the old roof in No. 1 compartment, allowing an air space between the two roofs. 2 x 4 battens, 3′ centers, were laid on top of the old roof, 1″ sheeting nailed on top of this, elaterite roofing with horizontal seams nailed to this, allowing a lap of about 4″: 2 x 2 battens spaced 3′ centers were put on top of the elaterite roofing and the two roofs were bolted to the 6″ channel purlins. To give added strength, 3″ x 6″ pieces 3′ long were put on the 6″ channels, between rafters, the 2 x 4 and 2 x 2 inch battens coming directly on top of these and bolts going down through them. This roof has been in use for over two years and is in excellent condition. The air space appears to prevent any condensation and keeps bags dryer than in other compartments. As this new roof over the old one was rather an experiment and

SKETCH SHOWING CONSTRUCTION
OF
BAGHOUSE ROOF

AMERICAN SMELTING & REFINING CO.

MURRAY, UTAH

DRAWING № 2075
DATE MAR. 16 - 1912
SCALE ½" = 1 FT
FILE № 378/L
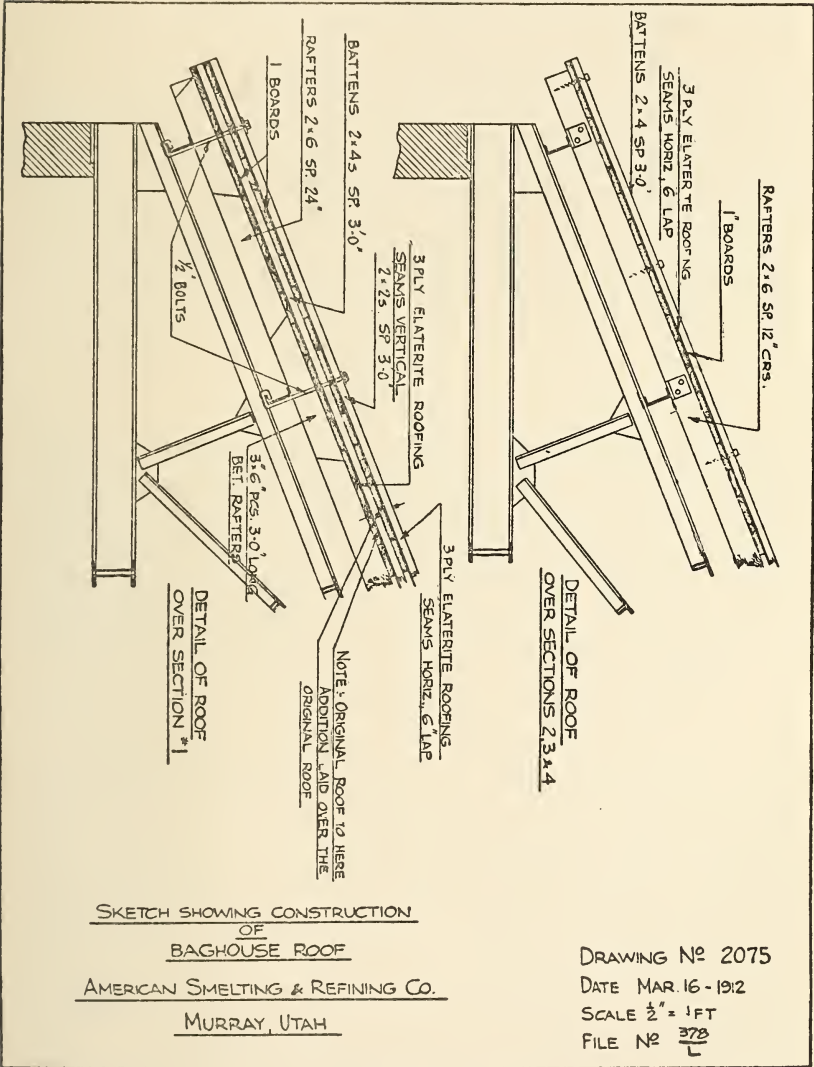
the other three compartments were badly in need of a roof, new roofs were put on them similar to upper sketch in drawing No. 2075.   The old rafters were in fair condition but additional rafters were put in, making the rafters 1 ft. centers; 1″ sheeting was nailed on top, then elaterite roofing with seams horizontal, then 2 x 4 battens 3′ apart.   Lag screws were used to hold 2 x 4 inch battens to the rafters, making a roof that will stand as long as the rafters are good, or until the elaterite roofing disintegrates. To prevent 2 x 4 battens from cutting into the elaterite roofing a washer of elaterite was used under all of them.   The roof of these three compartments has been in use a trifle under two years but has never leaked and is in good condition.

Exhausting Gases from Bag House

Each compartment has a steel stack coming out from the apex of the roof and about the center of the compartment.   The four stacks are each 6′ in diameter and 78′ high.   The general arrangement of these stacks on the Bag House can be found in drawing 1869.   When the stacks were built the joints were lapped in such manner, that when there is condensation some of the moisture finds its way to the outside of the stack, continually giving them an unsightly appearance.   This also weakens the stacks due to the corroding influence of the weak acid solution going through the joints.   The four stacks were found to be corroded to a considerable extent and one or two holes can be observed at a distance of about 32 feet from the top.   About every two years the stacks have been painted on the outside.

To take care of the condensed moisture that trickles down the inside of the stacks, a lead lined pan is hung under the stack at a sufficient distance so as not to interfere with the draft.   From this lead lined pan a lead gutter runs out on to the roof.   At certain times of the year there is considerable solution caught in these pans and runs on to the roof.   We had some very sad experiences by trying to use an iron elbow and iron pipes to lead this solution to the roof, as the iron pipe would eat through and before we knew of the condition several bags would be ruined by the weak acid falling on them.   The lead gutter with the

lead connection from the pan solved this difficulty and gives very good satisfaction.   In the lower part of drawing No. 1869 is shown the lead lined pan and gutter under the stacks.

Compartments and Cellars

The Bag House is completely divided into four compartments by means of brick partitions running from the floor line to the apex of the roof.   These partitions are the same thickness as the outside walls.   Below the thimble floor, each compartment is divided into four chambers or cellars by means of 13″ brick partitions.   These cellars are 87′ x 12′, each one having a 4′ x 8′ iron door through which the dust can be removed.   For Plans of the different floors see Drawing 1756.

Distributing Flue

A 16′ x 16′ distributing flue built of brick, jack arch roof construction, runs the entire length of the Bag House and connects with each of the 16 cellars by means of 42″ bulls eye "Valves." The dampers or gates for these valves are made of 1-4″ sheet steel stiffened by angle irons.   These dampers are hinged at the top and open into the distributing flue.   They are operated by means of 2″ gas pipe stems running through the distributing flue.

Return Flue

On top of this distributing flue is another smaller flue 11′ x 16′ running the entire length of the Bag House and connected to each cellar by means of 34″ "bulls eye" valves.   The dampers in these valves open into the cellars and were originally operated by means of gas pipe stems running through the flue, but later, due to the stems corroding and not operating with ease, they were abandoned and the dampers are now closed and opened by means of a chain on the thimble floor.   This upper flue is connected to the Bag House Fan flue by means of a 6′ sheet iron downtake containing a butterfly damper, and is used to conduct the gases from any chambers that are in the sintering process, back to the fan and thence into the distributing flue, where they are distributed to the cellars in which the fume is accumulated.

Thimble Floor

The Bag House thimble floor, as has been previously stated, is divided into four compartments, each compartment containing 1008 steel thimbles 17″ inside diameter and 10″ high.   These thimbles are made of No. 14 steel plates and are riveted to the steel floor which is made of No. 10 steel plate.

Hanging and Shaking of Bags

The bags are 18″ in diameter and 30 ft. long.   The original 4032 cotton bags were wired at the top and suspended by means of this wire from hooks fastened to small 3″ channel irons.   To shake the bags it was necessary to completely air out the compartment and then go in between two rows of bags, shaking the bags on each side by hand.   In 1910 each row of bags was hung on a 1-$\frac{1}{2}$″ gas pipe which was suspended by a "U" bolt from the old bag hooks.   By means of a lever on the outside of the building, this gas pipe could readily be given a horizontal motion of about 5″, the pipe coming to a sudden stop by means of a collar striking an iron plate on the inside of the Bag House wall.   In addition to the horizontal motion, about 1-2″ vertical motion is realized, which motion is rather beneficial in removing the dust from the bags. 42 bags can thus be shaken by means of one lever; this requires not more than 3 or 4 minutes.   This shaking device has proved extremely effective at the Murray Plant and has many advantages, I believe, over the system used at the Selby Plant, where gas pipes are run on ball rollers.   The ball runs and ball rollers have a tendency to accumulate moisture and fume making it necessary to clean same at intervals.   The construction of ball runs and rollers is a more expensive construction than the "U" bolt hangers; also no vertical motion is realized when the gas pipe is run on ball rollers.   When this shaking device was installed it was decided to have all bags folded over for 3 inches and sewed across at the lower part of the fold.   This allowed us to put a broom handle or any piece of round stick through the bag and hang up by means of this stick.   The old method of gathering the top of the bag and fastening it with wires has a tendency

to allow the dust to stick in the folds and in time to rot the bags at the top. With the stick running through the fold, the bags stay open in better shape and also there is no wire to come in contact with the fibre of bags. Drawing No. 1901 will show the general design of the bag-shaking device. Our method for hanging bags to the shaker by means of "S" hooks is given in the lower and middle portion of this drawing; no wire is used at all. We find this means of hanging the bags very simple and effective.

Bags, Cost, Etc.

4032 cotton bags 18″ in diameter by 30 ft. long were originally installed and put into use July 7th, 1907. These cotton bags cost $2.136 apiece, F.O.B. Murray, and were purchased from the Denver Dry Goods Co., Denver, Colorado. At the end of two years it was computed that the average life of the cotton bags was 17 months, 11 days. Owing to no drip pans being provided under the four steel stacks many bags were lost in the first eight months, due to condensation of moisture in these stacks and the consequent dripping of this moisture (containing small traces of acid) on to the bags. In fact during the first eight months 588 bags were replaced. The leaky roof after about one year's run was also instrumental in destroying many bags. Had it not been for these two items of weak construction, there is no doubt but what the life of cotton bags would have been longer than 17 months, 11 days. The rotting of these cotton bags generally commences near the thimble and to a less extent near the top.

In May 1909 No. 4 compartment was equipped with woolen bags purchased from Buell Mfg. Co., St. Joe, Mo. These bags cost $4.7185 apiece F.O.B. Murray. The compartment was put into commission May 19th, 1909, and to date there have been no bags replaced. About 12 bags have been lowered one foot during this time due to rotting near the bottom. The fibre of the bags is quite strong as yet, and I would estimate that the average life will be somewhere around four years. Shortly after these bags were put into commission about 1-3 of the roof on No. 4 was blown off, and a rain storm occurring at the same time,

rather drenched at least half of these woolen bags with no apparent ill effect. The woolen bags in this compartment are all hung by means of wire twisted around the tops of the bags. At the present time we have no cotton bags in service, and are completely equipped with woolen bags with the exception of 273 thimbles that are not now in use, but will have woolen bags put on them shortly. These 273 thimbles were reserved to use up the old cotton bags, the last of which were rotted away this last winter.

Below is a tabulation of the bags used to date.

			Cotton	Woolen
Original installation July 7, 1907,	4032	bags	4032	. . . .
Year ending May 1st, 1908,	752	"	752	. . . .
Year ending May 1st, 1909,	3107	"	3107	. . . .
Year ending May 1st, 1910,	4073	"	2088	1985
Year ending May 1st, 1911,	1493	"	. . . .	1493
10 mos. ending March 1st, 1912,	281	"	. . . .	281
Total,	13738	"	9979	3759
On hand in Bag House,	3759		none	3759
Total destroyed	9979		9979	. . . .

I might add that 90 woolen bags were purchased from the LaPorte Woolen Mills, LaPorte, Ind., at an average cost F.O.B. Murray, of $4.784. These bags were installed about two years ago and appear to be of a good quality.

570,012 square feet of filtering surface was provided for filtering 165,000 cu. ft. of gas per minute or at the rate of 3.45 sq. ft. of filtering surface per cu. ft. of gas per minute. When one compartment was shut down for sintering, however, there would be 2.59 sq. ft. of filtering surface per cu. ft. of gas per minute, and as it was expected that one compartment would be down most of the time, 2.5 sq. ft. was the basis used for calculating the number of bags needed.

Fan

A Buffalo Forge exhaust fan with impeller 6′ x 15′ brings the gases from the blast furnaces and forces them into the distributing flue and through the bags. This fan is capable of handling 1500 cu. ft. of gas per revolution at a pressure of one inch water. A 100 H. P. motor was formerly used when the fan was running at a speed of 93 revolutions and handling gases from eight blast furnaces. 60 indicated horse power required. Later a 50 H. P. motor was installed as we had only 5 and 6 furnaces in blast. Indicated H. P. required when running at 80 rev. per minute with the 50 H. P. motor is about 45. The counter shaft is equipped with the ring oil bearings and inner fan bearings are water jacketed. Very little difficulty has been experienced with this fan since its installation. Pressure in the distributing flue is usually about $\frac{1}{2}''$ of water, at times going up to $\frac{3}{4}''$ just before shaking the bags.

Operations

Conducting gases from furnaces

The blast furnace gases are conducted through 795 feet of brick flue and 70 ft. of sheet iron goose neck. 725 feet of this brick flue is 16′ x 17′, walls 17″ thick, roof 8″ jack arches, 5′ centers. 70 ft. of the brick flue is 16′ x 20′, walls 17″ thick, roof of reinforced concrete slabs 4″ thick. The sheet iron goose neck is 20′ in diameter made of $\frac{1}{4}''$ steel. Back of the blast furnaces is 262′ of 16′ x 20′ flue with sloping bottom, so that No. 1 Blast Furnace is about 865 feet from Bag House Fan, while No. 8 Blast Furnace is about 1127 feet from Bag House.

The temperature of the gases leaving the blast furnaces varies somewhat. Certain readings showed 119°F. in flue near No. 1 Blast Furnace and 85° F. at Bag House, showing a drop of 34° F. in the 865 ft. of flue when outside atmosphere was 64° F. No tests have ever been made at this Plant to find the SO3 and Arsenic that were thrown out in the flues due to the drop of temperature. The gases entering the Bag House were analyzed, however, at a time when there was quite a deterioration of the bags and were found to contain .003% SO3. During the winter

months the temperature of gases entering the bag house fan is often quite low.   The tabulated statement below will show the variation of temperature due to the season of the year.

AVERAGE TEMPERATURE OF GASES ENTERING FAN

	1909	1910	1911
January........................	81.4°F.	83.3°F.	79.9°F.
February.......................	73.0	93.0	79.9
March..........................	83.9	113.4	83.47
April..........................	91.3	107.2	93.4
May............................	102.0	106.3	108.65
June...........................	115.5	124.2	114.5
July...........................	117.1	131.0	123.9
August.........................	115.3	130.9	120.5
September......................	99.46	110.7	114.3
October........................	105.68	93.7	93.9
November.......................	103.02	89.15	90.7
December.......................	94.6	87.0	81.9
Average..................	98.52	105.82	98.75

Details of Bag House Operation

Gases are discharged from the fan into the distributing flue; dampers in lower "bulls eye" valves being open the gases enter the 16 cellars, filter through the 4032 bags; filtered gases discharging through the four steel stacks on top of Bag House. Once every 24 hours all the bags are shaken ,the operator using the outside lever and shaking each row of 42 bags.   All doors in compartments are kept closed while shaking bags so as not to allow the temperature in the Bag House to go down too low. The shaking operation is done by first closing the four dampers in the lower valves of one compartment, shaking the bags, opening the valves again and taking the next compartment in turn until the bags in all four compartments have been shaken.   This usually occupies the attention of one man for about three hours each day and could be done much quicker if necessary.

On the first of each month one cellar door in each compartment is opened, dust measured and representative sample taken. This is done for the purpose of taking stock. The doors are then closed and sealed by means of mortar. When the dust has accumulated to a thickness of about 24″, the entire compartment is closed off from the distributing flue, cellar doors are opened, valves connected with the return flue are opened and fume in the four cellars is lighted by means of hot coals. Formerly we set fire to the fumes across the front allowing the burning to gradually creep to the back end, but as this generally took from 10 to 14 days to complete the sintering process, we changed the method and now throw the hot coals all over the surface of the fume. This apparently gives a much better sinter and the operation is complete within 5 or 6 days so that the fume can be removed. Spontaneous ignition was rather frequent the first two or three years we were operating the Bag House, but of late it is rather rare. The exact cause of this spontaneous ignition has never been definitely determined at this plant, but we have found that it generally occurs when at least 24″ of fume has accumulated and believe it to be due to some development of heat when a large mass of the fume has accumulated. Of late we do not allow the fume to accumulate over 24″ and spontaneous ignition has been very rare.

The sintered fume is removed from the cellars by means of laborers using wheelbarrows and shovels, and is loaded into standard gauge box cars and shipped to the Globe Plant at Denver. While removing the fume from the cellars every tenth wheelbarrow is reserved as a sample to determine the values of the lot, this sample being carefully coned, quartered and the reject loaded into cars with the main portion of the fume. Except at times when sintered fume is being removed from the cellars, one man on day shift attends to the operating of the Bag House. For the 16 hours he is away from the Plant our H & H oilers come over to the Bag House once every two hours, take a temperature reading and look after the oiling of the bearings. These temperatures are taken just at the inlet of the fan. All laborers are required to take a shower bath and change their clothes before leaving the bath house. The men removing sintered fume have

	Oz. Au.	Oz. Ag.	Lbs. Pb.	Lbs. Cu.
Total metals in charge smelted...............	17,319.854	3,472,681.90	50,079,080	1,834,355
Total metals in 1124 tons fume recovered....	16.754	2,948.35	638,861	1,295
Per cent of metal in charge recovered in fume...	0.097	0.085	1.276	0.071

	Oz. p. ton	Oz. p. ton	% Pb.	% Cu.
Assay of the 254,794 tons of charge.......	.068	13.629	9.827	.36

	Oz. Au.	Oz. Ag.	% Pb.	% Cu.	% SiO^2	% Zn.	% S.	% As.	% Fe.
Assay and analysis of the 1124 tons unburned fume recovered......	.0149	2.62	28.42	.058	2.9	3.2	5.2	36.6	5.1
Assay and analysis of sintered fume	.014	2.4	27.3	.05	2.5	4.2	4.2	33.5	5.2

Recovery of fume etc. for entire operation of Murray Plant Bag House, July 7th, 1907, to Feb. 29th, 1912

	Charge smelted	Fume recovered	Assay per ton Fume recovered	Percent of charge Contents smelted	Value of metals recovered
Weight........	1,666,857 tons	13,857,645 lbs.		0.415%	
Gold.........	256,874.986 oz.	125.595 oz.	0.018 oz. p. ton	.049	$2,587.10
Silver.......	24,725,274.19 oz.	14,885.62 oz.	2.15 oz. p. ton	.063	8,028.25
Lbs. Lead....	354,693,161 lbs.	4,492,288 lbs.	32.4%	1.269	141,745.02
Lbs. Copper..	16,521,715 lbs.	1,947 lbs.		.0118	330.15
Total					$152,690.52

Operating Expense

Labor........	$16,440.05
Motive Power....	13,542.10
Supplies & Repairs....	46,871.28
Total....	$76,853.43

Outcome

Treatment charge 6929 tons at $10.00...	$69,290.00
Operating cost....	76,853.43
Total cost....	$146,143.43
Value of metals recovered....	152,690.52
Gain for 4 years & 8 months....	$6,547.09

very little trouble with sore legs and faces, providing they are careful to clean thoroughly after their day's work.

When the Bag House was first started, rubber suits were furnished to the Bag Shaker and Foreman, also helmets, but later it was found that these were not essential to the men, and that the main secret of good health of the laborers was to bathe properly and to not eat their lunches with dirty hands and faces. When the bags were shaken by hand, and it was necessary for a man to go into the compartment, between the rows of bags, we had one man shake bags for two years with no ill effect. He realized the necessity of thorough cleanliness.

Assay and Analysis, also recovery of fume for 8 months ending December 31st, 1911

Tons of charge smelted in 8 months . . . . 254,794 tons.
Tons of fume produced in 8 months . . . . 1,124 tons.
Per cent. of charge recovered as fume 0.44114%
Pounds of fume recovered per ton of charge 8.8228

The following additional figures may be of interest:

Cost per ton of original charge smelted for operating Bag House $0.0461
Gain per ton of charge smelted in operating Bag House 0.00393
Fume recovered per ton of charge smelted . . . . 8.31 lbs.
Average cost of operating Bag House per month $1,372.38
Average gain in operating Bag House per month 116.91
Total cost for replacing bags in 56 months 30,453.6285
Average cost per month for replacing bags during 56 months 543.8148
Cost per ton of charge smelted for replacing bags .0186
Total bags replaced in 56 months 9979
Total bags replaced per month 178.2

	Au.	Ag.	Pb.	Cu.
Average assay of unburned fume for 56 months	.018	2.1	32.4	.014

The above shows the apparent gain from the installation of the Bag House in 4 years and 8 months.   But the figure does not include interest on the original investment, nor a fair percentage of same for amortization.

If these two items were included, as they should be, we would have:

Gain in 4 years and 8 months................ $6,547.09
6% interest on cost of $127,194.89 for 4 yrs. & 8 mos.
                                    $35,614.57
5% Amortization for 4 yrs. & 8 mos.... 29,678.80   65,293.37

Net loss in 4 years and 8 months $58,746.28

On account of the advisability of extracting the high percentage of As_2O_3 from the fumes, The Murray fume has to be shipped to another Works of the American Smelting & Refining Company, which is equipped with a Plant for that purpose and works up fumes from a number of lead smelting plants.   It is not good financial policy to provide every Plant with an Arsenic establishment, for it would be idle three quarters of the time.

But if the Murray Plant had happened to be the one provided with its own Arsenic establishment, the cost of working up the fumes would not have been over $5.00 per ton, thus diminishing the loss shown above by ....$34,654.00 leaving the loss for 4 years and 8 months still....$24,101.28

The above figures show, that, taking the immediate financial outcome only, bag houses are not profitable in lead smelting works, which treat ores low in the precious metals and especially in lead.   But there is another consideration to be taken into account, when Smelting Works, are like Murray, located in the midst of farms and villages.   This is the elimination of constant law suits, which are bound to be of frequent occurrence, when As_2O_3 and the sulphates of lead and other metals are permitted to drop down into the fields.   It must be decided therefore in the case of each smelting plant, whether it is not good business to expend the money for a bag house and for operating the same, though an immediate profit is not apparent.

Mr. W. T. Page, Manager, of the Omaha Plant of the American Smelting & Refining Company has kindly furnished me with data on the bag houses of that Plant, which appear below with slight changes and some explanations. The figures given are reproduced as originally sent me.

Bag Houses of the Omaha Plant[1]

A distinctive feature of the Omaha Plant is said to be its fume condensing facilities. The lead loss of the Plant does not exceed 1% though none of the gases of the Lead Refinery reverberatories enter the bag house. There are three systems of fume condensing, as follows:

1. Converter Bag House system.
2. Blast Furnace Bag House system.
3. Zinc Oxide Bag House system.

No. 1.   Converter Bag House System

The gases from two stands of Acid Lined Converters (upright) and one Basic Lined Converter (Peirce-Smith type) lead into a suspended flue of cross section as shown in accompanying drawing. It is made of sheet steel and heavily painted inside and outside with a graphite paint. Its length is 1250 ft. This flue leads to a fan 12 ft. in diameter and 6 ft. wide, which runs at 140 R.P.M. and requires 25 H.P. The fan discharges into a bag house containing 940 bags, divided into two compartments. The cellar is on the ground level so that carts can get in to load out the fume. It is 10 ft. high from floor to thimble level. Many have advocated a cellar with a sloping floor and a conveyor of some design to remove the fume, but every practical design submitted to date represents too large an expenditure for the advantage in labor saving obtained.

Sometimes it is found necessary to clean out the house three

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[1] No drawings of these bag houses are published, as the construction is in general the same as at the Murray Plant. The one difference is in the shaking arrangement of the bags, which is here done by beating the bags with long slender wooden rods, suspended from a shaft for each row of bags. This shaft can be turned to and fro by means of a lever outside of the building thus giving the rods a beating motion.

times during the life of one set of bags. The water put on the fume (which is nearly white and like flour) to make it possible for the men to handle it, necessarily allows the bags to absorb moisture to an excessive degree. This will shorten the life of the bags.

With the acid lined converters one set of bags, on an average, will filter 1600 tons of lead sulphate fume, representing 4500 tons of Blister Copper.

The secret of long life of bags seems to be the use of long flues conducting the gases to the bag house and constructed of a material which permits rapid radiation, also a good vacuum above the bags, so that the gases are drawn away rapidly. With this in view, Omaha has connected the converter bag house to a 300 foot, self-supporting, brick-lined, steel stack.

Average pressure and vacuum are as follows:

Vacuum at fan up to 0.8 inch (water).
    "   " converter 0.07 to 0.15 inch.

In the bag chamber there is a variation in pressure of 0.025″ to a vacuum of 0.07″. At the base of the stack, with no other gases entering, there is a vacuum of 0.3″. With a small reverberatory furnace leading into the stack to supply it with heat in cool weather, there is a vacuum at the base of 0.7″, while the figures on the bag chamber remain the same as given.

The pressure in the cellar of the bag house goes up to 0.56″. With a vacuum of 0.3″ at the fan a vacuum of 0.1″ is found at the converter, with a very regular increase along the flue towards the fan.

Following are some data concerning the converter bag house:

Size: 102′ x 60.8′ x 40′ Outside.
Walls 17″ with two compartments up to the thimble floor; 13″ above thimble floor.
Thimbles 18″ diameter, placed 2′ 4″ centers.
Bags 28′ long, 940 in number.
Cubic feet in flue, 68,000.
Cubic feet in cellar, 67,000.
Cubic feet in bag chamber, 174,000.
Sq. Ft. filtering area of bags, 124,000.
Production about 45 tons of blister copper a day, from Matte assaying 45% Cu., 23% Pb., 18% S.

The blowing engine at converters runs about 15 hours out of 24, or about 52,000 revolutions per day with delivery of 100 cu. ft. of air per revolution.   Pressure: 17 lbs. per sq. inch.

An example of flue temperatures, etc., follows: These vary of course with the temperature of the outside air.

Flue temperatures (F) on first or slag blow, outside air at 80° F.

Back of converter.....................	700.
80 ft. from converter	557.
200 " " "	464.
400 " " "	390.
600 " " "	345.
800 " " "	300.
1100 " " "	266.
At Fan	246.
Bag House	152.
Stack	136.

FUME ANALYSIS

	Ag. Ozs.	Pb. %	Zn. %	Combined SO_3	Free SO_3			
200' from Converter	4.2	56.0	1.6	25.13			0.25	
400' " "	3.0	60.0	1.6	23.13			0.55	
600' " "	2.4	64.2	1.3	22.82			0.15	
800' " "	2.0	64.6	1.3	22.60			0.00	
1100' " "	2.1	63.0	1.5	23.33			0.00	
1200' " "	1.6	64.4	1.6	20.60			0.00	
Caught in Bag House ⎫ Caught in Bag House ⎭	2.8	66.0	1.9	21.63	As %	Sb %	Te %	Se %
					2.0	0.2	0.2	0.24

SO_3 DETERMINATIONS IN GASES.

At Converter	At Fan
.038%	.020%
.027%	.025%

SIMULTANEOUS SO₂ DETERMINATIONS

At Converters	At Fan	At Middle of Flue
3.62%	2.055%	1.70%
2.34%	1.21%	1.82%
4.46%	2.68%	
2.38%	1.45%	

SO₂ IN BAG CHAMBER

Top	Bottom
2.06	2.06
1.30	1.60
1.52	1.40

Other temperatures taken with corresponding outside air temperatures are as follows:

Temperature at Fan F°	Air Temperature
225	45
192	59
239	61

and at another time:

TEMPERATURE 60 FT. FROM CONVERTER

	Deg. F.
Slag blow	525
" "	520
Copper blow	625
Slag.........................	675
"	675
Copper "	600

The average temperature in the bag chamber is 140° F. with a variation from about 100 to 180.

The material used in bags is Osnaburg Sheeting put up in rolls of about 600 yards, 57″ wide at 15c per yard. (It costs about 15c per bag to cut and sew the material). This width allows the bags to be made 56 1-2″ in circumference, or just right to fit an 18″ thimble.

Bags are made up 28 ft. long with a loop at the closed end through which passes a 1″ pipe or stick to support the bag.

The beater arrangement for the bags is as previously mentioned in a foot note. The fan is always shut down while beating the bags. This is done about twice a day, or as often as necessary.

FUME PRODUCED FROM NO. 1 BAG HOUSE

Calendar Year	Flues Tons	Bag House Tons	Total Tons
1906.	1,531	1,670	3,201
1907.	1,045	3,187	4,232
1908.	900	3,333	4,244
1909.	568	1,207	1,775
1910.	619	2,532	3,151
1911.	546	2,703	3,249
	5,220	14,632	19,852
To May 1st, 1912. . .	350	725	1,075

As giving some measure of fume collected per ton of blister copper produced the following may serve:

Recovered Feb'y, 1911, Fume   454 Tons.
                                                    Fan   152   "

                                                            606   " -15% H_2O = 515 Tons.
                                                            100   "   from flue   100   "

1667 Tons Blister Produced . . . . .615 tons fume.
       1 ton   "   "   . . . . 0.368 tons "
       1   "   "   "   . . . . 0.221   " lead.

	Costs	Tons of Blister	Per Ton of Blister
1905–6................	$3,995.14	7,166	$0.558
1906–7................	6,590.24	8,128	0.811
1907–8................	5,319.36	11,095	0.474
1908–9................	6,451.62	11,084	0.582
1909–10................	10,032.37	8,260	1.214
1910–11................	5,842.50	9,022	0.648
1911–12................	8,871.51	7,639	1.163
1912 to May 1st..........	3,762.80	2,200	1.210
Total...............	$50,865.54	64,594	
Average per ton Blister	....	....	$0.787

In order to give some idea of the importance of the Bag House in connection with the converter establishment I present a number of pertinent data, and finally the financial outcome for the fiscal year 1910.

These data have been kindly furnished by Mr. Judd Stewart, Gen'l Auditor, American Smelting and Refining Co.

LEADY MATTE TO CONVERTERS

	Matte	Iron	Coppery Litharge	Ore	Flux & Linings	Tons Total	Slag	Cleanings	Fumes	Total Tons
Basic Lined..........	11885	167	2424	668	1890	17034	7765	1964	1253	28016
Acid Lined..........	9659		2496		2671	14826	6427	3443	1055	25751
	21544	167	4920	668	4561	31860	14192	5407	2308	53767

CONTENTS

			In slag, etc. Ret'd		Tons Total	
	Lead	Copper	Lead	Copper	Pb.	Cu.
Basic..........	4369	5766	3233	595	7602	6361
Acid..........	4309	4176	3003	649	7312	4825
	8678	9942	6236	1244	14914	11186

AVERAGE ANALYSIS MATTE

Au.	Ag.	Pb.	Cu.	NiCo.	SiO$_2$	Fe.	Zn.	S.	As.	Sb.
.17	98.1	26.9	43.1	.40	.3	8.0	2.5	15.5	1.7	.76

FUME PRODUCED

Part returned to Converter

	Tons	Pb.	Cu.
	1849	1115	......
	1629	974	......
Basic	3478	2089	......
Acid			

SAVED IN BAG HOUSE

Tons 1551–5559.44	Ag @ .54	$3,002.00
2,650,774	Pb @ 4.30	113,983.00
10,818	Cu @ .08	865.00
		$117,850.00

DIRECT COST OR EXPENSES

Labor	904.16		
Power	1985.29		
Sup & Re	2953.05	5842.50	
Smelting 1551@$2.	3102.00		
Refining 1325@ 5.	6625.00	9727.00	
		15,569.50	
	Apparent Net Profit		$102,280.50

Less 6% interest on cost of Baghouse No. 1 (about $42,000.00)..........$2520.00
2½% Amortization.................... 1050.00
3,570.00

Net profit $98,711.50

No. 2.   Blast Furnace Bag House System

The blast furnaces are served by two bag houses similar to the converter bag house, and while either one can be cut out and cleaned, yet most of the time both houses are connected up. The flue leading to houses has cross section as shown in accompanying drawing. Length of Flue is 1,400 ft. Fan 14′ diam. with 8′ 11″ inlet running 140 R. P. M. and requires 62 H.P. The following figures give an idea of the duty of the blast furnace bag houses.

Period from May 1909 to Sept. 1911 inclusive

Fume collected in #2 house, 1275 Tons . . . . 50% Pb.
Fume collected in #3 house, 1348 "   . . . . 50% "

This carried about 1,300 tons of lead. The lead charge to the furnaces during this period was:

Lead in Matte . . . .   7287 Tons.
Lead in Ores   492   "
Lead in Ref. Bi-Prod . . . . : 23475   "
                                                    ———
    Total   31254   "

Of this 1300 tons, or 4%, went to the bag houses.

Analysis of Raw Fume before Sintering

Pb.	Cu.	Zn.	S.	As.	Sb.	Te.	Ag.
52.5	Tr.	3.0	5.4	14.2	1.6	Some	10 oz.

This contains a large amount of carbon and As_2O_3 and occasionally it ignites by spontaneous combustion and shrinks considerably in bulk.

In sintering the Arsenic not already present as As_2O_3 is oxidized and a portion volatilized. The fume sinters lightly and assumes a structure somewhat similar to coke. The arsenic is concentrated to about 20% by this operation and is found in beautiful crystals of As_2O_3.

Some Experiments were made on Roasting Burnt Fume to see how far the Arsenic could be Eliminated

Roasting 24 hours without fusing

1. Original . . . . 13.1% As.
   Final . . . . 3.6% As.
2. 3000 lbs. Fume . . . . 1000 lbs. Coal.
   Original . . . . 13.1% As.
   Final . . . . 3.7% As.

Burnt Fume taken from House since 1906

August 1906, #2 . . . . 1200 Tons.
June 1908, #3 1380   "
May 1909, #3 200   "
May 1909, #2 1150   "
October 1911, #2 1270   "
December 1911, #3 1498   "

There is also some fume collected in the last 300 ft. of flue to the extent of 280 tons per year.

The blast furnace *flue dust* taken from the flue (exclusive of the last 300 ft. mentioned) has been as follows:

1905–6 . . . . 453 Tons.
1906–7 666   "
1907–8 1180   "
1908–9 944   "
1909–10 471   "
1910–11 714   "

This *flue dust* is pugged with about 5% of lime and smelted in the blast furnace.

The bulk of the bags in these houses generally lasts for several years and is replaced when fume is burned. This has been as long as three years, filtering out 1498 tons of fume. Of course a few bags may have to be replaced in the interval.

The burnt fume carrying, say, 18% to 20% arsenic and 2% antimony, is fused in a small reverberatory furnace and the gases

led to a 300 foot stack.   This gives a slag, a mixture of oxides of Pb., Sb., and As., with about 10 ozs. Ag. per ton.   The arsenic remaining is about 5%.   A large amount of As_2O_3, very pure, could be recovered here, but the market does not warrant the necessary expenditure.

The fused material is allowed to cool in pots and is eventually charged alone into the residue furnace of the lead refinery (with a little coal if necessary), reducing a little lead from it to remove the silver.   The antimonial slag thus produced (called smoke slag at the Works) carries about 5 to 6% antimony and 3 to 4% arsenic.

This slag, when reduced in a blast furnace with lead skim-mings, makes a metal too high in As. and Sb. to make softening practical and is hardly a fit metal to dispose of as antimonial lead, as the arsenic is about 3% and the antimony less than 10%. Therefore the As. is removed by a special process of fusion.

Blast Furnace Bag House Operating Costs

1905–6	$7,990.28
1906–7	13,180.49
1907–8	10,638.73
1908–9	12,903.21
1909–10	10,347.43
1910–11	6,557.54
1911	8,228.45
1912 to 5/1	4,068.74
Total	$73,914.87

During the period 91,850 tons of lead were treated in the Blast furnaces, making an average bag house cost per ton of lead treated, $0.847.

And during this same period the total lead treated in the entire plant was 810,082 tons.   .0122 of 1% of this lead found its way to the bag houses, the saving thereon being 42c per ton of total lead treated in the entire plant.

When the burnt fume from the bag houses is fused, there is a material resembling speiss which separates out and freezes

to the bottom of the furnace. It can be smelted out by increasing the fire. Some of it is tapped with the slag and forms buttons in the bottom of the pots. The composition is as follows,— evidently mainly Telluride of Lead:

Pb...............................	62.0%
Cu...............................	3.0%
Fe...............................	Tr.
S...............................	2.0%
As...............................	4.0%
Sb...............................	2.0%
Te...............................	19.0%
Se...............................	2.0%

Pure Tellurium is recovered from this material by a chemical process. The market for this metal is very small and unsatisfactory.

<div align="center">SACK RECORD</div>

Converter Bag House No. 1		Blast Furnace Bag House No. 2	
		1st Bag House	
July, 1905,	960.		
Dec., "	940.		
Feb., 1906,	940.	Oct., 1906,	1040.
Sept., "	940.		
Nov., "	960.	Oct., 1908,	799.
May, 1907,	860.		
Sept., "	905.	Aug., 1909,	1020.
*May, 1908,	978.		
July, "	1040.	Dec., 1911,	595.
Dec., "	931.		
Mch., 1909,	940.	*2nd Bag House*	
June, "	940.		
July, "	940.	July, 1906,	1020.
Aug., "	940.		
Feb., 1910,	940.	June, 1908,	1062.
Aug., "	940.		
July, 1911,	1880.	July, 1909,	1020.
Dec., "	940.		
Apl., 1912,	940.	Oct., 1911,	690.
*Feb., 1908,	905.		

As some measure of the financial outcome of the operation of the bag houses used in connection with the blast furnaces (B.H. System No. 2) I give the following data kindly furnished me by Mr. Judd Stewart, General Auditor, American Smelting & Refining Company. They are purposely given for a year in which less than half as much material was treated in the furnaces than is usually the case. This is done to show that even under very unfavorable conditions the gain is satisfactory.

Material to Blast Furnaces

Fiscal Year 1910–11

	Tons
Copper By-Products	23,090
Ore	22,762
Lead Refinery By-Prod	13,710
Matte	2,672
Limerock	6,345
Burnt Lime	39
Iron	1,391
	70,009
Blast Furnace By-Prod	4,103
	74,112

Fume Collected in Bag Houses

580 Tons 1,615 ozs. Ag. at 54c		$872.00
460,000 lbs. Pb. at $4.30		19,780.00
		$20,652.00
Expenses	$6,558.00	
Smelt. 580 tons at $2.00	1,160.00	
Ref. 230 tons at $5.00	1,150.00	8868.00
Apparent net Profit		$11,784.00

Less 6% interest on cost of the two
    Bag houses, about $60,000.00..... $3,600.00
2 1-2% Amortization[1]............... 1,500.00   4,100.00

    Net Profit.......................... $7,684.00

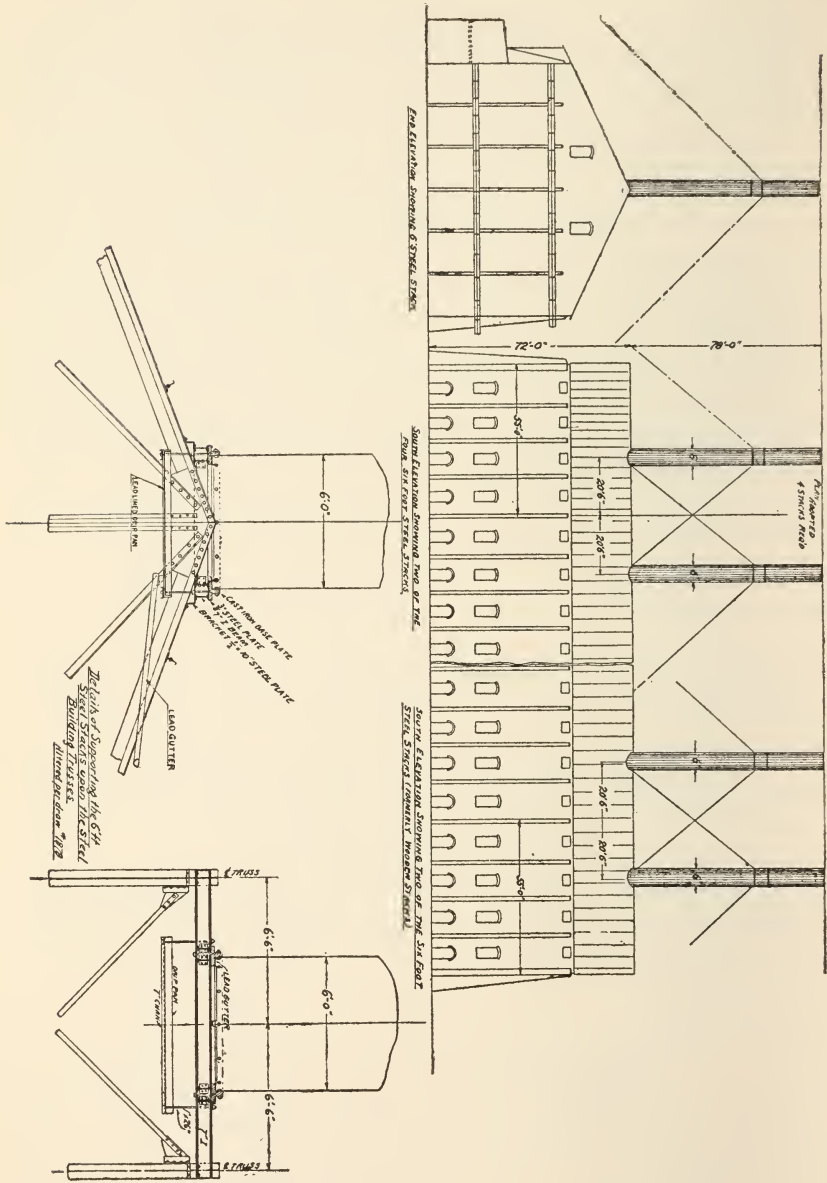### No. 3. *Zinc Oxide Process Bag House*

The fume from the Zinc Oxide Furnaces which treat certain by-products of the Refinery is led through 100 feet of brick flue and 500 feet of steel flue, same as converter flue, to a bag house of similar construction, having 120 bags, a 60″ fan, using 15 H. P. for draft. Gases enter bag house at about 190° to 200° F. This bag house filters about 40 tons per month of crude Zinc Oxide, which contains about:

Zn............................... 53.0%
Pb............................... 20.0%
Sb............................... 1.5%
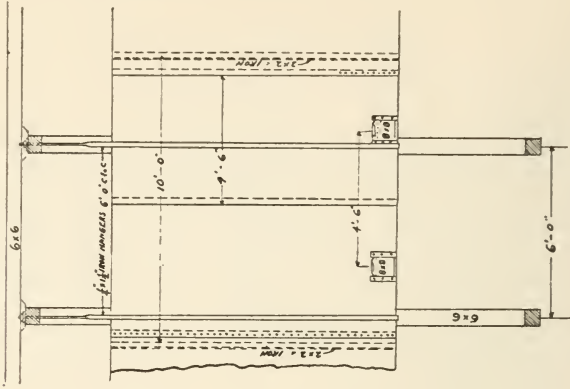As............................... 0.3%

Several Chemical methods have been proposed to convert this fume to pure ZnO, none of which have been put in operation as yet.

Fume from the cupels was at one time passed through a bag house, but a bagless chamber was found sufficient to collect the values, therefore the use of the bags was discontinued.
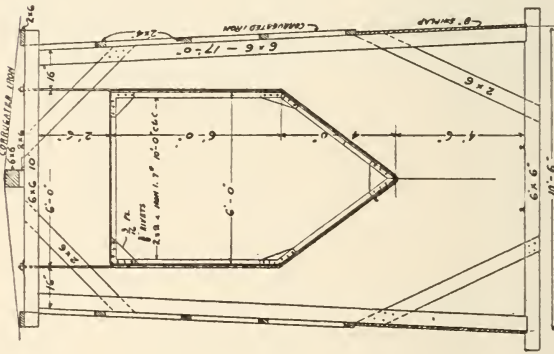
[1] The Works being a Refinery centrally located, their life is assumed to be twice as long as that of the Murray Plant, which has only a few mining districts to draw from.
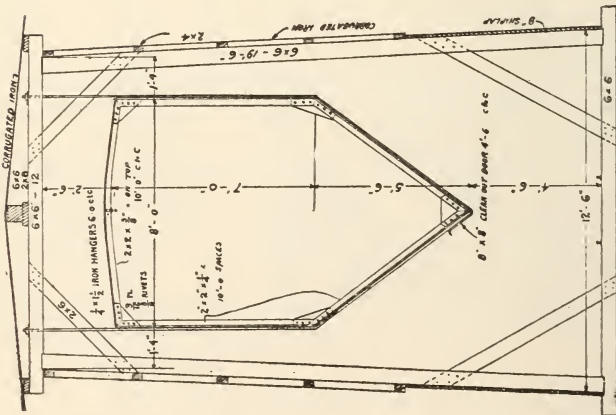
OUTLET STACKS for
Baghouse
American Smelting & Refining Co.
Murray Plant   Murray Utah
Drawing No. 1089

PLAN AND MACHINERY FOR FAN HOUSE
AMERICAN SMELTING & REFINING CO.
MURRAY PLANT
MURRAY - - - UTAH
DRAWING No. 1853

COPPER CONVERTER FLUE
1250 FEET LONG
12' DIA. FAN x 6'
140 R.P.M.
25 H.P. MOTOR

BLAST FURNACE FLUE
1400 FEET OF FLUE
14" DIA. FAN
8'-11" INLET
140 R.P.M.
55 H.P. MOTOR

THE DEVELOPMENT OF THE PARKES PROCESS IN THE UNITED STATES

Ernst F. Eurich

15 William Street, New York

Alexander Parkes patented in England in 1851,52, 53, a process for desilvering lead by means of zinc, making use of the greater affinity of silver for zinc than for lead, discovered by Karsten in 1842. The efforts in England to develop it commercially were not particularly successful, nor was the attempted introduction into the United States. This last was not surprising, as it was about the beginning of our Civil War, and at that period there was very little lead bullion produced in this country and there was no field for the process here. There was at that time on Staten Island an establishment that pattinsonized lead, probably from Rossie, N. Y. and Chester, Pa., and possibly some Spanish lead, but the production of lead bullion in the Far West had not yet begun. In 1864 Edward Balbach of Newark, N. J., patented in this country a process for desilvering lead by means of zinc, and in 1867 Edward Balbach Jr., patented a "movable blacklead retort with a neck, placed in a furnace" for the purpose of distilling the alloy of zinc, lead and the precious metals.

A brief comparison of the Parkes and the Balbach processes is interesting. The Parkes process as practised in England consisted in stirring into silver-bearing lead melted in a kettle, from one to two per cent of zinc, cooling, skimming the alloy of lead, zinc and silver that rose to the surface, liquating it in an iron retort, distilling the residual rich alloy in fireclay retorts, thus regaining the zinc, and refining the disilvered lead at a low heat in a shallow cast iron pan that formed the hearth of a reverberatory furnace. If the lead was too impure, it was first softened in a reverberatory furnace similar to the refining furnace. The Balbach process liquated the softened bullion in a reverberatory

with a cast iron inclined hearth, the eliquated metal flowing into a kettle which contained the requisite quantity of melted zinc, thoroughly stirred the contents and ladled into molds, transferred the bars to a second liquating furnace where they were heated to the melting point of lead. The eliquated lead was to be measurably free from silver, the residual rich alloy was distilled in a blacklead retort, the zinc given off condensed and the rich lead resulting, cupelled. The desilvered lead was refined at a high heat in a reverberatory built of firebrick. Impure lead bullion was first softened in a reverberatory furnace similar to the one most used for refinery. The Balbach process as patented had some of the features of the Parkes, and after several years of operation, about 1872, was so modified as to resemble it still more closely.

It is interesting to note that in the desilvering of lead bullion by means of zinc as practised to-day, the different operations are fundamentally identical with those outlined by Parkes. The failure of the process, or rather the lack of commercial success, was due solely to the apparatus employed; the shallow cast iron pans frequently cracked, and the fireclay retorts were not adapted to the work required of them. By the use of firebrick furnaces for softening and refining, and of blacklead retorts for distilling, those works, which later in the United States followed the lead of Parkes in their operations, made the Parkes process an unqualified success. This is the process by which most of the lead bullion in the civilized world is desilvered, and in many respects it has found its best development in American plants.

With the opening of the Pacific railroads, the exploitation of the Far West received a great impetus. Furnaces for smelting lead and silver ores were soon in operation and the lead bullion they produced, as well as some of the ore mined, sought a market in the East. Smelting and Refining Works were built at different points from the Atlantic to the Pacific, and by 1873, there were fourteen establishments in operation at which smelting of ores and refining of lead bullion was carried on. Two of these, the Pennsylvania Lead Co. near Pittsburgh, Pa., and the St. Louis Smelting & Refining Co., near St. Louis, Mo., certainly the former, followed exactly the lines laid down by Parkes, using

however blacklead retorts and firebrick softening and refining furnaces. The Germania Works at Salt Lake City soon followed suit, and of the remaining refineries, the Balbach Works at Newark, N. J., the Omaha Works at Omaha, Neb., and the Selby Works at San Francisco were soon the only survivors. These three used the Balbach process, but it had been so modified as to approach closely the Parkes model. By the end of the 70's or the early 80's various processes had been tried out, and only the Parkes remained.

However much the American Works differed in the details of operations, there was one feature that soon became common to all, the terrace construction of the desilvering plant. The softening furnace occupied the highest level, the desilvering kettle the next, the refining furnace below this, and lowest of all was the floor on which were the molds for the merchant lead. The charge finished in one apparatus was tapped and flowed into the next, and from the last was run into molds.

To trace the improvements in the construction and operation of the constituents of the plant, it will be simplest to consider each operation separately.

Softening Lead-Bullion. This operation is carried out in a reverberatory furnace. We early find the hearth of the softening furnace built inside a rivetted sheet iron pan, the material of the hearth being firebrick. In this construction, in use in 1873 and possibly earlier, the wrought iron pan prevents all leakage of lead out of the furnace, and the bottom of firebrick, laid as an inverted arch of six to nine inches rise, effectually resists the plumbostatic pressure of the charge. Adopted later in Europe and wherever Parkes process is used, it has become the standard construction.

To counteract the corroding action of the oxides at the lead line, and reduce the frequency of repairs, watercooling in various forms was resorted to. A two and a half or three inch water pipe was placed along the leadline next to the pan, or the sides and ends of the hearth were provided with water-jackets, or again the pan proper was placed in a second rivetted pan, leaving a space between the sides and bottom in which water circulated. This last was eventually abandoned as the cooling of

the bottom was not required and increased the fuel consumption. With the advent of magnesite bricks, these were used along the leadline in addition to the watercooling. Indeed in some cases they have led to the abandoning of watercooling except under the skimming door.

With the increase of the regular supply of lead bullion that could be depended on, the size of the softening furnaces increased from a capacity of 10 tons to 25, 40 and up to 90 tons. With hand-charging this was about the economical limit, but the introduction of a charging apparatus (first used at the National Works of the A. S. & R. Co.) which by means of an endless belt elevated the pigs and dropped them into a chute down which they slid into the furnace through a hole in the roof, has made still larger furnaces feasible, and a 210 to 240 ton furnace is now in operation at Omaha. In the matter of size of furnaces, England was for quite a time in the lead as 140 ton furnaces were in use there for a number of years before furnaces of a similar size were built here. The time required for refining a large charge in a properly proportioned furnace is no longer than is required for a smaller one, and, provided sufficient lead bullion is available to keep it in steady operation, a large furnace is more economical in labor, fuel and repairs than a smaller one.

The proper method of conducting softening operations has long been known, but not always observed. When its laws were transgressed the penalty followed,—a smaller direct yield of soft lead, a larger quantity of secondary products to be reworked, and greater costs. It is not necessary to go minutely into the details of the operations. The important points are melting the charge at a low heat and removing the dross that accumulates on the surface, which contains the greater part of the copper present: next, increasing the heat, admitting air, oxidizing the impurities susceptible of oxidation (mainly arsenic and antimony) and skimming off the arsenical and antimonial lead oxides. In some works crushed litharge free from copper was regularly used for the purpose of oxidizing the impurities. The advantages of reducing the lead of the litharge and at the same time oxidizing the impurities are evident. It is quite noticeable that as soon as the arsenic is pretty well eliminated, the oxidation of the charge

proceeds more rapidly, and it is for oxidizing the arsenic that litharge has been found particularly helpful.

For a time the lead smelter in the West, with few exceptions loaded into the lead bullion which he shipped, as much as possible of the dross the furnace made. Gradually it was appreciated that this procedure was not to his advantage, and it was modified. The Philadelphia S. & R. Co. at Pueblo, Col., was the first to systematically prepare clean bullion for shipment. It was drossed in the cooling pot before ladling, the resulting pigs were remelted in a large kettle holding a carload and after careful drossing, run into molds. At the furnaces controlled by the American S. & R. Co. the same practice is now followed. Such preliminary drossing, carefully carried out, renders drossing in the softening furnace unnecessary, and softening then consists merely in eliminating the arsenic and antimony. After softening, the charge is tapped into a launder through which it flows into the desilvering kettle.

Desilvering With Zinc. The kettles employed for this operation originally had a capacity of 10 tons, but this capacity has been increased to 60 or 70 tons. At first kettles were cast with a spout at the bottom through which the charge was tapped into the refining furnace. In 1877 at the works of the St. Louis S. & R. Co., the use of a syphon for transferring the charge from the desilvering kettle was introduced by A. Steitz. His example was followed by other works and the spout, no longer necessary, disappeared. In recent years a horizontal centrifugal pump driven by an electric motor has come into use for emptying the kettles. With large kettles, the centrifugal pump saves time. Furthermore its use will affect the design of the kettles and will have an influence in the arrangement of the constituent parts of plants that may be erected in the future, and may use it. No longer limited to a depth of forty inches, the kettle can be made deeper and capacities in excess of sixty tons will be entirely feasible.

The apparatus for liquating silver crusts varied at different works. At some it was a reverberatory furnace with an inclined cast iron hearth, at others, a kettle with or without a spout. In the former case an additional kettle was provided to receive the

eliquated lead. These various devices have been superceded by the silver-crust press which, producing a richer crust, is now in general use. It was first introduced at the works of the Pueblo S. & R. Co. by W. H. Howard, its inventor and patentee.

It is of prime importance that the zinc which is to effect desilvering, be well incorporated with the lead bullion. At first this was done solely by manual labor. In the later 70's steam-stirring was introduced at the works of the Pennsylvania Lead Co. It consisted in passing dry steam through the charge, producing a circulation which, combined with a rabbling of the surface, effected a good incorporation of the zinc which had been previously melted on the surface. Both of these methods have given way to the mechanical stirrer, invented by W. H. Howard and patented in 1894 and first used by the Pueblo S. & R. Co. Like its prototype, the device of Cordurie, it uses a propeller wheel for establishing a circulation and incorporating the zinc. The details of its construction, however, make it more efficient. It works by steam or electrical power and effects a thorough and uniform mixing of the metals.

For quite a time there was no material change in the manner of conducting zincings from that practiced in the early trials of the Parkes process in England. The general custom was to add the zinc in three doses, stirring in each addition, and then cooling and skimming until a ring of chilled lead had formed around the kettle. Only the first crust was liquated and the dry crust sent to the retorts, the second and third crusts were used again in the next charge. Some works used four zincings, the first to collect the gold along with a small part of the silver in a so-called gold crust. When this was done, the first and second crusts were liquated separately and the dry crusts were worked up each for itself.

With a wide range in the silver contents of the lead bullion received from various sources, it was desirable to have tables for the zinc additions. Lead desilvered by zinc, contains at its melting point 0.6 per cent of zinc. The total quantity of zinc required to extract the silver from any given quantity of lead bullion is reppresented by a formula containing a constant (0.6 per cent of the charge) plus a variable that increases with the silver contents, but

at a decreasing ratio.   On this basis, tables for zinc additions were worked out covering a wide range of silver content.

The time required for desilvering a charge was from 15 to 20 hours.   Attempts to reduce the time by reducing the number of zincings were successful, but at the expense of larger zinc additions.   That such was the case was due to the incomplete incorporation of the zinc.   With the use of the Howard stirrer, supplemented by the Howard press, the total time of desilvering has been reduced to not to exceed ten hours.   The zinc additions are not greater and there are no longer coolings to a "ring," until the very last, when the silver content is to be brought down to the assay required in the merchant lead.

Refining.   Eliminating the zinc retained by the desilvered lead has, in the United States, been effected either in a reverberatory by oxidation at a comparatively high heat, or in a kettle, by passing dry steam through the lead previously heated to redness, as was first practiced by Cordurié.   Steaming in kettles was used by some of the early unsuccessful works; it has been repeatedly tried experimentally, the Pueblo S. & R. Co. alone adopting it permanently for refining.   The use of the reverberatory has found most favor with American refiners, and at the present time it is exclusively employed.

The construction of the refining furnace is exactly the same as that of the one used in softening.   Its size has kept pace with that of the softener, so that while the charge of a large softener is divided among two or more desilvering kettles, the contents of these kettles are again gathered together for treatment, in one large refiner.

After the zincy oxides have been skimmed, the ordinary practice is to raise the heat for a short time and to admit air, in order to remove the last of the antimony.   At some works this final removal of the antimony is accomplished by steaming for a short time without raising the heat.   The same procedure has been employed in accelerating the elimination of the antimony in the softening operations.

Tapping the refined lead into a merchant kettle and ladling into molds were early replaced by more expeditious methods. In 1876, at the Pennsylvania Lead Co.'s works, this was done

by running the lead through a swinging inch and a half pipe into molds arranged in the segment of a circle, the flow being controlled by a cast iron cock.   With the introduction of the syphon, some works continued to use the merchant kettle and employed the syphon to run the lead into molds.

The next improvement was the molding machine, a moving endless belt on which the molds are mounted.   As each mold comes under the discharge pipe it is filled, is carried then forward, cooled and as it passes over the end sprocket wheel the pig of lead is dumped.   This machine, introduced at the National Works by the then superintendent, A. Oechsler, is especially adapted for handling large charges economically.

Distilling the Zinc Crust.   The early use of the black lead retort by Balbach has already been noted.   It was mounted in an inclined position in a coke crucible furnace, the neck of the retort projecting through the front wall, the zinc distilled off being condensed.   When the charge was finished, the supporting front wall was removed, the crucible seized by tongs, turned down and emptied.   A. Fabre du Faure, in 1870, swung the crucible furnace on trunions so that by tilting it the retort could be emptied.   This tilting furnace is to-day used by all the lead refiners in the United States, and has been largely adopted in Europe.

About 1865 the Parkes process was taken up on the continent of Europe.   It is interesting to note the various ways that were followed in treating the zinc crust.   At some works it was smelted in a blast furnace and the resulting lead cupelled; at others it was scorified on a lead bath.   It was also treated with steam to oxidize the zinc, the resulting oxides being scorified on a test or treated with acids.   In all cases the zinc was sacrificed and there was an appreciable silver loss.   These various methods are in striking contrast to the simplicity of the original Parkes method made practicable by the American modifications.

From a capacity of 250 lb. the retorts were gradually increased in size until a capacity of 1200 to 1400 lb. had been reached.   As the tilting furnace was a patented device, it was not used by all refiners during the life of the patent.   Various furnace constructions were devised, some using coke, others coal, and all using

the same kind of retort, but instead of emptying the retort by tilting, the contents were tapped or dipped out. With the expiration of the patent, these various constructions were replaced by the Fabre du Faure tilting furnace. The only change in the operation has been the substitution of fuel oil for coke. The advantage is not so much in reduced fuel costs but rather in the longer life of the retort, the more uniform and better results obtained, and with the fact that the work is easier for the men.

Cupelling Rich Lead. Although cupelling rich lead is not peculiar to the Parkes process, and together with the treatment of the various secondary products belongs rather to the general metallurgy of lead, yet as the large tonnage handled by American works using the Parkes process has brought about a characteristic development of this operation, a brief sketch of its development in this country will not be out of place.

In the United States, cupellation has always been effected in the English type of cupel furnace. Small at first, they have increased gradually from a test capacity of two thousand pounds or less, to one of six tons. The use of boneash for making the test soon gave way to other, cheaper and more satisfactory, materials. Varying with individual preference there has been used Portland cement, mixtures of Portland cement with ground firebrick, Portland cement with ground limestone, ground limestone with fireclay, and test frames have been lined with magnesite brick. At present, a mixture of crushed or ground limestone (preferably dolomite) with fireclay is generally used. To counteract the corrosion of the test by litharge, and the eating out of the litharge channels, various waterjacket devices have been introduced. A. Steitz at the St. Louis works, being the first to use water-cooling for this purpose.

The test of the English cupelling furnace, while it is removable, is stationary when put in position in the furnace. American practice early developed a tilting test, and both stationary and tilting tests are at present in use. The latter is mounted in such a manner that, by means of a differential block or by a hand-wheel, it can be tilted up or down and the flow of the litharge controlled. The prototype of this device is found in a silver-fining furnace in use in Sweden at Konigsberg in 1854. The

tilting test was especially convenient before the introduction of litharge water jackets, and even under present conditions it is preferable to the stationary test.

For a long time it was customary to cast the rich lead from the retorts into bars and gradually feed these bars into the cupelling furnace. A marked improvement was effected about 1901, when F. B. F. Rhodes at the National Works at South Chicago, poured the charge from the retort into a pot, raised the pot by means of a pneumatic lift and poured the contents into the test. This procedure has been followed with advantage at other works.

Thus the simple small English cupelling furnace has been developed into its American descendant, which, in its most complete form, has a tilting watercooled test holding six tons or more of rich lead, is capable of oxidizing seven tons of lead in twenty-four hours, and produces doré silver 998 fine. Depending on the method of parting to be used, whether by electrolysis or sulphuric acid, the contents are either ladled into thin slabs, or they are poured into molds by tilting the test.

Except in the case of the retorts, no reference has been made to the use of oil as fuel. In a general way it will suffice to say that where the price warrants, as at the Selby Works at San Francisco, Cal., it has satisfactorily taken the place of coal.

SAMPLING AND ASSAYING OF SILVER ORES, CONTAINING COBALT, NICKEL AND ARSENIC

James Otis Handy

Pittsburgh, Pa.

The ores found in the Cobalt, Ontario (Canada) Mining District are arsenides of cobalt, nickel and silver. A part of the silver exists in crude metallic form. These fragments are of various shapes and sizes. This renders the crushing and sampling operations difficult; each buyer, or his representative, has worked out his own methods. In assaying these ores also there has been a gradual evolution in method.

Success in hand sampling is based, of course, on the careful determination of the number of quartering operations to be made, and the size of the ore each time a division is made.

Mechanical sampling is used chiefly by those buyers who crush *all* of the ore in ball and tube mills and screen out practically all of the "metallics." These are weighed and then melted into bars which are weighed and assayed for fineness. The pulp containing finely divided silver in various forms is assayed separately.

Hand sampling is used by many buyers and by some sampling works. The practice in sampling by hand at the works of the Pennsylvania Smelting Company (Carnegie, Pa.) and in making control assays at the Pittsburgh Testing Laboratory, are the subjects of this paper.

Sampling

The ore arrives at the works in car-loads of approximately 30 tons. Low grade ore is in bulk. High grade ore and concentrates are in sacks or barrels.

The low grade ore may contain rock fragments as large as 4″ size, but they usually approximate 2″.

Moisture Sample

Low grade ore in bulk is sampled in the car while unloading. Four separate samples weighing approximately 6 pounds each

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are taken at intervals while unloading.   Drying and weighing and calculation follow.   Lump ore containing no fines is usually nearly free from moisture.   Fine ore shipped when frozen may contain 5% or more.

High grade ore is sampled for moisture when its containers are opened after weighing.   Several separate samples are selected from different parts of the pile after shoveling over thoroughly.

Weighing

Tared push cars hold the ore while being weighed.

Low Grade Ore

Crushing

If the ore is less than $1\frac{1}{2}''$ in size it passes directly to the sampling-floor, which is covered with smooth, well-fitted iron plates. If the ore is over $1\frac{1}{2}''$ it is put through a jaw crusher which reduces it to $1\frac{1}{2}''$ and smaller.

Division by Shoveling

The first division of the ore is made by shoveling two parts to the storage bin and one to the sampling floor.   The floor pile is then divided by shoveling one to bin and one to sample, which is now one sixth of the whole lot and should never be less than 6000 lbs.

Crushing and Quartering

The sample is passed through rolls, reducing it to $\frac{1}{2}''$ size and quartered to not less than 1700 lbs.   (From this point, coarse and fine ore are treated in the same way.)

The ore is crushed again till all passes the $\frac{1}{4}''$ screen, then turned and shoveled over at least twice.   It is then quartered down to not less than 300 lbs. which is crushed by rolls to pass a 10 mesh screen, then turned and mixed thoroughly and quartered to not less than 60 lbs.   This sample is ground in a mill to pass 20 mesh and is cut down by a Jones Sampler to about 4000 grams which is ground in a pebble mill till the ore (except metallics) all passes a 120 mesh sieve.   Each part of the final sample consists of "pulp" and "metallics" weighing together about 1000 grams per sample.

Four sets are made. If the total amount of "metallics" from the 4000 grams is not over 3 grams, it is jointly assayed by the assayer's representing buyer and seller. The "Metallics" are scorified with test lead and borax glass and the button is cupelled: The weight of the silver bead is then recorded on each sample envelope with the weights of moisture, pulp and "metallics." If the "metallics" exceed 3 grams they are divided into two or more parts, to accompany the pulp samples.

High Grade Ore

Under this head is included all ore containing over 500 ounces of silver per ton or ore of 300–500 ounces containing its metallics in coarse form and unevenly distributed. The method of sampling is subject to some variation depending on the character of the ore but the usual method follows.

The ore is all crushed to 1″ size and divided into halves, alternate shovelfuls being taken to the storage bin and to the rolls which are set to crush to $\frac{1}{2}$″.

The metallics on the screen are collected, weighed, melted and cast into bars. The bars are weighed, sampled and assayed. The ore passing $\frac{1}{2}$″ screen is made into two piles of equal weights by alternate shoveling, one man doing the work. Each pile constitutes a sample and is treated separately as follows:

The ore is coned three times in order to mix it thoroughly, then coned and quartered until the weight is about 2400 lbs. This amount is crushed by rolls to pass a $\frac{1}{4}$″ screen.

This product is coned three times to mix thoroughly, then quartered and coned until reduced in weight to from 400 to 600 pounds. The ore is then crushed in rolls to pass 1-10″ screen. Any metallics are ground in a drug mill to pass the 1-10″ screen. The sample is then cut down by use of a Jones Sampler to about 75 pounds. This is then ground to pass a 1-20″ screen and reduced by the Jones Sampler to about 5 pounds. The sample is ground fine in a pebble mill, screened on a 120 mesh screen, giving 120 mesh "metallics" and "pulp."

The other half of the ore (through $\frac{1}{2}$″) is sampled in the same manner as described above. Both sets of samples are assayed and results averaged. If the ore is very rich (over 1000 ounces

Ag. per ton) it is divided into three parts after passing through the $\frac{1}{2}''$ screen. Each part is then run down separately. Small amounts of metallics on $\frac{1}{4}''$ screen are ground up in a drug mill and returned to the sample. Larger amounts are melted into a bar and the bar assayed.

Method of Assaying

There have been at least three distinctly different methods in use for assaying these arsenical cobalt-silver ores.

(a) The Combination of Wet and Fire Assays.

(b) The Scorification Fire Assay.

(c) The Crucible Fire Assay.

The writer has had best success with method "(c)". The manipulations are fewer and simpler than in "(a)" and the loss of silver by volatilization has been found to be more perfectly controlled than in either of the other methods. We know, however, that other assayers use methods "(a)" and "(b)" with excellent results. We shall describe only our own practice.

The Crucible Fire Assay

Fluxes (a) Silver Ore Flux

        Soda Ash (Na_2CO_3) — 4 parts

        Pearl Ash (K_2CO_3) — 4 parts

        Borax Glass ($Na_2B_4O_7$) — 2 parts

  (b) Litharge

  (c) Borax Glass for Cover

Assay of Low Grade Ores

Fusions are made in a gas fired muffle furnace (Case type with fan blower attached). "20 Gram" "Denver" or "Battersea" crucibles are used. 6 assays are run at once with two empty crucibles in front.

Charge: Pulp   0.2 A. T.

       Silver Flux 30 grams

       Litharge   60 grams

       Flour   1 gram

       Borax Glass Cover 8 grams

Mix the charge by rolling on a rubber cloth and transfer to crucible.   Cover with the borax glass.   Place crucibles in muffle at medium red heat.   Hold for 20 minutes then close muffle door and raise temperature to bright yellow and hold for 30 minutes.   Pour into conical iron molds and when solidified, hammer free from slag.

Cupel on single rows across the muffle.   The temperature and draft control are the important factors.   The temperature is kept as low as possible and conditions must be freely oxidizing without excessive draft.   The temperature must be low enough at the finish so that litharge feathers form on the cupel around the button.   Large silver buttons after the "blick" are covered with hot cupels.   Small buttons are simply cooled slowly to avoid "sprouting."

"Metallics" are run separately by scorification, using test lead with borax glass and silica flux (equal parts borax glass and sand both ground fine).   Temperature control both in scorification and cupellation are vital.

"Silver bars" from nuggets or coarse "metallics" are sampled by drilling top and bottom.   The drillings are assayed for fineness.   300 milligrams are wrapped in pure sheet lead and cupelled with a pure silver proof beside it.   The weight of silver taken for the proof is 85 to 90% of the weight of the drillings of the sample (250 to 270 m. g.).   The percentage loss of silver suffered by the cupelled proof is added to the silver recovered by cupellation of the drillings.

Assay of High Grade Ores

Charge: Pulp   0.2 A. T.
        Silver Flux 30 grams
        Litharge   90 grams
        Flour   none (The ore is a reducer)

Borax Glass Cover 8 Grams

Fusion is made as in the case of low grade ores.   The larger quantities of "metallics" from high grade ores are best assayed by the Crucible process rather than by Scorification.

Charge: Metallics   0.5 gram
        High Grade Ore Flux 50 grams
        Flour   1 gram

CALCULATION OF RESULTS

Let x = total weight of sample (a) pulp: (b) metallics

" y = assay of pulp (ounces Ag. per ton of pulp)

" z = milligrams of Ag. in total weight of metallics.

Then $\dfrac{(a)}{x}$ = % pulp, and $\dfrac{(b)}{x}$ = % metallics.

" $\dfrac{x}{29.166}$ = A. T. (assay tons) of sample

" (1) y × % pulp = ounces Ag. per ton of ore (from pulp sample)

Then (2) $\dfrac{z}{\text{A. T. of sample}}$ = ounces Ag. per ton of ore (from metallics)

(1) plus (2) = ounces Ag. per ton of ore (total)

If there is a bar of silver made from coarse metallics or nuggets, it is usually paid for on the basis of its weight and assay value. If the total original assay value of the ore is to be calculated, it must include the Ag. in coarse metallics.

This involves the factors:—

% of coarse metallics or nuggets.

Weight of bar from nuggets.

% of screenings from nuggets.

The calculation is easily made.

AGREEMENT BETWEEN ASSAYERS

The following figures illustrate how well independent assayers agree as to the value of samples of these complex ores prepared by the methods described.

	Mark	P. T. L.	P. S. Co.
37396........:.....	*Seconds*		
	Pulp	74.25 ounces	73.53 oz. Ag. per ton
	Metallics	61.96 "	62.79 "
	Total.....	136.21 "	136.32 "
	Fines		
	Pulp	114.23 ounces	114.42 ounces
	Metallics	28.23 "	28.22 "
	Total.....	142.46 "	142.64 "
37691..........	Total.....	54.07 "	53.79 "
37252..........	*L.G.X.*		
	Pulp	1225.85	1225.00
	Metallics	1295.53	1293.95
	Total.....	2521.38	2518.95
37252..........	*H.G.*		
	Pulp	503.17	502.14
	Metallics	558.34	559.24
	Total.....	1061.51	1061.38

These agreements are exceptionally close and represent a high degree of skill in manipulation acquired by the operators, especially in the close control of cupellation losses.

The methods of sampling and assaying are almost entirely those worked out by the Pennsylvania Smelting Company. Their accuracy has been confirmed by repeated re-sampling and re-assaying.

CHEMISTRY OF THE REDUCTION PROCESSES IN USE AT ANACONDA, MONTANA

By Frederick Laist

Anaconda, Montana

The ores received at the Washoe Smelter come almost entirely from the mines in Butte and contain the following minerals:

Minerals	*Composition*
Chalcocite	Cu2S
Covellite	CuS
Chalcopyrite	Cu FeS_2 (Trace)
Bornite	Cu_3FeS_3
Enargite	$Cu_3As\ S_4$
Malachite	$Cu_2Co_4 + H_2O$ (Trace)
Pyrite	FeS_2
Sphalerite	ZnS
Rhodochrosite	$MnCo_3$ ⎫ Less than 1%
Rhodonite	$MnSiO_3$ ⎭

The country rock of Butte is of the granite family and is a quartz monzonite containing a little less silica than typical granite. It is made up of the following rock minerals in approximately the proportions indicated.

Minerals	*Composition*	*Per Cent*
Quartz	SiO_2	20
Andesine	$\frac{1}{2}NaAlSi_3O_8 - \frac{1}{2}CaAl_2Si_2O_8$	42
Orthoclase	$K\ Al\ Si_3O_8$	23
Hornblend	$\frac{1}{2}Ca\ (Mg\ Fe)_3\ Si_4O_{12}.\quad \frac{1}{2}(MgFe)_2\ (AlFe)_4$ ⎫ Si_2O_{12} ⎭	5.
Biotite	$(H\ K)_2(Mg\ Fe)_2Al_2Si_3O_{12}$	9.
Titanite	$Ca\ Ti\ SiO_5$	0.1
Apatite	$(Ca\ F)\ Ca_4(P\ O_4)_3$	0.05
Magnetite	$(FeO)\ (Fe_2O_3)$	0.15
Pyrite	FeS_2	0.02
		99.32

The main country rock (quartz-Monzonite) is cut by dykes of aplite, a granite having the following composition:—

Quartz	30%
Andesine	20
Orthoclase	45
Ferro Magnesian Minerals	3
Pyrite	0.1
	98.1%

Closely associated with the copper deposits are dykes and bodies of "Modoc" porphyry, technically known as quartz porphyry and having the following composition:

Quartz	24%
Andesine	40
Orthoclase	25
Ferro-magnesian minerals	6
Pyrite	0.4
	95.4%

The following is an analysis of a typical altered quartz-monzonite occurring in immediate contact with the copper bearing veins and was taken from Bulletin No. 228 U.S.G.S.

SiO_2	62.09%
Al_2O_3	15.49
Fe_2O_3 and FeO	8.52
MgO	.42
CaO	.20
Na_2O	.37
K_2O	4.34
H_2O at 110°C	1.20
H_2O above 110°C	3.01
TiO_2	.51
P_2O_5	.13
S	5.47
Cu	.25
Total	102.00%
Less Oxygen	2.05
	99.95%

How the various rock components may be combined is immaterial so far as the purposes of this paper are concerned. For instance, in the furnace, silica will combine with iron and lime to form a slag whether it occurs as quartz, andesine, orthoclase, or any other mineral.

It *does*, however, make a difference in the coal and coke consumption of the furnaces and in the grade of matte produced by the blast furnaces whether the silica charged is free or combined. The former is less and the latter higher when the silica is mostly present as quartz. This is, of course, to be expected and follows from thermo chemical considerations.

It would be beyond the scope of this paper to trace out all of the reactions entered into by all of the components of the ores and furthermore such an attempt would be largely guess work.

Following is a list of minerals and compounds the reactions of which are responsible for by far the larger part (say 95%) of the activity in the various furnaces and the writer intends to confine himself to discussing the reactions going on between them.

Chalcocite	Cu_2S	Sphalerite	ZnS
Covellite	$Cu\,S$	Haematite	Fe_2O_3
Bornite	Cu_3FeS_3	Iron oxide	FeO
Enargite	$Cu_3As\,S_4$	Limestone	$CaCO_3$
Pyrite	FeS_2	Silica	SiO_2
		Alumina	Al_2O_3

The departments of the reduction works which are of interest to us are as follows:

Concentrator   Roaster   Reverberatory   Blast Furnace
Converter

The chemistry of each of these will be taken up in the order named, excepting the concentrator, which has no chemistry. The function of the concentrator, it may be said in passing, is purely to wash out enough of the silicious gangue of the ore (with a minimum loss of copper, silver and gold) to leave a *concentrate* containing just the right proportions of silica and iron for the subsequent operation of smelting. The concentrate is, therefore,

merely an enriched ore and differs chemically from the ore in no respect, except that it contains a larger percentage of copper, iron and sulphur and a smaller percentage of silica, alumina, and other gangue constituents.

The products of the concentrator which will be referred to in the following pages are coarse and fine concentrates and slimes. The slimes consist of ore which has been too finely ground for successful concentration, together with the clay-like decomposition products of the rocks. This slime is caught in settling ponds, allowed to dry partly and smelted in the blast furnaces after being briquetted together with fine concentrates and fine ore.

Roasting Furnace Reactions

All concentrates which are finer than 5 mm. are known technically as fine concentrates and go into the McDougal roasting furnaces. This material makes up over 90% of the charge, the remainder being fine ore and fine limestone. The latter undergoes no change in the roasters, the temperature being insufficient to cause dissociation into CaO and CO_2.

The concentrates contain about 33% of sulphur and about 75% of this must be roasted out in order to yield a calcine suitable for reverberatory smelting. The calcine contains about 8% of sulphur. The concentrates (as well as fine ore) may be regarded as essentially a mixture of gangue and FeS_2, Cu_2S, CuS, $Cu_3Fe\ S_3$, and $Cu3AsS4$.

On the first hearth the concentrates are dried. On the second hearth the roast commences with the breaking down of some of the FeS_2 as follows:

(a) $7\ FeS_2 = Fe_7S_8 + 6S$

(b) $6S + 12\ 0 = 6\ SO_2$
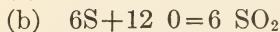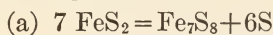
At this point and this temperature, the sulphides of copper are not materially affected. On the following hearths, the temperature is gradually increasing to a strong red heat and the various constituents are roasting according to the following reactions:—

$FeS + 3\ 0 = FeO + SO_2$
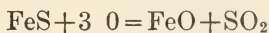
When too much air is admitted, the calcine becomes red, owing to oxidation of FeO thus:—

$2 \text{ FeO} + \text{O} = \text{Fe}_2\text{O}_3$
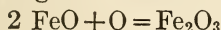
The copper sulphides undergo the following reactions:—

$\text{Cu}_2\text{S} + 4 \text{ O} = 2\text{CuO} + \text{SO}_2$

$\text{Cu S} + 3 \text{ O} = \text{CuO} + \text{SO}_2$

$\text{Cu}_3\text{FeS}_3 + 10 \text{ O} = 3\text{CuO} + \text{FeO} + 3\text{SO}_2$

The enargite breaks up as follows:—

$2 \text{ Cu}_3\text{As S}_4 + 25 \text{ O} = 6 \text{ CuO} + \text{As}_2\text{O}_3 + 8\text{SO}_2$

All of the arsenic, however, is not driven out as a portion unites with copper to form an arsenite or arsenate which does not decompose until it gets into the reverberatory furnace, where a portion of its arsenic goes into the slag and another portion into the matte.

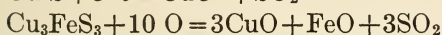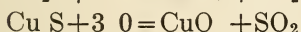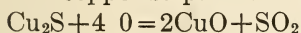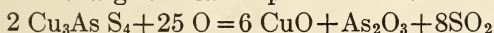In the reactions given so far the formation of sulphates has been ignored. This takes place as follows:—

$\text{FeS} + 4 \text{ O} = \text{FeSO}_4$
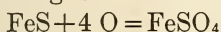
It is uncertain whether the oxygen combines directly with FeS to from FeSO_4 or whether the sulphate is formed indirectly as follows:

$\text{SO}_2 + \text{O}$ (in presence of Fe_2O_3) $= \text{SO}_3$

$\text{SO}_3 + \text{FeO} = \text{FeSO}_4$
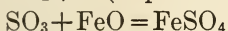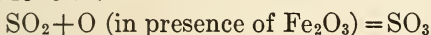
Fe_2O_3 is known to be a good catalytic agent; therefore, a part at least of the iron sulphate is formed in this manner.

Copper sulphate occurs in the calcine and is partly formed the same as iron sulphate and part on the hotter hearths by the decomposition of iron sulphate.

$\text{FeSO}_4 = \text{FeO} + \text{SO}_3$

$\text{CuO} + \text{SO}_3 = \text{CuSO}_4$
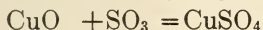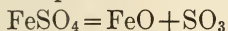
Iron sulphate is considerably less stable than copper sulphate. Therefore, if a mixture of FeSO_4 and CuO are heated together to a temperature intermediate between the decomposing temperatures of FeSO_4 and CuSO_4, the iron sulphate will break up and the liberated SO_3 will immediately unite with CuO.

REVERBERATORY FURNACE REACTIONS

A reverberatory furnace is essentially a melting furnace, the atmosphere of which is approximately neutral. Hence, the

material fed into it must have been oxidized to the desired degree in the roasters, i.e., sufficient iron sulphide must have been oxidized to iron oxide to form a suitable slag with the silica present.

The heat required for melting is generated by the combustion within the furnace of coal, wood, oil or gas. The active constituents of the first three named fuels are carbon and hydrogen combined as hydrocarbons. Fuel gas is a mixture of CO_2, CO, H, CH_4, (and other hydrocarbons) and N. The CO_2 and N are inert. CH_4 is formed by distillation from the coal in the gas producer. In the furnace it burns thus:

$CH_4 + 4\ O = CO_2 + 2H_2O$
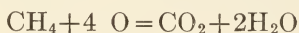
The CO and H are formed by the action of oxygen (air) and steam respectively on the incandescent carbon of the coal.

(a) $\ C + 2\ O = CO_2$
(b) $\ CO_2 + C$ (red hot) $= 2CO$
(c) $\ H_2O + C$ (red hot) $= 2H + CO$
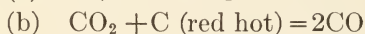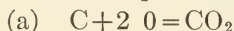
Heat is generated in the furnace by the re-combination of H and CO with oxygen.

(d) $\ 2H + O = H_2O$
(e) $\ CO + O = CO_2$
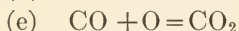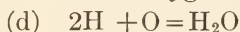
Reactions (a), (d) and (e) are exothermic. Reactions (b) and (c) are endothermic. Reaction (c) is often used to reduce the temperature in firebox or producer when the coal clinkers badly or to increase the length of flame in the furnace where the coal used is deficient in volatile hydrocarbons.

The charge to the furnaces may be considered to be a mixture of gangue, the principal constituent of which is SiO_2 with some Al_2O_3, and FeS, FeO, Fe_2O_3, Cu_2S, CuO, $CaCO_3$ with small quantities of $CuSO_4$ and $FeSO_4$. At the temperature prevailing in the furnace the limestone is first decomposed thus:

$CaCO_3 = CaO + CO_2$
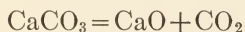
The iron and calcium oxides unite with the silica to form a complex silicate which constitutes about 86% of the slag. The simplest silicate reactions are:

$FeO\ + SiO_2 = FeSiO_3$
$CaO + SiO_2 = CaSiO_3$
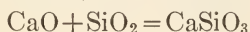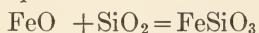
The slag which is being made at Anaconda corresponds very nearly to the formula (8FeO). 9SiO₂ and is a bi-silicate.

(CaO).
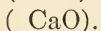
The particular type of slag to be made is a matter of expediency and of the action of the slag on the furnace lining. It is necessary, periodically, to fettle reverberatory furnaces, i.e., to throw silica against the fire bridge and side walls, along the slag line, for a distance of about 15 ft. back. This is necessary, owing to the tendency of the slag to cut, i.e., dissolve silica from the walls of the furnace where the temperature is highest. The particular type of silicate which will form is largely a matter of temperature. Hence, in order to reduce cutting to a minimum, we must proportion our charge to make a slag which is practically saturated with silica at the particular temperature in our furnace.

Of the remaining 14% of the slag, the principal constituent is Al_2O_3 (about 7%). Just how the alumina behaves, whether as an acid or as a base, has been a most fruitful subject of discussion among metallurgists. I believe the general opinion is that it is capable of filling both roles according to whether the surrounding medium is acid or basic in its character. Thus, in the presence of an excess of base, the alumina is likely to combine with it, forming aluminates, while in the presence of an excess of silica, it will play the part of base and we will have silicate of alumina in the slag. Both series of compounds, however, seem to be difficultly fusible and are viscous and gummy at the temperatures ordinarily attained in smelting furnaces and these characteristics are imparted to the slag. In the presence of much alumina it is not possible to run successfully as siliceous a slag as in its absence. Aluminates of calcium and of magnesium, of the formulae Al_2O_3, CaO and Al_2O_3, MgO have been made by heating mixtures of Al_2O_3 with CaO and MgO, respectively, in an electric furnace. Fire clay is a well known silicate of alumina.

The remaining 6% or 7% of the slag consists of MgO, MnO, Cu, S, alkalies, carbon, etc.

So much for the slag formation. The reactions which yield matte and influence its grade are dependent, first of all, upon

the power of sulphur combined with iron to reduce copper from combination as oxide or silicate, thus:

$$4FeS + 6CuO = 4FeO + 3Cu_2S + SO_2$$

The simple reaction:

$$FeS + CuO = CuS + FeO$$
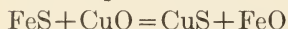
is impossible, owing to the instability of CuS at high temperatures, this compound breaking down into $Cu_2S + S$. Copper silicate, chrysocolla for example, is reduced similarly with formation of iron silicate:

$$4FeS + 6CuSiO_3 = 4FeSiO_3 + 3Cu_2S + SO_2$$

Thus sulphur acts as an excellent protective agent preventing loss of copper in the slag.

It has been said under roasting that Fe_2O_3 forms to a certain extent, particularly when too much air is admitted during the operation of roasting. Much has been said as to the harmful effect of this, due to the difficult fusibility of ferric silicate slags. It is the opinion of the writer that this effect cannot be very serious in the presence of sulphides since these would reduce ferric to ferrous oxides thus:

$$3Fe_2O_3 + FeS = 7FeO + SO_2$$
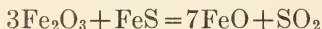
Practically all of the iron sulphide which is not decomposed by one of the last three reactions goes into the matte and it is for this reason that it is necessary to adjust the calcination so as to leave just enough sulphur to cause the formation of, say a 40% Cu. matte. Otherwise, the matte will be too low grade, due to the presence of too much FeS and, on the other hand, the slag be too siliceous owing to the absence of FeO.

Small quantities of sulphates present in the calcines break up in the fierce heat of the reverberatory.

$$CuSO_4 = CuO + SO_2 + O$$
$$FeSO_4 = FeO + SO_2 + O$$

Blast Furnace Reactions

The copper blast furnace, smelting sulphide ores, combines the functions of roasting and melting furnace, i.e., it oxidizes and melts simultaneously incidentally utilizing the heat generated by the oxidation of sulphur and iron for melting.

The ore, concentrates and briquettes charged to the blast furnaces we may consider a mixture of gangue, consisting principally of SiO_2 (with same Al_2O_3), and FeS_2, Cu_2S and $CaCO_3$. (Other compounds of sulphur and copper behave in an analogous manner to Cu_2S and oxides of copper or iron may be left out of consideration, owing to the small quantities of these which are received). The converter slag which constitutes about 12% of the charge is a silicate of iron containing copper, probably also present as silicate. It has the formula $2FeO \ SiO_2$. Coke is charged into the furnace to supply whatever heat cannot be obtained from oxidation of Fe and S.

In the upper part of the charge column, a certain amount of roasting goes on and FeS_2 and $CaCo_3$ breaks up with liberation of S and CO_2 respectively. The utilization of the oxygen blown through the tuyeres of a blast furnace is very perfect and very little free oxygen is found immediately over the top of the charge. This is shown by the clouds of yellow sulphur which make their appearance whenever a fresh charge is dropped on a hot top. There is not enough O present to burn the S set free from FeS_2.

The oxidation (bessemerizing) of the sulphides and coke as well as the union between oxides and silica to form slag take place mainly in the tuyere zone. The reactions are the same as those taking place in roasters and reverberatories, but proceed with much greater intensity and rapidity.

(a) $FeS + 30 = FeO + SO_2$

(b) $C + 20 = CO_2$
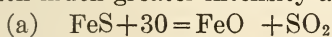
The Anaconda Blast Furnace slag corresponds closely to the formula:

$$\left. \begin{array}{l} (7FeO) \\ (9CaO) \end{array} \right\} .12 \ SiO_2$$
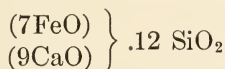
and is a sesqui-silicate.

From reactions (a) and (b), we can readily see that an excess of coke in the sulphide smelting furnace must have an injurious effect. The amount of oxygen in a blast furnace is limited and what there is will most readily combine with the carbon present before going to FeS.

Therefore, an excess of carbon (coke) reduces the amount of iron available for slag by preventing oxidation of FeS. Further-

more, it drives the latter into the matte and lowers its grade with respect to copper.

The converter slag, as has been said, corresponds to the formula 2 FeO SiO₂ and is a uni-silicate. At the temperature prevailing in the smelting zone of the blast furnace it is capable of combining with more silica to form a bi-or sesqui-silicate thus:

$$(FeO)_2 . SiO_2 + SiO_2 = 2\ FeO . SiO_2$$
$$2(FeO)_2 . SiO_2 + SiO_2 = (FeO)_4 . 3SiO_2$$
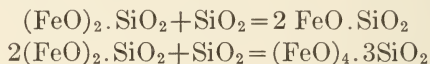
Any silicate of copper which may be in the converter slag or which may be in the blast furnace charge is reduced by FeS according to the reaction given under reverberatory smelting.

Converter Reactions

The charge to the converters consists of molten matte. Matte may be regarded as $Cu_2S . FeS$, although the matte received at the converters averages about 7% less in copper than is called for by this formula.

The linings of the present converters are magnesite brick and the silica required to flux the iron oxide formed by the action of the air blast on FeS must, therefore, be supplied by addition of ore. The reactions during the first half of the "blow" are as follows:—

$$FeS + 3\ O = FeO + SO_2$$
$$2\ FeO + SiO_2 = FeO_2 . SiO_2\ (slag)$$
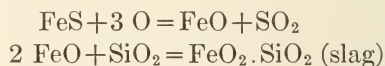
The slag is a uni-silicate of approximately the formula given.

It is important that the slag be poured out of the converter as soon as all the iron (all but about 1½%) is oxidized. Otherwise, the charge will foam out through the mouth. The exact reason for this phenomenon is not quite clear, but it is probably due to the formation of copper silicate which thickens the slag and prevents free passage of the air.*

$$CuO + SiO_2 = CuSiO_3$$
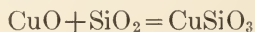
* It has been suggested that the foaming of the slag may be partly due to excessively rapid evolution of gases from the converter charge at about the time when the iron is all oxidized. That the elimination of sulphur is very rapid at about this period of the blow has been proved by several series of samples taken from the converter at ten minute intervals. An abnormally rapid evolution of gases in conjunction with a thickening of the slag would readily account for the foaming phenomenon.

The copper oxide, and therefore, silicate, cannot form in the presence of much FeS and for this reason, no trouble is encountered from this reaction till the iron sulphide is nearly all gone.

The slag having been removed, oxidation of the copper sulphide commences:

$$CuS + 4\ O = 2\ CuO + SO_2$$
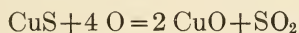
The copper oxide immediately reacts with sulphide:

$$Cu_2S + 2\ CuO = 4\ Cu + SO_2$$
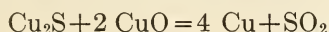
The progress of the various reactions is made manifest to the converter men by changes in the flame color. This is green while the iron is oxidizing, blue, just before removing the slag and red while the Cu_2S is blowing to copper. The end point of the charge can hardly be told from the color of the flame, however, and other indications are made use of, a discussion of which does not come within the scope of this paper.

The above are the main and essential reactions in the converter. The matte always contains impurities, the chief ones at this plant being lead, zinc and arsenic.

Experiments made here have shown that these are oxidized and eliminated as fume and in the slag throughout the slag forming period of the blow, and this elimination is practically complete in the case of lead and zinc, and nearly so in the case of arsenic. Very little elimination of arsenic takes place during the second, or copper forming period.

In closing, a word may be said as to the reactions in the refining and casting furnace. The copper is poured into this furnace in the molten state as it comes from the converter and it contains about 98.5% Cu. together with the silver and gold for which the copper has served as a collector, and fractional quantities of sulphur and iron. Here the copper is blown with air which oxidizes these and also some of the copper. The copper is now brought back to the proper "pitch" by poling which reduces the CuO. The impurities form a slag on the surface which is collected by means of cinders and skimmed off. It may be of interest to note, in passing, that the arsenic elimination in the

Anaconda casting furnaces is very light, this being almost entirely completed in the converters.

I append herewith a colored chart which shows graphically the elimination of the valueless constituents of the ore in the various operations. The widths and lengths, respectively, of the blocks are proportional to *copper* tonnages and *cupreous material* tonnages treated in the various plants. The blocks are subdivided according to the *quantities* of copper, iron sulphur and silica entering each plant. The *black* areas show what is eliminated in each plant.

<div align="center">SUMMARY</div>

Roasting Furnace Reactions

Charge consists of:

Gangue, FeS_2, Cu_2S, CuS, Cu_3FeS_3 and Cu_3AsS_4

(1)　(a)　　$7FeS_2 = Fe_7S_8 + 6S$
(1)　(b)　　$6S + 12\ O = 6SO_2$
(2)　　　　$FeS + 3\ O = FeO + SO_2$
(3)　　　　$2FeO + O = Fe_2O_3$
(4)　　　　$Cu_2S + 4\ O = 2CuO + SO_2$
(5)　　　　$Cu\ S + 3\ O = CuO + SO_2$
(6)　　　　$Cu_3FeS_3 + 10\ O = 3CuO + FeO + 3SO_2$
(7)　　　　$2Cu_3AsS_4 + 25O = 6CuO + As_2O_3 + 8SO_2$
(8)　　　　$FeS + 4O = FeSO_4$
(9)　(a)　　$SO_2 + O$ (in presence of Fe_2O_3) $= SO_3$
(9)　(b)　　$SO_3 + FeO = FeSO_4$
(10)　(a)　　$FeSO_4 = FeO + SO_3$
(10)　(b)　　$CuO + SO_3 = CuSO_4$

Reverberatory Furnace Reactions

Charge consists of:

SiO_2, Al_2O_3, FeS, FeO, Fe_2O_3, Cu_2S, CuO, $CaCo_3$, $FeSO_4$, and $CuSO_4$.

(11)　　　　$CaCO_3 = CaO + CO_2$
(12)　　　　$8FeO + CaO + 9SiO_2 = \dfrac{(8\ FeO)}{(CaO\ )}.9SiO_2$
(13)　　　　$4FeS + 6CuO = 4FeO + 3Cu_2S + SO_2$

(14) $4FeS + 6CuSiO_3 = 4FeSiO_3 + 3Cu_2S + SO_2$
(15) $3Fe_2O_3 + FeS = 7FeO + SO_2$
(16) $CuSO_4 = CuO + SO_2 + O$
(17) $FeSO_4 = CuO + SO_2 + O$

Blast Furnace Reactions

Charge consists of:

SiO_2, Al_2O_3, FeS_2, Cu_2S, $CaCO_3$ and C.

(18) $FeS + 3\ O = FeO + SO_2$
(19) $C + 2\ O = CO_2$
(20) $7FeO + 9CaO + 12SiO_2 = \dfrac{(7FeO\ )}{(9CaO)} \cdot 12SiO_2$
(21) $2(\ (FeO)2.SiO_2) + SiO_2 = (FeO)_4.3SiO_2$
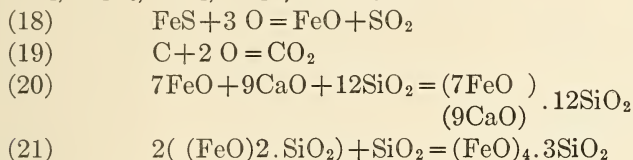
Converter Reactions

Charge consists of:

$Cu_2S.FeS$ and Siliceous ore.

(22)   (a) $FeS + 3\ O = FeO + SO_2$
(22)   (b) $2FeO + SiO_2 = (FeO)_2.SiO_2$
(23) $CuO + SiO_2 = CuSiO_3$
(24)   (a) $Cu_2S + 4\ O = 2Cu.O + SO_2$
(24)   (b) $Cu_2S + 2CuO = 4Cu. + SO_2$

CHART SHOWING THE ELIMINATION
OF THE IRON, SILICA AND SULPHUR
IN THE REDUCTION PROCESSES AT
ANACONDA

I   PLANT RECEIPTS
      FIRST AND SECOND CLASS ORES

II   CONCENTRATOR
      PRODUCTS—CONCENTRATES AND TAILINGS

III   BLAST FURNACE
      MATTE, SLAG AND SULPHUR DIOXIDE

IV   ROASTERS
      CALCINE AND SULPHUR DIOXIDE

V   REVERBERATORY
      MATTE, SLAG AND SULPHUR DIOXIDE

VI   CONVERTERS
      COPPER, SLAG AND SULPHUR DIOXIDE

VII   CASTING FURNACE
      COPPER

Cu —
Fe —
S —
SiO_2
ELIMINATED SUBSTANCES

NOTE.— THE HEIGHT OF EACH BLOCK IS PROPORTIONAL
TO THE PRIMARY CUPREOUS MATERIAL RECEIPTS.
NO ACCOUNT IS TAKEN OF SECONDARIES SUCH AS
CONVERTER SLAG OR FLUE DUST.

I   II   III   IV   V   VI   VII

IRON COKE

By Dr. A. P. Lidoff

Charkov, Russia

After studying the composition of dust from blast furnaces, I have come to the conclusion that by repeatedly passing it through magnetic machines it is possible to obtain a product the greater part of which is ferric-oxide.

If we mix the product thus obtained with gas coal slack, capable of giving a good coke (in proportion of one part of dust to two or three parts of coal), coke thus mixed at a high temperature, under pressure, there is obtained a porous coke containing metallic iron regularly distributed through it.

The specific gravity of this coke, and its strength and porosity are dependent on the quality of the coal and the composition of the dust.

The iron coke thus made by the utilization of two very cheap substances of by-products is useful in the mechanical industry in several ways, such as for the manufacture of electrodes and for the working of metals at high temperatures. It has also a great ability for decolorizing solutions, and is useful as a filter.

THE DEVELOPMENT OF THE REVERBERATORY FUR-
NACE FOR SMELTING COPPER ORES

E. P. Mathewson

Anaconda, Montana

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 the 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 19th century, was commonly 11ft. x 8 ft.

No material development was made until the Welsh process was brought to Colorado, where Mr. 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 Figure 1, elaborated by permission from Prof. 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″ x 15 ft. in hearth; and in 1894, 16 ft. x 35 ft. while the capacity of the furnace had been increased from 12 tons per 24 hours to 50 tons in the same period.

The next step in development was made in Butte, Montana, by the Colorado Smelting Company,—this plant being, at that time, affiliated with the Argo Works,—so that Mr. Pearce's influence was apparent. The step referred to, was the length-

ening of the hearth to 50 feet, with consequent increase in capacity to 105 tons in 24 hours.

The first furnace of this size,—built from the Colorado Smelting Company'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 Prof. H. O. Hofman's paper entitled "Notes on the Metallurgy of Copper of Montana," Transactions A. I. M. E., Vol. 34, pages 258 to 316.

The 50 ft. furnace became very popular in Butte and Anaconda, and when the Washoe Smelter was built in 1900–1902, the 50-ft-reverberatory was adopted and fourteen of these furnaces installed.

The next improvement was the addition of lime rock 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 increasing the capacity of the furnace greatly.

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 feet long was tried, with corresponding increase in tonnage. Then a radical step was taken at the Washoe plant, of connecting two 50-ft. furnaces, making a single furnace with a hearth 102 feet long. The saving in fuel, and the increased tonnage therefrom, caused the management to build a furnace with a hearth 116 feet long. The results of these experiments are given in the following tabulation, compiled by Mr. William Wraith, Superintendent of the Washoe smelter.

"Effect of Lengthening Reverberatory Furnaces at the Washoe Smelter, Anaconda, Mont."

The reverberatory furnaces as originally built at the Washoe Smelter, had a hearth-area of 19 feet in width x 5o ft. in length. After some months of operation it was decided that a longer furnace could be operated to a better advantage. To determine the length best suited to the conditions, one furnace was lengthened to 60 feet; another to 85 feet; another to 102 ft.; another to 112 feet; and, finally, to 116 feet in length.

The draft at the bridge wall was from 0.75″ to 1″ of water; the fire-box area 7 ft. x 16 ft.

The coal used was from the Anaconda Copper Mining Company's mines at Diamondville, Wyoming, having the following average proximate analysis and thermal values:

Analysis		Thermal Value	
		Wet Coal	Dry Coal
		11,710 B.T.U.	12,390 B.T.U.
H_2O	6.13%		
Volatile Matter	36.28%		
Fixed carbon	45.42%		
Ash	12.17%		

There was some variation in the quality of the coal,—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 pound of dry coal.

There is also some variation in material smelted and in manipulation by the different furnace crews.

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

The copper assays of the slags from the different furnaces, averaged as follows:

% Copper in Reverberatory Slags			
Furnace No. 1, with 50 ft. hearth		Sept. 1903–March 1904	0.50%Cu.
Furnace " 6	60 " "	March–Sept. 1904	0.44%
Furnace " 1	85 " "	May–Dec. 1904	0.42%
Furnace " "	102 " "	Jan. 1906–May, 1911	0.39%
Furnace " 4	112 " "	July, 1906–May, 1911	0.38%
Furnace " 1	116 " "	Jan. 1906–May, 1911	0.36%

DETAILS OF MODERN REVERBERATORY FURNACE SMELTING IN AMERICA.

	Anaconda (Montana)	Cananea (Mexico)	Great Falls (Montana)	Garfield (Utah)	Steptoe (McGill) (Nevada) (Coal) Fired	Steptoe (McGill) (Nevada) (Oil) Fired	Tooele (Utah)
Number of Furnaces	8	2	3	6	5		5
Average tonnage per day	253— (229.5 T.M.)	192.3 (174.4 T.M.)	200— (181.4 T.M.)	240— (217.7 T.M.)	239— (216.8 T.M.)	322— (292.1 T.M.)	225— (204.1 T.M.)
Average weight of Charge	15 tons (13.6 T.M.)	7½ to 9 tons (6.8–8.2 T.M.)	35 tons (31.75 T.M.)	6 to 24 tons (5.44–21.77 T.M.)	14 tons (var.) (12.7 T.M.)	14 tons (12.7 T.M.)	13 tons (11.8 T.M.)
Average tons charge per ton Fuel	4.25	5.86	2.00	3.50 to 4.00	3.24	5.80	3.80
Average temperature of charge entering furnace	950°F (Est) (510°C)	500° to 550°F (260°–288°C)	788°F (420°C)	400° to 700°F (204°–371°C)	500°F approx. (260°C)	500°F (260°C)	750°F (399°C)
Kind of Fuel used	Run of Mine Coal	California Crude Oil	Producer Gas	Bituminous Coal		California Crude Oil	Run of Mine Coal
Character of Charge: Calcine (Hot)	84.95%	50.0%	66.00%	75% to 78%	61.3%	56.8%	95.00%
Calcine (Cold)	......	......		19% to 16%	......	......	
Flue Dust	15.05	50.00	34.00	6.00	1.2	1.4	5.00
Conv. Slag	......	......		......	9.8	7.1	......
Secondaries (Cold)	......	......		......	12.5	15.2	......
Flux and Fettling Ores (Cold and Wet)	......	......		......	15.2	19.5	......
Dimensions of Hearth	111'-8" x 19'-0" (34.04 x 5.79m)	100' x 19' (30.48 x 5.79m)	41'-6" x 15'-9" (12.65 x 4.80 m) Gas Fired	112' x 19' (34.14 x 5.79m)	120'-10" x 19' (36.83 x 5.79 m)		102' x 19' (31.09 x 5.79 m)
Dimensions of Fire Box	7' x 16' (2.13 x 4.88m)	None	"	7' x 18' (2.13 x 5.49m)	None		7' x 16' (2.13 x 4.88 m)
Top of Grate Bars to top of Bridge	24" (61 cm)	None	"	33" (84 cm)	33–1.4" (84 cm)		40" (102 cm)
Top of Grate Bars to under side of Roof	70" (178 cm)	None	"	45" to 48" (114–122 cm)	85–3–8" (217 cm)		78" (198 cm)
Height of "Verb" or Vulcatory above Skim Plate	36" (91 cm)	36–1–2" (93 cm)	"	36" to 38" (91–97 cm)	36" (91 cm)		36" (91 cm)
Dimensions at Throat of Furnace	7' x 4'-11" (2.13 x 1.50m)	7' x 1'-9" (2.13 x .53 m)	"	7' x 3'-3" (2.13 x .99 m)	6'–10" x 3' (2.08 x .91 m)		7' x 4'-1" (2.13 x 1.24m)
Dimensions of flue beyond Throat (damper flue)	36" by 54" (91 x 137 cm) (natural draft)	7' x 6' (213 x 183 cm)	"	39" x 64" (99 x 163 cm)	(Not Given)		36" x 60" (91 x 152 cm)

Draft in inches of water at Bridge....	.75"-1" (1.9-2.54 cm)	(No Data)	.2" (.5 cm)	.6" (1.5 cm)	2"-.5" (.5 cm-1.3 cm)		.8" (2 cm)
Draft in inches of water at Throat....	1.3"-1.7" (3.8-4.8 cm)	.13" (.33 cm)	.3" (.76 cm)	.8" (2. cm)	.7"-1.2" (1.78-3.05 cm)	.9" (2.29 cm)	.6" (1.52 cm)
Draft in inches of water—Main Flue beyond Boilers....	1.5'-2" (3.81-5.08 cm)	.9" (2.29 cm) 115 sq.ft.	1.5" (3.81 cm)	1.65" (4.19 cm)	1.25" (3.18 cm)		1.75" (4.45 cm)
Dimensions of Main Flue....	15' x 20' (4.57 x 6.10 m)	10' x 12' (3.04x3.66m)	20' x 48' (6.1 x 14.63 m)	19'-8"x22'-6" (5.99x6.86m)	29'-3" x 8'-6" (8.91 x 2.59 m)		18' x 20' (5.49 x 6.1 m)
Dimensions of Chimney....	300' x 30' (91.44 x 9.14 m)	187' x 12'-6" (57. x 3.81 m)	506' x 50' (154.22 x 15.23 m)	300' x 30' (91.44 x 9.14 m)	300' x 15' (91.44 x 4.57 m)		350' x 25' (106.68 x 7.62 m)
Height of top of Stack above Grate....	740' (225.54 m)	198' (60.35 m)	768' (234.07 m)	504' (153.61 m)	316'-9-1/2" (96.55 m)		412' (125.57 m)
Temperature of Gas at Bridge....	2800°F (1538°C)	2700°-2800°F (1482°-1538°C)	2550°F (1455°C)	2600°-2800°F (1427°-1538°C)	2700°-3000°F (Est) (1482°-1649°C)		2600°F (1427°C)
Temperature of Gas at Throat....	2200°F (1205°C)	2300°-2400°F (1260°-1316°C)	2550°F (1397°C)	2000°F (1094°C)	2200°F (approx) (1205°C)		2000°F (1093°C)
Temperature of Gas at Far Side of Boilers....	680°F (360°C)	500°F (260°C)	(No Boilers)	900°F (482°C)	800°F (approx) (427°C)		775°F (412°C)
Average Boiler H. P. per Furnace from waste heat....	560 H.P. (568 C. V.)	971 H.P.* (984 C. V.)	None	330 H.P. (335 C. V.)	486 H.P. (493 C. V.)	632 H.P. (641 C. V.)	390 H.P. (396 C. V.)
Per cent Fuel recovered as steam....	32.81	56.62	None	27.00	32.3	33.8	41.2
Per cent Fuel recovered from ashes....	7.50	None	5.00	9.50	5.05		None
Ratio of Concentration....	4.79	4.96	4.30	3.50	3.31	3.40	9.00
Tons charge per million B. T. U....	0.182	0.1574	0.115	0.147-0.167	0.122	0.171	0.165

* Economizers are installed at this plant beyond boilers.

DETAILS OF MODERN REVERBERATORY FURNACE SMELTING IN AMERICA

Analyses of Materials Used and Produced

		Copper percent.	Silver Ozs. (Per Ton)	Gold Ozs. (Per Ton)	% SiO$_2$	% Fe & Mn.	% CaO	% S.	% Al$_2$O$_3$
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	26.80	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	trace	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*	0.08		46.50	20.58	17.50	1.00	5.50
	Steptoe (McGill) (Coal) fired	0.45*	0.06	0.005	40.20	28.90	10.50	app'x	7.60
	(Oil) fired	0.50*	0.02	0.007	42.00	25.90	10.90	0.40	8.00
	Tooele	0.43*	0.45	0.008	40.50	36.93	4.00	0.20	
Matte	Anaconda	41.68	28.40	0.130 (Insl)	0.27	26.54		25.76	
	Cananea	31.91	11.34	0.07	0.54	37.17		26.87	
	Great Falls	31.96	12.00	0.085	No data	35.54		26.70	

Matte

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	.91	0.22	0.50	29.10	24.80
Tooele	23.00	27.50	0.85	0.50	43.00	26.50

Fuel

	% Moisture	% V.C.M.	% Fixed Carbon	% Ash	B. T. U. (dry)	B. T. U. (wet)
Anaconda..........	6.13	36.28	45.42	12.17	12,390—	11,710—
Cananea...........		23.70	46.00	28.60	No data	18,600—
Great Falls........	7.70	39.40	46.40	6.20	No data	9,382—
Garfield...........	8.00	39.10	51.50	7.40	No data	12,000—
Steptoe (McGill) (Coal) fired	2.00				13,000—	......
(Oil) fired........					18,220—	......
Tooele.............	6.00	38.00	45.00	11.00		11,500—

		% CO₂	% CO	% CH₄	% H	% N
Gas—Great Falls....................		10.50	15.80	2.20	13.70	57.80

* Converter slag is poured into Reverberatories at these plants, thus raising copper assay, as only 75% of copper in Converter slag is recovered.

"Reverberatory furnaces in different localities present their own conditions, and the lengths of furnaces will be found to be functions of the type of fuel, draft, tonnage to be smelted and investment."

About this time, the Garfield plant of the American Smelters Securities Company, was being built near Salt Lake City, Utah, and for that plant furnaces with 100-ft. hearths were adopted. Then the plant at Humboldt, Arizona, of the Arizona Commercial Copper Company, built two furnaces of 100-ft. hearth, using oil for fuel. The Steptoe plant, at McGill, Nevada, followed with five furnaces with hearths 110 feet long, coal-fired, changed to oil-firing in 1911. The Cananea Copper Company put in one furnace with hearth 100 feet 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 Dr. Ricketts' article on experiments in reverberatory practice at Cananea, Transactions, Institute of Mining and Metallurgy, Vol. 19, page 147.

The following table gives important data on the best modern installations of reverberatory furnaces,—and the table of analyses gives the details of the chemical composition of the charge, the slags, the mattes, and the fuels.

The features that distinguish the modern reverberatory are its length,—solid bottom (monolith of quartz fused) structural steel conker plate for 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″ 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 unburnt coal from the ashes.

During the development of the reverberatory furnace to its present state of efficiency, a great many schemes were tried out 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 used, and is being used in reverberatory practice; for example, at Kyshtim, Perm Government, Russia, producer gas from wood, is in use, with very gratifying results. At this plant the hearth of the furnace is about 35 feet between ports and 15 feet wide; the gas was made from pine wood and the moisture and tars scrubbed from the gas before going to the furnace. The following are data regarding the furnaces:

Surface of air checker 869 square meters; 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,—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, CO_2 12.2%; O 6.3%; N 71.5%.

Pressure at reversing valve 3 m.m. water. Suction at Stack 25 m.m.

Temperature gas entering regenerator 68° C. Temperature entering stack 385°C.

Average temperature 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 you the composition of the charge, but 90% of it was flue-dust and green fines.

From January to September, 1910, the furnace was in operation 166 days with results as follows:

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

The fuel is chiefly interesting for the extremely low fuel cost, which has been verified by a first class engineer on the ground; wood at Kyshtin cost $1.05 per cord delivered, so that the average fuel cost over the 9 months period was 33.6 cents per ton of 2000 pounds of charge. It comes about from the fact that the local Russians, in charge of the furnaces have had a lot of experience with gas producers in iron works, and made a gas of very high calorific power.

Kyshtim Flue Dust and Ore Analyses

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

SiO_2	8.8%
Fe	50.9%
Cu.	3.3%
S.	9.6%

Sample from 6000 ton pile flue dust:

SiO_2	8.9%
Fe	48.9%
Cu.	3.3%
S.	10.0%

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

Average sample of raw ore used with flue dust:

SiO_2	2.1%
Fe	38.5%
CaO	0.5%
S.	47.0%
Cu.	3.2%

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 (arranged as shown in the accom-

(Footnote: data in regard to Kyshtim plant communicated by John H. Allen, Consulting Engineer 82 Beaver St. New York City)

panying Fig. 2) using air for atomizing the oil, has given the best results.

A record performance at McGill, is communicated by Superintendent Sorensen on December 17th, 1911,—No. 1 Furnace smelting 660 tons of total charge on an oil consumption of five-eighths of a barrel of oil per ton of charge.

Steptoe Plant at McGill, Nevada

Record Tonnage on December 17th, 1911

Record of Running of No. 1 Reverberatory Furnace and Analysis of Charge

Total charge per furnace day, tons..............	666
Oil fired per furnace day, bbl...................	421
Coal equivalent of oil fired, tons...............	124.0
Total Charge per bbl. of oil, tons..............	1.58
Oil, bbl. per ton of total charge................	.63
Equivalent gross coal as % of total charge.......	18.60

Components of Charge as % of Total Charge

Calcines..................................	60.1%
Seconds..................................	16.8%
Converter hot slag.........................	9.0 %
Fettling..................................	3.9 %
Lime Rock................................	9.6 %
Flue Dust................................	0.6 %

SLAG—*Assay and Analysis*

Cu......................................	.40%
SiO_2......................................	44.0 %
Fe.......................................	34.3 %
CaO.....................................	8.6 %
Al_2O_3......................................	7.4 %
Oxy. Ratio	2.72%
Grade of Matte, percent in Cu..............	40.4 %

DRAFT—*In inches H_2O*

Bridge........0.35
Throat.......0.83
Stack........1.25

Temperature of verb..........................	1910°F
Infusibility factor calcines....................	1.5
Burners, large Steptoe, high-pressure...........	7

At the Washoe plant, in Anaconda, coal is used for fuel and a record run of one furnace was the smelting of $402-\frac{1}{2}$ tons of charge in 24 hours with a fuel ratio of one ton of fuel for 6.45 tons of charge.

At Great Falls, Montana, the fuel is producer gas, but the producers are located too far away from the furnaces to get the best results; new furnaces are contemplated for 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 Company is experimenting with pulverized coal as fuel, but the furnaces have not been in operation long enough to give data of any great value. Supt. David H. Browne 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 unburnt fuel,— and the clogging of the flues with ashes. The Canadian Copper Company have a rather peculiar condition, in that their charge is basic, and that the addition of silica, in the form of coal-ash, is rather welcome than otherwise,—and 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 have been installed and the accompanying diagrams show the important variations of these installations.

Silica brick are 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.

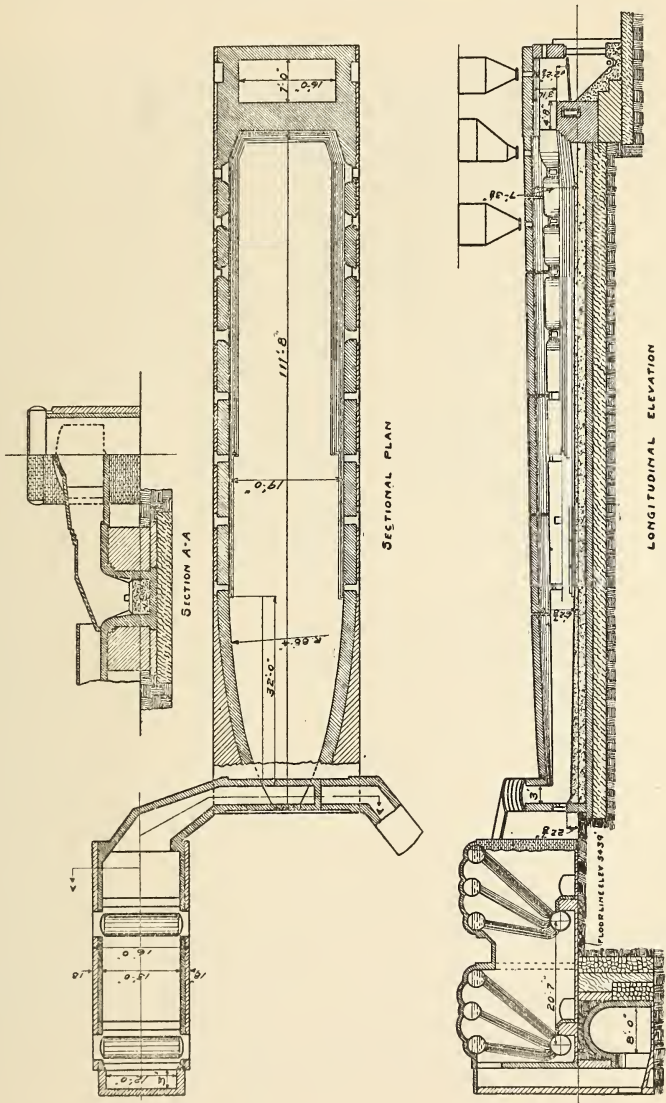I desire here to thank the following gentlemen for valuable data supplied:

Dr. E. D. Peters............ Dorchester, Mass.
Dr. H. O. Hofman.......... Mass. Inst. of Technology, Boston
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H. N. Thomson............ Tooele, Utah
John H. Allen............. 82 Beaver St., New York City
David H. Browne.......... Copper Cliff, Ontario

E. P. MATHEWSON

LIST OF ILLUSTRATIONS

ANACONDA , MONTANA
REVERBERATORY FURNACE

No. 2

SECTION A·A

SECTIONAL PLAN

LONGITUDINAL ELEVATION

N.º 3

Reverberatory Results
at Anaconda.

Tons Smelted per
Furnace per 24 hrs.

Tons Smelted per
Ton of Coal.

Percent Cu

0.50

0.45

0.40

0.35

Percent Copper
in Slag.

Tons of Cupreous Material per 24 hrs.

Tons Cupreous Material per Ton of Coal.

-Legend-
• =Tons of Charge per 24 hrs.
x =Fuel Ratio
△ =Percent Copper in Slag.

Length of Hearth in Feet.
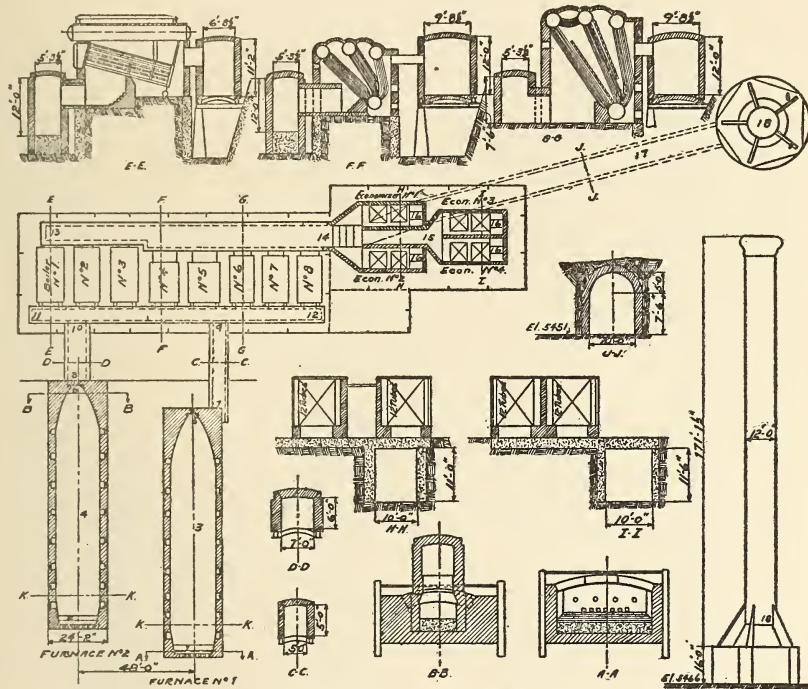
CANANEA CONSOLIDATED COPPER CO.

CANANEA-SONORA-MEXICO

ARRANGEMENT OF REVERBERATORIES, BOILERS AND ECONOMIZERS

GENERAL DATA.

2 Reverberatory Furnaces 100'x19' hearth-KK Area	69 ft	Boiler Nº4 Area Entering 38 sq.ft ~ Leaving - 20 sq.ft				
7-Burner holes	6"x10"	3 "	" Nº5	"	" 38 "	" "	20 " "
4 Peepholes	10" Dia.	2 "	" Nº6	"	" 30 "	" "	19 " "
Verbiter	3'-10⅝ wide 6'-2" long	24 "	" Nº7	"	" 30 "	" "	19 " "
Throat of furnace Nº1 5'-0" wide 2'-8" high		14 "	" Nº8	"	" 30 "	" "	19 " "
" " Nº2 7'-0" wide 1'-9" "		14 "	" Total	"	" 288 " "	"	151 " "
Furnaces No 1 & 2 throats		28 "	Flue from boilers at 13			Area	76 " "
Crossover flue Nº1 5'-0" wide 5'-0" high 50'-0" long		28 "	" " " 14			"	120 " "
" " " Nº2 7'-0" " 6'-0" " 27'-0" "		44 "	Economizer Nº1-Area Ent. 83 " ~ Leaving 45 " "				
Flue to boiler at 11		60 "	" Nº2 " 83 " "			"	45 " "
" " " 12		40 "	" Nº3 " 83 " "			"	45 " "
Boiler Nº1 Area entering 34 sq.ft ~ Leaving		18 "	" Nº4 " 83 " "			"	45 " "
" Nº2 " " 34 " "		18 "	Underground flue			Area 115 " "	
" Nº3 " " 34 " "		18 "	Stack 12'-6"~ 171-1⅙ high Base			"	123 " "
Boilers Nº1, 2, 3 ~ Aultmann & Taylor 250 H² Each			California crude oil used S.G. 0.966				
" Nº4 & 5 Stirling 250 H² "			Weight per barrel of 42 gal. = 339 lbs.				
" Nº6, 7, 8 300 H² "			Heat contents per pound = 18700 B.T.U.				
Economizers Green Economizer Co. ~ 288 Tubes							
70 Oil burners per furnace 0.43 lbs of steam rq'd to atomize one pound of oil.							
Total travel of gases in furnace Nº1 690'-0" Total travel of gases in furnace Nº2 - 730'-0"							
Smallest area for passing gases 14 sq.ft. Largest area for passing gases ~ 123 sq.ft.							
Temperature of gases at 9 and 10 = 2300°~2400° Fah. ~ Draft 0.25.							
" " " " " 14 - 500° Fah. ~ " 0.60.							
" " " " " 18 - 350° Fah. ~ " 0.90.							

Nº4

5

THE GARFIELD SMELTING CO.
GARFIELD UTAH.

GEN'L DW'G OF REVERB. FURNACE.

THE GARFIELD SMELTING CO., GARFIELD UTAH.

GEN DWG FLUES & BOILERS #6 REVERB. FURNACE.

HEARTH AREA 2125 □'
18½" × 120·10½

SECTION AA.

GROUND LINE

SECTION ON CENTER LINE.

REVERBERATORY FURNACE.

STEPTOE VALLEY SMELTING AND MINING CO.
McGILL NEVADA

INTERNATIONAL SMELTING AND REFINING PLANT,

General Drawing of 19'-0"×102'-0"
Reverberatory Furnace-

COPPER QUEEN CONSOLIDATED MINING COMPANY
DOUGLAS, ARIZONA.

Plan of Roaster & Reverberatory Furnaces with Connecting Flues, Dust Chamber and Stock

CANANEA CONSOLIDATED COPPER COMPANY
CANANEA—SONORA—MEXICO

ANACONDA COPPER MINING COMPANY
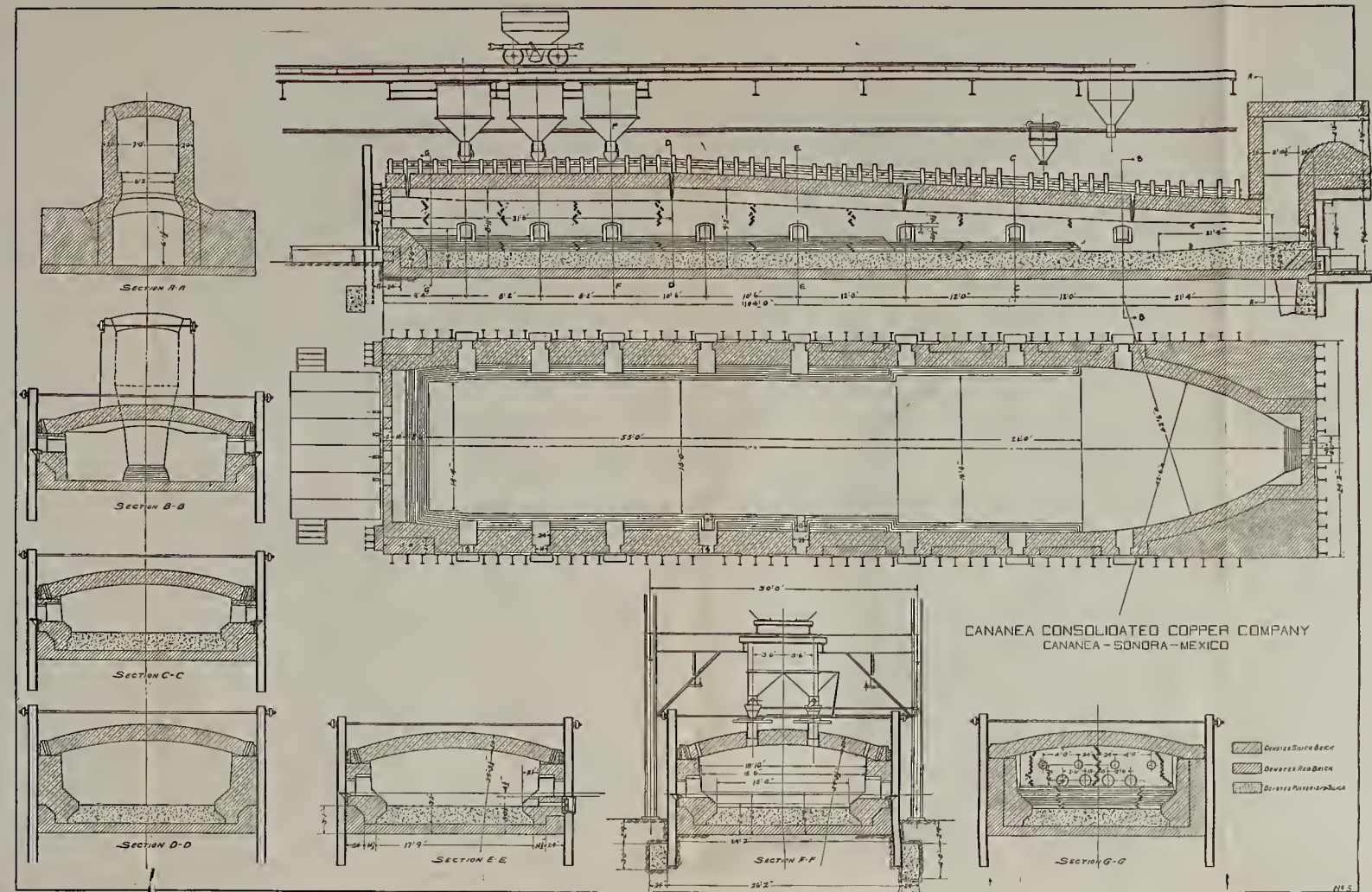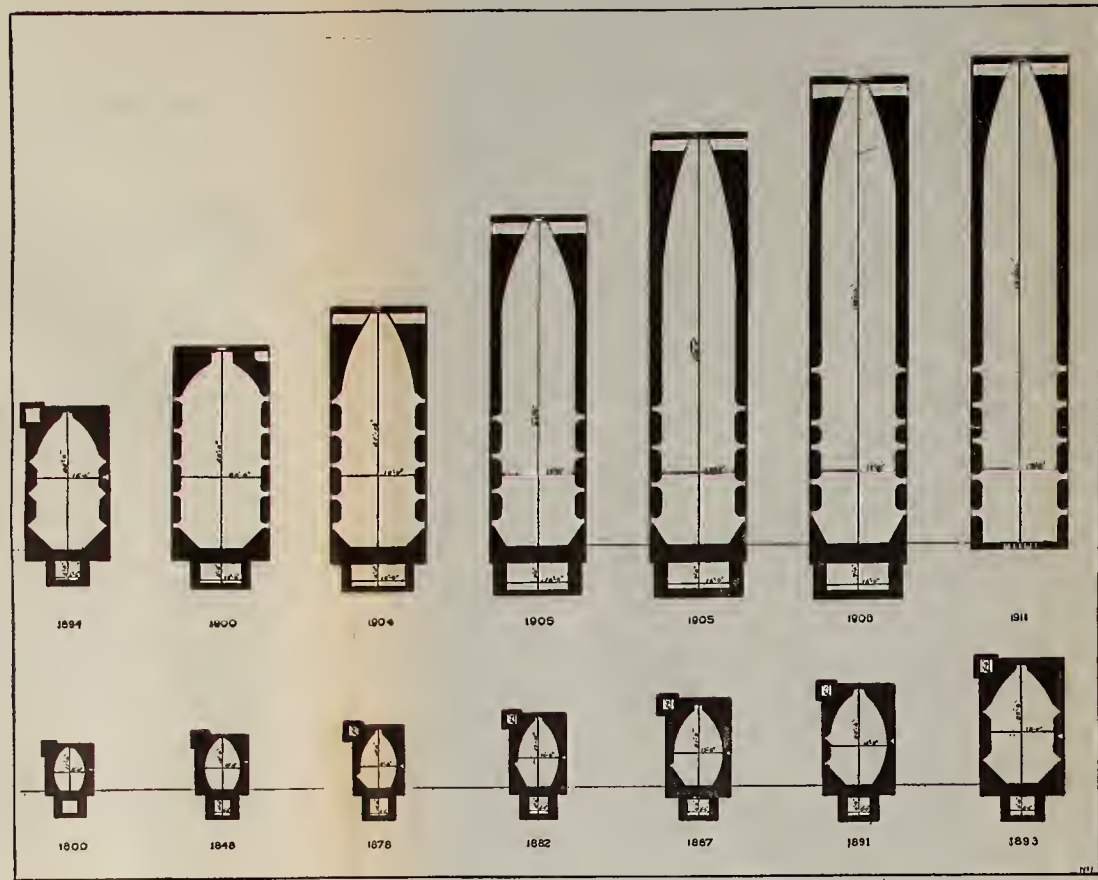GREAT FALLS MONTANA

ANACONDA COPPER MINING CO.
GREAT FALLS, MONTANA.

THE FUEL EFFICIENCY OF THE CUPOLA

By John Jermain Porter
Staunton, Va.

The chief purpose of this paper is to indicate the laws governing the fuel economy of the cupola, to examine the feasibility of some of the proposals for increasing its fuel economy and to show that the conditions limiting fuel economy here are altogether different from those prevailing in the blast furnace.

The two methods available for studying the thermal efficiency of the cupola and determining the factors limiting fuel economy are,—first, by the determination of the total heat balance; and second, by the application of the theory of critical temperatures.

The case selected as an example represents a cupola operating under fairly efficient conditions. The data are as follows: Cupola 60 inches inside diameter, 15 feet high to the charging door, with a 9 inch lining. Bed charge 2000 lbs. of coke and 4000 lbs. of iron. Subsequent charges 400 lbs. of coke and 4000 lbs. of iron. Total number of charges 20. 800 lbs. of coke recovered from the drop, hence total coke burned is 8800 lbs., or .11 lbs. of coke per lb. of iron. Coke contains 90% fixed carbon and 2% moisture. 300 lbs. of kindling wood are used in lighting. 80 lbs. of limestone (95% calcium carbonate) used per charge, or .02 lbs. per lb. of iron. Melting loss 4%, distributed thus: Iron 3.50%, Silicon 0.25%, Manganese 0.25%. Average analysis of top gases is carbon dioxide 15.1%, carbon monoxide 10.0%. Average temperature of top gases 1600° Fah. Temperature of air and stock charged 60° Fah. Dew point of air 50° Fah. The items entering into the total heat balance and their calculation are as follows:

1. Heat of combustion of fuel. Total heat evolved = 14,580 × lbs. of carbon burned + 7200 × lbs. of wood burned. Hence B. t. u. per lb. of iron charged = $\dfrac{8800 \times .9 \times 14,580 + 300 \times 7200}{80,000}$ = 1470.4.

2. Oxidation of iron to FeO.   B. t. u. per lb. of iron charged = .035 × 2112 = 74.0.

3. Oxidation of silicon to SiO_2.   B. t. u. per lb. of iron charged = .0025 × 12,600 = 31.5.

4. Oxidation of manganese to MmO.   B. t. u. per lb. of iron charged = .0025 × 2975 = 7.4.

5. Sensible heat in coke.   B. t. u. per lb. of iron charged = .11 × 60 × .16 = 1.1.

6. Sensible heat in iron.   B. t. u. per lb. of iron charged = 1 × 60 × .12 = 7.2.

7. Sensible heat in limestone.   B. t. u. per lb. of iron charged = .02 × 60 × .21 = .252.

8. Sensible heat in Blast.   From the gas analysis 9 lbs. of air are used per lb. of carbon burned, hence B. t. u. per lb. of iron charged = .11 × .9 × 9 × 60 × .235 = 12.6.

9. Heat of formation of slag.   This is a matter of some uncertainty, but is of minor importance.   The heat of formation of $CaO + SiO_2$ is 278 B. t. u. per lb., and of $FeO + SiO_2$ 121 B. t. u. per lb., and if we assume that the slag consists of equal parts of each and that 0.06 lbs. of slag are made per lb. of iron the heat of formation of the slag is in B. t. u. per lb. of iron charged 0.06 × 200 = 12.0.

1a. Heat in molten iron.   B. t. u. per lb. of iron charged = .96 × 450 = 432.0.

2a. Heat in molten slag.   B. t. u. per lb. of slag = 1 × [t × (.17 + .00004 t) + latent heat of fusion + (t′ − t) × .35], where t = the melting point of the slag or say 2000° Fah., and t′ = the temperature at which it issues from the cupola or say 2250° Fah.   Hence B. t. u. per lb. of iron charged = .06 (2000 × .25 + 160 + 250 × .35) = 44.8.

3a. Heat to decompose limestone.   B. t. u. per lb. iron charged = .02 × .95 × 813 = 15.4.

4a. Heat to evaporate moisture in coke.   B. t. u. per lb. iron charged = .11 × .02 × 966 = 2.1.

5a. Heat stored up in lining.   The weight of the lining below the charging door figures out to approximately 27,400 lbs.   Estimating its average temperature to be 1000° Fah. the B. t. u. per lb. of iron charged = $\dfrac{27400}{80000}$ × 1000 × (.193 + .000043 × 1000) = 80.8.

6a. Heat to decompose moisture of blast. A dew point of 50° Fah. corresponds to .0075 lbs. water per lb. of moist air. Hence the B. t. u. per lb. iron charged $= 9 \times .9 \times .11 \times .0075 \times 5800 = 38.8$.

7a. Heat sensible in gases. The weight of the gases per lb. of carbon burned works out as follows:

Carbon dioxide	2.200 lbs.
Carbon monoxide	.933
Nitrogen	6.910
Hydrogen	.007
	——
	10.050

The average specific heat is $.23 + .000023\,t$. Hence the B. t. u. per lb. iron charged $= .11 \times .9 \times 10.05 \times 1600 \times .2668 = 424.7$.

8a. Heat potential in gases. B. t. u. per lb. iron charged $= .11 \times .9 \times .933 \times 4370 = 403.7$.

9a. Heat lost by radiation plus error and unaccounted for is found by difference to be 174.2 B. t. u. per lb. iron charged.

Summarizing these items we get the following heat balance expressed in B. t. u. per lb. of iron charged:

Sources of Heat		Heat used and lost	
1. Combustion of fuel	1470.4	1a. In molten iron	432.0
2. Oxidation of iron	74.0	2a. In molten slag	44.8
3. Oxidation of silicon	31.5	3a. To decompose limestone	15.4
4. Oxidation of manganese	7.4	4a. To evaporate moisture	2.1
5. Sensible in coke	1.1	5a. To heat up lining	80.8
6. Sensible in iron	7.2	6a. To decompose moisture	38.8
7. Sensible in limestone	0.3	7a. Sensible in gases	424.7
8. Sensible in blast	12.6	8a. Potential in gases	403.7
9. Formation of slag	12.0	9a. Radiation and error ...	174.2
	——		——
	1616.5		1616.5

It is of course true that the limitations to fuel economy set by the total heat balance are not necessarily the only limitations. Johnson (Trans. Am. Inst. Mining Engineers, XXXVI, 454) has pointed out that in the case of the iron blast furnace fuel economy is limited by the heat available and necessary above a

"critical temperature," which is in this case the free running temperature of the slag, or about 2750° Fah. A similar limitation undoubtedly exists in many other processes and to test its applicability to the cupola the following calculations have been made.

In this case the critical temperature may be taken as 2250° Fah., and the heat available per lb. of carbon in the fusion zone and above this temperature works out as follows:

Combustion of carbon, 60% to CO_2, 40% to CO. 10500.0 B. t. u.
Heat brought in by blast at 60° Fah.   12.6
Heat brought into hearth by carbon at 2250° Fah.   905.0

                                                 ————
                                                 11417.6
Less heat to decompose moisture of blast   38.8
Less heat carried out of hearth by gases 6371.1   6409.9
                                                 ————
            Heat available   5007.7

The heat necessary per lb. of iron above the critical temperature works out as follows:

Heat to fuse iron, latent heat only   65.0 B. t. u.
Heat to fuse slag, latent heat only   9.6
Heat stored up in lining, lower part, 2-5 of
    total   32.3
Heat lost by radiation, 1-4 of total   43.5
                                                    ————
                                                    150.4
Less heat of oxidation of iron   74.0
Less heat of oxidation of silicon   31.5
Less heat of oxidation of manganese   7.4
Less heat of formation of slag   12.0   124.9
                                               ————
        Heat necessary per lb. iron   25.5

From this we get the somewhat surprising result that there is almost no externally supplied heat necessary in the melting zone, since the oxidation of the metalloids supplies almost all that is needed to melt the iron and slag. Even in case there were no

oxidation here the theoretical pounds of carbon needed per lb. of iron $= \dfrac{150.4}{5007.7} = .003$ lbs., which is so much below the actual as to show that the heat requirements of the melting zone do not impose any limitation to the fuel economy of the cupola and that the theory of critical temperatures is not adequate to explain its heat requirements.

Returning to a consideration of the total heat balance it is evident that the great source of wasted heat in the cupola is in the gases escaping at the top. If these losses could be eliminated it should be possible to charge some 22 lbs. of iron for each lb. of coke, have the gases come off from the top perfectly cold and containing no CO, and the iron satisfactorily melted. Actually this cannot be done, and it remains to be determined why this is so and how nearly perfection can be reached.

In the cupola there is a deep bed of carbon (coke) which is being replenished from above as fast as it is consumed. Under these conditions, with carbon always in excess, the products of combustion depend upon the temperature and time of contact of the gases with the excess carbon. According to the well known equilibrium diagram of Boudouard (Ann. de Chem. et de Phys. VII, 24) the tendency is towards the formation of CO at high temperatures and CO_2 at lower temperatures. Now in the cupola there is a zone immediately in front of the tuyeres which is cooled by the inrushing blast of cold air and in which CO_2 is formed. This formation of CO_2 being also aided by the fact that in this space oxygen is supplied faster than the surface of coke present can combine with it. Further in and up in the cupola the temperature is much higher and conditions are such as to favor the reduction of the CO_2 to CO according to the reaction $CO_2 + C = 2CO$. Time, however, is necessary for this reaction to take place, and since the velocity of the gases is very great and they are in contact with the hot carbon for only an instant, more or less CO_2 invariably passes through unchanged. On the other hand it is impossible to make the velocity of the gases so great as to entirely prevent reduction of CO_2 without creating intensely oxidizing conditions inside of the cupola and hence destroying its usefulness as a melting furnace.

The temperature of the top gases depends on the amount of heat absorbed by the stock in proportion to the total amount generated in the zone of combustion. More heat is generated when carbon is burned to CO_2 and the rapid rate of blowing necessary to the formation of a large percentage of CO_2 increases the velocity of the gases and gives less opportunity for the absorption of heat by the stock.

I have recently investigated the quantitative relationship between rate of blowing and temperature and composition of

FIG. 1
Time-Temperature Curves of Top Gases
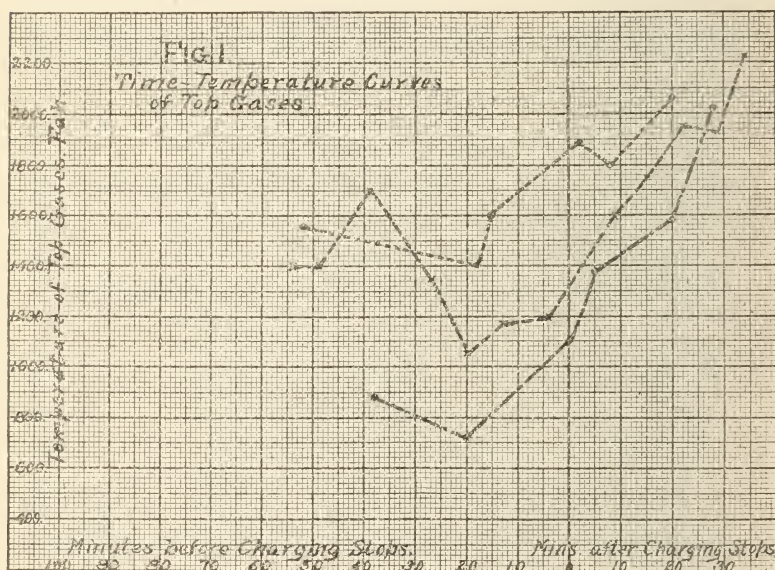
top gases in the operation of commercial cupolas, and the results of my experiments as far as they go are summarized in Figures 1 to 5.

Figure 1 shows some typical top gas temperature records during the progress of the heat. Each line represents a separate cupola. It will be noted that there is great irregularity in the temperature line up to the time charging ceases, this being due to the intermittant cooling by freshly charged material, the influence of temporary holes in the stock, etc. After charging ceases, however, it is invariably the case that the temperature rises rapidly

to a maximum approximately equal to the melting point of cast iron. This, of course, is due to the shortening column of stock and the absence of fresh additions of cold stock.

Figure 2. shows some typical top gas composition records during the progress of the heat. In this case there is even greater irregularity since the disturbing influences are greater. There is no well defined tendency towards either increase or decrease in CO, the effect of the higher temperature after charging stops being more or less neutralized by the shortening column of fuel.

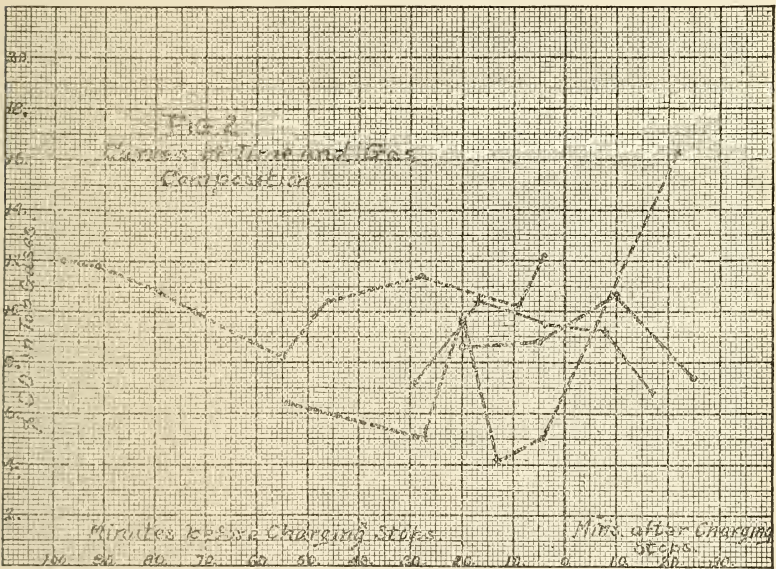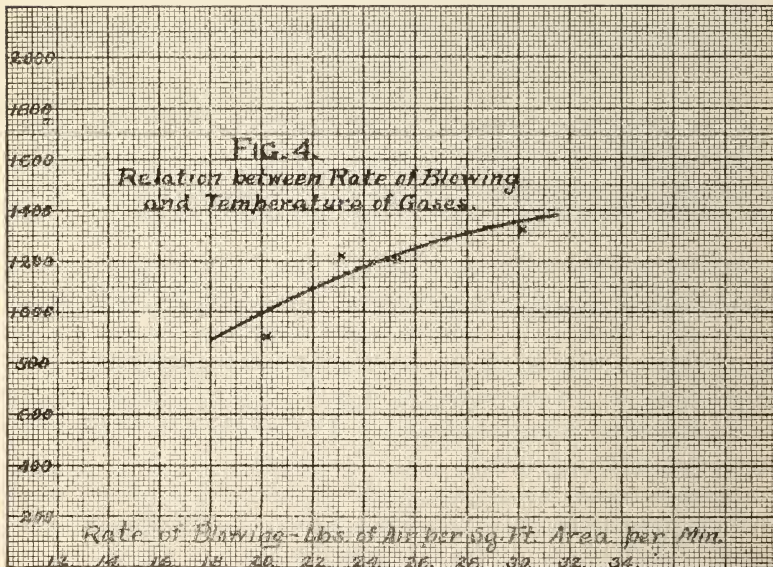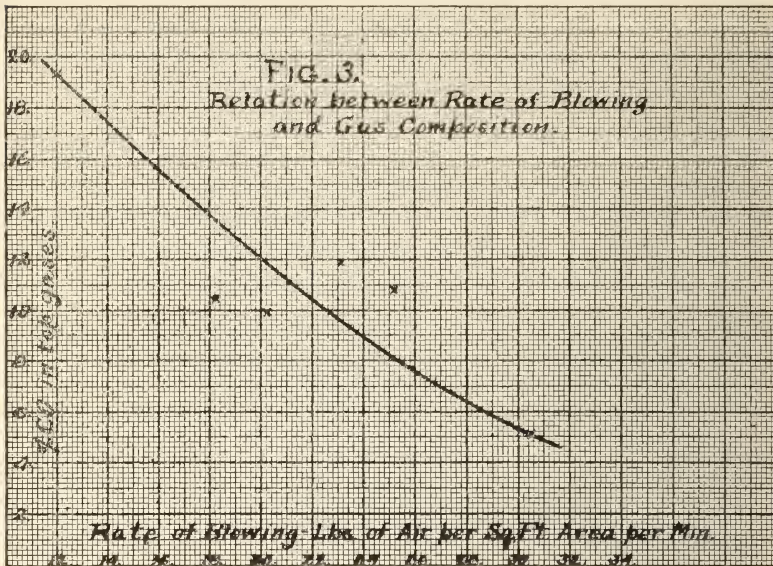

Fig. 2.
Curves of Time and Gas
Composition

Figure 3. shows the approximate relation between rate of blowing and average composition of top gas up to the time charging stops. Each of the 6 points from which this curve was plotted represents a separate cupola, and owing to the disturbing effect of different heights and different brands of coke it is natural that there should be some irregularity. The results as a whole, however, are fairly consistent.

Figure 4. shows the approximate relation between rate of blowing and average temperature of top gases up to the time charging stops. Each of the points represents a separate cupola, and as

FIG. 3.
Relation between Rate of Blowing
and Gas Composition.

CO_2 in the gases.

Rate of Blowing-Lbs. of Air per Sq.Ft. Area per Min.

FIG. 4.
Relation between Rate of Blowing
and Temperature of Gases.

Rate of Blowing-Lbs. of Air per Sq.Ft. Area per Min.

the height of the cupolas and other factors varied considerably it is natural that there should be some irregularity. On the whole, however, there is a marked increase in temperature with increased rate of blowing.

Figure 5. shows graphically the approximate relationship of heat loss, sensible and potential, to rate of blowing. The curves are calculated from the preceding data, which, however, are too scanty to give very reliable results. Incidentally it may be stated that from 25 to 30 pounds of air per square foot of cupola

FIG. 5.
Relation between Rates of Blowing and Heat Losses in the Cupola.
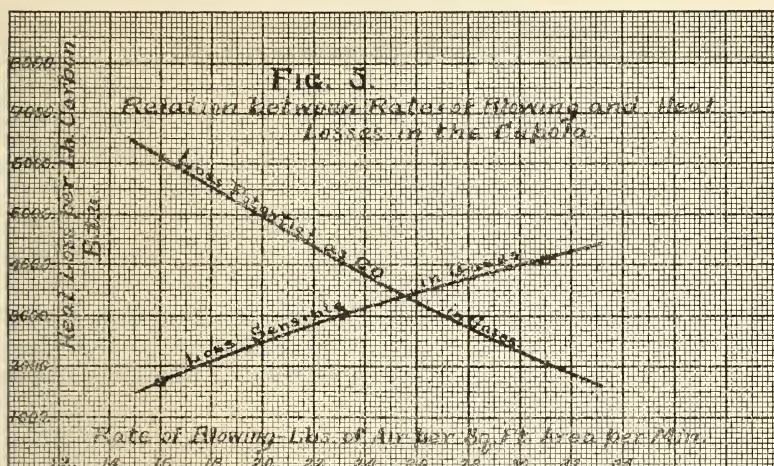
area per minute has been found by experience to be the most advantageous rate of blowing all things considered.

Within recent years there have been five principal proposals made with a view to fuel economy in cupola operation. These are, increased height of cupola, a second row of tuyeres, heated blast, and dry blast the Balliot heat recovery cupola.

The first of these, increased height of cupola, has been shown to rest upon a sound theoretical basis, while the results of practice indicate that it accomplishes the results desired. The quantitative relation between increase in height and fuel saving does not, however, seem to have been determined.

A second row of tuyeres is supposed to effect the more complete combustion of the carbon by the addition of oxygen at a point

where it can burn the CO. Theoretically this appears to be an excellent idea, but actually it usually fails to accomplish the result desired, i.e. to materially decrease the proportion of CO, although it may produce other desirable results.

Since CO is not formed directly at the tuyeres but only at some distance above or in front of them it is evident that the second row of tuyeres must be located a considerable distance above the lower ones in order to reach the CO. Now if this is done the result is to produce another zone of combustion and a practical duplication of conditions at the lower tuyeres. Where the addition of a second row has proven advantageous the benefit can usually be traced to the enlargement of the total tuyere area and the consequent greater volume of air admitted to the cupola.

In the blast furnace the factor limiting fuel economy is the heat available above a critical temperature of about 2750° Fah., which is only a small percentage of the total heat. The heat in the blast, while only a small item in the total heat balance is a direct addition to the available heat, which it may increase 100% or more. In the cupola on the other hand, the limiting factor is the total heat present, of which the hot blast can only be a very small percentage. For this reason the saving due to hot blast in blast furnace practice can not be taken as a criterion of what to expect in cupola practice.

In the case of the cupola previously considered heating the blast to 1000° Fah. would mean an addition of 2204 B. t. u. per lb. of carbon burned, which theoretically should reduce the carbon consumption to .083 lbs. per lb. of iron, or a saving of 16%. Hence the saving in cupola practice is approximately only one-third of that found in the blast furnace.

The use of an air blast artificially dried by refrigeration has proved of very great advantage in blast furnace practice from the standpoint of fuel economy and increased output. Reasoning by analogy some metallurgists have proposed to apply the process to the cupola, predicting equally good results here. The experiment has never been actually tried, but it can readily be shown that the saving would be very much smaller in the case of the cupola.

The value of dry blast in the blast furnace depends upon the great increase in heat available above the critical temperature

which it creates. As already explained this is not a limiting factor in the cupola, and the only fuel saving to be expected here is given by the ratio of the heat of decomposition of the moisture in the blast to the total heat generated per pound of carbon; or in the case used as an example $\frac{388.8}{10500} = .038$ lbs. carbon, or about 3.8%.

In the Balliot cupola[1] the main feature is the return of a portion of the gases of combustion to the wind box and thence into the zone of combustion again, the idea being similar to the Eldred process of flame tempering as applied to lime burning. The result of this procedure is to dilute the air entering the cupola with inert gases, to enlarge greatly the zone of combustion, to lower the temperature of this zone and probably to make it more uniform, and finally to greatly increase the volume of gases passing up through the cupola stack, though not to increase the total amount passing out of the cupola.

It is claimed that the net results of this procedure is to enable melting to be done with less fuel and less oxidation of the iron. Our theories do not seem adequate to test these claims satisfactorily, but they at least indicate their possibility. The effect of introducing inert gases with the air is exactly opposite to the effect of hot and dry blast as regards size and temperature of the combustion zone. If the cupola were similar to the blast furnace as regards dependence upon heat available above a critical temperature this process would necessarily cause a great loss in fuel economy. As we have seen, however, this is not the case.

On the other hand it is difficult to show theoretically where there is any gain in fuel efficiency. If there is such a gain it is probably to be attributed to more uniform working caused by the larger zone of combustion. It would seem that the claim of less oxidation due to lower melting temperature and more uniform distribution of the heat is at least plausible.

[1] See "Chemical Reactions in Foundry Cupola Practice" by J. DeClercy, Trans. Am. Foundrymen's Assoc. 1908, page 7.

"Distribution of Temperatures in the Cupola" by J. DeClercy, Trans. Am. Foundrymen's Assoc. 1908, page 103.

And discussion on cupola practice, same journal, page 154.

UN NUOVO PROCESSO DI PANIFICAZIONE DI STERRI E SABBIE DI ZOLFO A GANGA PRIVA DI CALCE ED ARGILLA

Di Prof. Angelo S. Skinzopoulos
Direttore delle zolfare di Milo, Grecia

Gli sterri o sabbie di zolfo non si possono sottomettere alla fusione in calcaroni o forni Gill che sotto la forma di panotti.

Mentre però i detti minerali contengono generalmente nella loro ganga gli elementi di calce ed argilla necessari per la lor panificazione diretta coll'acqua, come in Sicilia, vi sono sterri e sabbie di zolfo che, non contenendo tali elementi, non si possono panificare semplicemente coll'acqua. Ciò accade nelle nostre zolfare di Milo, dove il minerale—di natura volcanica—consiste in una quarzite più o meno impregnata di zolfo, secondo ch'essa è piu o meno disaggregata. Cosi la ganga del nostro minerale contiene.

SiO_2	96.943
$Fe_2O_3 + Al_2O_3$	2.300
CaO	0.660
MgO	0.036

Mescolare tali sterri con ganga od altro minerale, povero in zolfo ma ricco in calce ed argilla, sarebbe svantaggioso a cagione della materia inutile aggiuntavi, e qualche volta, come in Milo, impossibile. È dunque necessario di ridurre al minimo la quantità dei materiali estranei, utili alla panificazione, senza però diminuire la solidità dei panotti, necessaria per la loro fusione.

Di questa quistione si era già occupato il Vittorio di Chāteauneuf, che il 5 Marzo 1885 propose parecchi mezzi allo scopo di agglomerare gli sterri, come *"gesso cotto in polvere, cemento idraulico, calce ed anche sali."* Domandando la privativa, egli non entra però in nessuna particolarità tecnica intorno al pro-

cesso della panificazione, alla composizione del miscuglio ed al rolo mecanico o chimico dei mezzi da lui proposti.

Senza entrare in discussioni minute intorno al valore pratico di questi mezzi, io ho trovato che la calce sola si presta alle esigenze industriali della panificazione e della fusione dei panotti. Essa si può preparare sul luogo, o si compra dapertutto facilmente ed a buon mercato e lega bene gli sterri, senza rendere i panotti troppo compatti, ciò che sarebbe svantaggioso per la loro fusione.

Io arrivai all'uso della calce nel trattamento degli sterri, partendo dallo studio dei polisolfuri di calcio e della loro ulteriore ossidazione in solfato di calce. Difatti, la calce ha una marcata affinità per lo zolfo, con cui essa, sotto forma di latte, forma diversi composti come CaS_2—CaS_3—CaS_5. Mescolando gli sterri con calce spenta ed aggiungendovi l'acqua necessaria si osserva una elevazione di temperatura, ciò che indica la reazione chimica, verificata anche dalla riduzione della ricchezza del minerale in zolfo, la quale arriva a 2–3%.

La quantità di calce che si deve aggiungere al minerale non si può fissare d'avanti, dipendendo essa dalla richezza del minerale e dalla sua struttura molecolare, però è piccola. L'operazione si fà per partite di una tonnellata a mano, ma si potrebbe naturalmente impiegar anche apparecchi mecanici d'impastamento, trattandosi di una grande produzione. La malta si mescola bene con zappe, aggiungendovi l'acqua bastande per formare una politiglia oonsistenta ed omogenea.

La poltiglia cosi formata si mette poi con palle in forme di ferro galvanizzato troncoconiche, come le Siciliane, dove il materiale si pesta bene colle palle stesse per ben riempire le forme. Queste si vuotano poi rovesciandosi sul suolo, a preferenza sopra i forni Gill, dove si lasciano una settimana o più, secondo la stagione, fino ad acquistare la solidità necessaria per il loro caricamento nei forni.

Alla panificazione non si procede che da Maggio ad Ottobre, siccome, durante l'inverno, la dissecazione dei panotti e troppo lunga, anche sotto tettoje. Cosi i nostri forni si caricano per sei mesi con panotti e gli altri sei mesi con minerale, come del resto si fà in Sicilia.

Seccati i panotti, pesano otto chilogrammi all'incirca e sono molto resistenti. Si può mettere sopra un panotto un peso fino a due cento trenta chilogrammi senza ch'esso sia rotto. Caricati in quantità di due mila ottocento per ogni cella Gill, mantengono la loro integrità sino a fusione completa e dànno tutto il loro zolfo. Anzi la loro solidità e tanto grande, che i canali stessi delle celle non si fanno con minerale ma coi panotti stessi. La durata della fusione è di tre giorni all'incirca. Quando poi si procede allo searico del forno, pocchi panotti restano intatti, la maggior parte di loro disaggregandosi.

Non si può spiegare la solidità dei mie panotti che ammettendo l'ossidazione, dei polisolfuri di calcio contenutivi, avanti ed anche durante la fusione, in solfato di calcio o gesso, che forma così una rete foltissima di cemento, che avvolge e lega bene gli sterri da tutte le parti. Una combinazione della calce colla silice della ganga si deve escludere. Il riscaldamento spontaneo del miscuglio e la diminuzione della ricchezza del minerale indica chiaramente la reazione fra calce e zolfo.

Non è che con questo processo che, dopo parecchi anni di abandonno, fu ripreso, l'esercizio delle zolfare di Milo. La maggior parte del minerale estirpato è friabile e dà fino a 45% di minuto, il quale, per mancanza di calce ed argilla, sarebbe inatto alla panificazione coll'acqua sola e resterebbe cosi ribelle alla fusione.

THE SULPHATIZING ROASTING OF COPPER ORES
AND CONCENTRATES

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In general the art of securing copper from sulphide ores or concentrates may be said to consist of

Separation, in the molten state, of copper sulphide with some iron sulphide, from the great bulk of gangue material followed by

Elimination of sulphur and iron from the molten matte, by oxidation and fluxing, leaving metallic copper.

The definition is made broad enough to cover the general practice, whether or not the ore or concentrate is given a preliminary roast for the removal of surplus sulphur.

The above process, which we will designate as the smelting process, makes necessary the presence in the ore, or the addition thereto, of such silica, lime and iron as may be necessary for the formation of a light and fusible slag, together with sufficient sulphur and iron for the formation of copper matte, and also the addition to the matte of sufficient silica for the oxidation and fluxing out of the iron.

These items together with the cost of fuel sufficient for melting the mass constitute important items of cost in the copper smelting process.

Numerous cases where fluxing materials are scarce or dear, or where fuel is expensive, have caused special attention to be given recently to the wet process of copper extraction, which may be designated as consisting of forming soluble salts of copper which are then washed or leached out of the ore or concentrate by dissolving in water or acid. Metallic copper may then be secured from the solution of copper salts by electrolysis; by cementation; or, copper can be brought out of the solution as a hydroxide or other insoluble precipitate and reduced to metallic copper by subsequent melting in a reducing atmosphere.

This paper presents facts relating to only one step in the wet process of copper extraction and deals only with the formation of sulphate of copper by roasting copper ores or concentrates.

One percent of copper contents requires, theoretically, one-half percent of sulphur to form sulphate of copper. In practice, it is necessary to have present an excess of sulphur over this theoretical amount. If insufficient sulphur is present in the material to be treated, it is then necessary to add sulphur in some form, either as sulphur, SO_2, SO_3, or, in the form of a sulphide or sulphate or sulphuric acid. Iron sulphides are very desirable in ores, when it is desired to sulphatize the copper values, and zinc, lead, lime and magnesia, if present in forms that will consume sulphur, are undesirable.

The principal reactions which enter into the sulphatizing of copper sulphides, are as follows:

CuS decomposes in roasting to $Cu_2S + S$ and the free atom of S burns to SO_2 at temperatures above 350° C. (662° F.). Cu_2S at furnace temperatures below 500° C. (932° F.), reacts principally as follows:

$$2\,Cu_2S + 5\,O_2 = 2\,CuO + 2\,CuSO_4$$
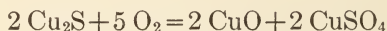
At higher temperatures, the principal reaction is

$$Cu_2S + 2\,O_2 = 2\,CuO + SO_2$$
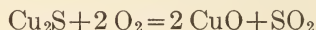
Different copper sulphide minerals behave differently at furnace temperatures, but the above formulae show the tendency common to all copper sulphides to form some quantity of copper oxide at all furnace temperatures.

In roasting to secure the highest percent of water-soluble copper, the constant formation of the copper oxide is the chief obstacle to be overcome.

The presence of iron sulphide with copper sulphide in the roasting process assists in sulphatizing the copper oxide thus formed. At furnace temperatures below 600° C. (1112° F.) there is some basic sulphate of iron formed, which, at temperatures above 530° C. (986° F.), roasts to Fe_2O_3 and SO_3 and the SO_3 gas may combine with the CuO to form $CuSO_4$ which is practically sta-

ble at temperatures below 650° C. (1202° F.), or, the CuO may react directly with the $Fe_2(SO_4)_3$, according to the formula,

$$3\,CuO + Fe_2(SO_4)_3 = Fe_2O_3 + 3\,CuSO_4$$
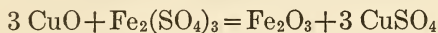
Even the portion of iron sulphide which roasts to Fe_2O_3 is of some assistance in sulphatizing the CuO, as at temperatures between 500° C. and 750° C. the iron oxide acts as an energetic catalyzing agent for the oxidizing of the SO_2 to SO_3, which in turn tends to combine with the CuO to form $CuSO_4$.

Also, SO_2 generated in the furnace, either from the copper or iron sulphides, tends to sulphatize the copper oxide.

These various principles are generally familiar from enunciation by Aubell, Kothny, Ottokar Hoffman and others, and attention is called to them here to make clear the significance of the following tests made by The Wedge Mechanical Furnace Company of Philadelphia.

The following tests were made without the addition of any material to the ore treated:

Test No. 621   *February 1910*
Analysis of Ore

Cu...............................	1.07%
Fe...............................	49.25%
S................................	32.60%
SiO	6.80%
MgO	2.18%
Al_2O_3........................	1.59%

The copper present was chalcopyrite; iron present was pyrite and pyrrhotite.

Ore was roasted in a Wedge multiple hearth furnace with the following results:

Water-soluble copper	0.35% = 27.8% extraction	
Acid-soluble "	0.80% = 63.4%	"
Insoluble	0.11%	
Total	1.26% = 91.2% extraction.	

The calcine from the above roast contained 4.7% sulphur and was then furnaced in a Wedge multiple hearth mechanical muffle furnace.   It was then leached with the following result:

Water-soluble copper	0.76% = 60.3%	extraction
Acid-soluble "	0.48% = 38.1%	"
Insoluble "	0.02%	

Total copper secured in solution   98.4%

In the above test, the copper designated as acid-soluble copper, was copper oxide, soluble in very dilute sulphuric acid.

It will be observed that in spite of the presence of liberal quantities of iron sulphide, there was considerable formation of copper oxide in the furnacing, but this occurred chiefly in the preliminary roast where, by reason of the considerable excess sulphur, temperatures became excessive.   Analysis of the gas leaving the furnace in initial roasting showed formation of sufficient SO_3 for the production of sulphuric acid to recover all of the copper oxide, or, if electrolysis were used in subsequent recovery of the copper from solution, more than sufficient sulphuric acid would be recovered to sulphatize the copper oxide.

Test No. 620   *March 1910*

Analysis of Ore

Cu.................................	3.52%
Fe.................................	9.76%
S.................................	9.54%
Insol...............................	66.5 %

Copper present as chalcopyrite; gangue consisted of schist.

This ore was given a single treatment in a Wedge multiple hearth mechanical muffle furnace 9' 9'' in diameter with five hearths; four hearths muffle-fired with coal.

The analyses made currently during the test showed:

Water-soluble copper from	60% to 70%	of total copper
Acid-soluble " "	20% to 22% " "	"
Total "	80% to 92% " "	"

In this test the SO_2 and SO_3 gases generated in the upper hearths of the furnace were drawn downward over the ore on the lower hearths of the furnace.

(Patent No. 966,277—August 2nd, 1910)

If electrolysis were used, the sulphate of copper solution would yield, in this case, a great excess of sulphuric acid for the recovery of the copper oxide.

Test No. A-100		*July 1911*
Analysis of Ore		
Cu		4.38%
Fe		9.61%
S		16.37%
CaO		1.10%
MgO		0.72%
Insol		61.21%

Copper present as chalcopyrite; the gangue was diabase.

This ore was given a single treatment in a Wedge multiple hearth muffle furnace (Patent No. 654,335—July 24th, 1900.) having a diameter of 9′ 9″ with five hearths, being the same furnace as was used in the previous test.

Actual leaching of this ore showed a recovery by

Water extraction	73.7% of copper in ore
Weak sulphuric acid	15.7% " " " "
Making a total of	89.4% of the total copper in ore.

It is believed that this ore, sulphatized in a larger furnace of the Wedge type, would yield still better results.

Tests so far given, were made with ores containing copper as chalcopyrite.

The following tests were made with an ore in which the copper was present chiefly as Bornite with some silicate and carbonate of copper.

Sample No. 125   *April 1911*
Analysis of Ore
Cu.................................   3.40%
Fe.................................   11.05%
S..................................   13.02%
SiO₂ 63.90%

This ore was ground to 20-mesh and was given a single treatment in a small furnace test and showed:

Water-soluble copper 80.80% of total copper contents
Acid-soluble   "   11.8%   "   "   "   "
                          _____
Total recovery   92.6% "   "   "   "

The above results have since been duplicated with this ore in a Wedge mechanical muffle furnace 20' 0'' in diameter with five hearths and sufficient sulphuric acid is recovered from the SO₃ in gases leaving the furnace to recover the oxide of copper formed and make the extraction as above shown in the subsequent leaching operation.

Sample No. 7   *December 1909*
Analysis of Ore
Cu.................................   1.85%
Ni.................................   2.75%
Fe.................................   47.78%
S..................................   30.85%
SiO₂ 15.30%

It was desired to extract the copper, leaving the nickel in the leached ore, together with the iron.

The analysis of the calcine showed:

Water-soluble   Copper 69.2%   Nickel 13.6%
Acid-soluble   "   12.8%   "   4.4%
                      _____   _____
Total soluble   "   82.0%   "   18.0%

The leached ore contained

    Copper............................... 0.29%

    Nickel.............................. 2.80%

It has been shown in tests made by The Wedge Mechanical Furnace Company that copper ores containing little or no sulphur or iron can be treated by sulphatizing roasting by the addition of sulphide of iron, in the form of iron pyrites.

Test No. 604   *August 1909*

Analysis of Ore

    Cu................................. 3.14%

    Fe................................. 2.25%

    S.................................. none

    Insol.............................. 85.5%

This ore contained copper as carbonate and the gangue was porphyry.

This ore was mixed with 5% of iron pyrites containing 47.5% sulphur.

The test was made in a 9' 9'' Wedge mechanical muffle furnace with five hearths and the analysis of the calcine showed 92% of the copper to be soluble in water. The copper solution was very pure, containing only from 0.1% to 0.2% Fe_2O_3 to 7% copper sulphate. Copper precipitated from the solution with clean scrap iron tested 96% pure cement copper.

Test No. 634   *November 1910*

Analysis of Ore

    Cu................................. 1.86%

    Fe................................. 2.79%

    S.................................. 0.80%

    Al_2O_3.......................... 2.58%

    Insol.............................. 88.3 %

    Combined water..................... 3.4 %

The sulphur contained in this ore seemed to be partly in cinders from the locomotive, which accumulated in the open car during transit from the west. Copper was present largely as silicate of copper. 6% pyrites containing 47% sulphur was

added to this ore and this mixture was given a single treatment in a Wedge Mechanical Furnace, muffle-fired with coal; diameter of furnace 9′ 9″ with five hearths and a preheating hearth. Tests were made of the material as it progressed through the furnace from hearth No. 1 (top hearth) and showed as follows:

Leaving Hearth No. 1   79.3% of copper water-soluble
    "   "   No. 2   81.4% "   "   "   "
    "   "   No. 3   93.4% "   "   "   "
    "   "   No. 4   95.2% "   "   "   "
    "   "   No. 5   95.2% "   "   "   "

There was no heat applied to the fifth hearth and the sulphatizing was completed in the fourth hearth. This calcine leached readily with cold water, using one ton of water per ton of ore. Solution secured was used on successive batches and brought up to 15% $CuSO_4$ containing only 0.1% Fe. The tailings from this leaching test contained 8-100ths of 1% Cu.

Up to this time, very few records are available from furnaces of large size. The furnace with which most of these records were made was a furnace 9′ 9″ in diameter and 12′ high, with five hearths and a preheating hearth. This furnace, a generation ago, would have been considered a large furnace, even with a capacity of only one carload of ore per week, but the commercial problems of to-day call for much larger appliances. Several large mechanical muffle furnaces are now installing by large copper companies and records from furnaces with an output of 20 tons to 40 tons daily, per furnace, will soon be available.

The above is a record indicating in advance the possibilities of a process which may soon, in special cases, displace the smelting process of copper production.

Date Due