PYRITE SMELTING

Edited by

T. A. RICKARD

FIRST THOUSAND

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INDEX.

PREFACE.

This volume is a reprint of the discussion on Pyrite Smelting which ran through various issues of *The Engineering and Mining Journal* between October, 1903, and February, 1905. The contributions have been revised by the respective authors, and the editorial comment appearing at intervals in the *Journal* has been modified into the introductory remarks, which now precede the succession of letters contributed by no less than twenty metallurgists. Emphasis was placed upon the value of the discussion, and a notable addition was given to the permanent usefulness of it, by a review prepared by Dr. Edward D. Peters, the professor of metallurgy at Harvard University.

Earlier, in 1903, a discussion arose consequent upon an article dealing with the relation between hearth-area and furnace capacity; this article and the criticism which it elicited are also published in this volume, together with three other papers germane to the subject.

QUESTIONS.

The following ten questions were sent out by the Editor of *The Engineering and Mining Journal*, and formed the basis for the ensuing discussion:

1. What types of ore are suited to the process?
2. Is hot blast advisable?
3. To what extent can fuel be eliminated?
4. What amount of copper is required for the collection of the precious metals?
5. What percentage of lime is necessary to a clean slag?
6. What percentage of zinc in the charge can be treated profitably?
7. What is the degree of desulphurization attainable?
8. What are the possibilities as to capacity of furnace?
9. What are the limitations of the process?
10. What is the relative economy as compared to rival processes?

INTRODUCTION.

The direct smelting of sulphide ores is a branch of modern metallurgy, the importance of which has been enormously increased during the last five years. As an aid to the beneficiation of low-grade copper ores, carrying small values in the precious metals, this method of treatment has formed the basis for the commercial success of several big mines in the United States, British Columbia and Tasmania, and promises to play a much larger part in the future development of other regions.

With a view to eliciting the best professional opinion upon the factors controlling the success of this type of smelting, we sent a list of questions to a number of successful metallurgists. Their replies were published in the discussion department of *The Engineering and Mining Journal*, with the hope that those of our readers who possessed experience in this branch of technical practice would contribute their views freely.

This hope has been abundantly fulfilled, and it is with much gratification that I am now enabled to republish, in more accessible form, the practical discussion which grew out of the questions put to my professional friends.

The term 'raw sulphide smelting' was used in starting the debate, and it was defined[1] as "the treatment of unroasted sulphide ores with a view to the formation of matte." It was held that 'pyritic' smelting and 'matte' smelting, as metallurgical terms, failed to cover the ground, because at first sight the former seemed to omit pyrrhotite ores, themselves an integral part of the mate-

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[1] *The Engineering and Mining Journal*, October 10, 1903.

rial to be considered in this connection. while the second included the treatment of pyrite which has been previously roasted, in part or altogether. But objections, well sustained, were made to the use of 'raw sulphide' as descriptive of the reduction of pyritic ores "by the utilization of the combustible qualities of the sulphur and iron with a view to the formation of matte."[1] Mr. W. R. Ingalls made the point that 'matte' smelting carried with it a reference to the product that is aimed at by the process; on the other hand, it is fair to say that 'pyritic' smelting suggests the kinds of ore susceptible of treatment; finally, 'raw sulphide' smelting describes the chemical condition of the material when sent to the furnace. Therefore it amounts to this, that the subject under discussion is the treatment of pyritic ores as raw sulphides with a view to the formation of a matte. To this may be added the corollary that matte is produced with the intent not only of extracting the copper, but also, and this is fully as important, of collecting the small quantity of precious metals occurring in the ores.

It is this last aspect of the inquiry that Mr. Godshall seeks to emphasize, and he also regards 'matte' smelting as a branch of 'pyritic' smelting; but this is scarcely an inference, because, as we have said, the one term refers to the material treated and the other to the product of that treatment; so they rank equally in descriptive value. The very fact that matte smelting may include the treatment of roasted ores does not make it more comprehensive, but, it seems to us, less definitive. Roasted ores are not a characteristic charge in this form of smelting, any more than the treatment of flue-dust is characteristic of ordinary blast-furnace work; it is merely incidental.

'Sulphide' smelting, indeed, may seem to include the treatment of any sulphide, and, therefore, for example,

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[1] *The Engineering and Mining Journal,* February 4, 1904.

the reverberatory smelting of galena or the reduction of cinnabar; 'matte' smelting fails to define the kind of smelting under discussion, because it also covers the production of copper matte in reverberatory practice.

'Pyritic' smelting describes the type of ore treated; that is, the iron sulphides which have a combustible value. It includes not only iron pyrite, but also the lower sulphide, pyrrhotite, often termed magnetic pyrite; it includes not only chalcopyrite, but arsenical iron pyrite as well. Thus it covers the ores which are essential for, and particularly adapted to, this kind of smelting—an oxidizing fusion in a shaft-furnace. But 'pyritic' is wrong; we do not speak of a 'leady' or a 'zinky' smelting when describing the reduction of lead and zinc ores. The termination 'ic' should refer to the nature of the process, and that is not 'pyritic,' but matting—the conversion of ore into matte.

'Pyrite' smelting may be objected to as covering only common pyrite, that is, iron sulphide, whereas the process covers the treatment of all the pyrites, not only the common iron sulphides—pyrite and pyrrhotite—but also the related varieties characterized by copper and by arsenic. However, the employment of the singular to designate the type which it represents is a usage neither infrequent nor incorrect. Therefore, we finally arrive at the conclusion that the oxidizing smelting of pyrite ores in a shaft-furnace with a view to producing a matte is best labeled 'pyrite smelting.' I have adopted this term.

The practice of pyrite smelting is based upon the utilization of the heat derived from the combustion of sulphide ores in a blast furnace, in contradistinction to the heat obtained directly from carbonaceous fuel. Of the sulphides, those of iron and copper are the best available, the sulphides of lead and antimony, for instance, being prejudicial to successful work by reason of their easy volatilization at the high temperatures required for a proper matte-fall, while the sulphide of zinc tends to form an infusible slag. The formation of matte and its downward passage through the charge, play the important part of collecting the precious metals which may be disseminated through the ores, and, in this way, render the process a form of concentration by fire as contrasted with ordinary wet milling practice. This feature of the process—the formation of a matte which acts as a collector of minute quantities of precious metals—has led to its extended application during recent years in the treatment of pyrite and pyrrhotite ores containing small quantities of gold and silver. Copper sulphide is usually considered a better collector than iron sulphide alone; but experience has proved that a very small percentage of copper, in the presence of an iron matte, is sufficient to yield a clean slag. What the irreducible minimum of copper is, remains still a matter for difference of opinion, and, we may add, a subject for discussion among specialists.

Just as in the concentration of ores by water, it is necessary to limit the number of operations, because, aside from the cost, there is also a loss entailed by each treatment; so in raw smelting it is economical to concentrate the valuable metals in as few smeltings as possible. Ores so heavy in copper and iron sulphides as to approximate the composition of matte, itself an artificial sulphide, do not gain by direct smelting unless they are mixed with silicious material, not as a flux alone, but as ore, able to defray a portion of the total cost of treatment. When one operation yields a 40 to 50 per cent matte with a concentration of less than two into one, it is obvious that the gain secured by the concentration may be largely offset by the cost and loss in treatment, so that it may be more profitable to dispose of the ore otherwise, as, for example, by reducing its sulphur content by roasting or by slagging off the excess of iron by a preliminary fusion, so that it may become part of a charge containing roasted material and silicious ores such as have not an excess of iron and sulphur, the matte-forming ingredients. When the ores to be treated are so poor in copper sulphides as to form a low-grade matte, requiring further concentration, it may be necessary, in order to secure a high-grade product fit for shipment, to repeat the process so many times as to entail a cost and a loss in treatment which will not justify the result. A large percentage of matte usually favors a clean slag and a small loss in precious metals, while a small proportion of matte indicates conditions unfavorable to the gathering together of the gold and silver, producing an unclean slag.

Nowadays, near well-established mining centers, the problem of metallurgical treatment is often overcome by the disposal of ore in the open market; but in the early days of a new mining region, before custom smelters offer a satisfactory market for the product of the mine, it is frequently economical to concentrate the metals into a high-grade matte, so rich that the expense of transport per ton of original crude ore is comparatively small. The shipment of bullion from a stamp-mill is the extreme phase of the same view of the problem of concentrating the values into as small a bulk as possible, in order to minimize the cost of transport to the refinery. For this reason small matte-smelters have flourished, in days now nearly past, in the mountainous parts of Colorado and in the deserts of Arizona—to mention two representative conditions. From this point of view, the matte-smelter may be regarded as a metallurgical pioneer, for it has helped notably in the early development of mining districts. With the advent of the railway and the consequent nearness to industrial centers, either the matte-smelter has been shut down in the face of the facilities afforded by central establishments which have brought the ores; or, when the mines of any one locality have warranted it, the matte-furnaces have been followed by the converter, so that shipments of matte have given place to shipments of blister copper.

This phase of the subject is now of less importance in the United States, although in the more remote places of the earth it may still be an important feature in the progress of mining. At the present time pyrite smelting, in this country, in Canada and in Mexico, is of particular importance as affording a cheap method of beneficiating those pyrrhotite and pyrite ores that carry small quantities of gold and silver, with enough copper to permit of a good matte-fall. As the costs of the operation are reduced until they approximate the expense of stamp-milling, the smelting of such ores renders valuable a proportion of gold and silver which not many years ago was considered negligible. With the attainment of so low a cost for raw smelting as 90 cents per ton, the operations of the furnace begin to compare in point of economy with the wet-concentration mill and the stamp-mill, itself held to be the cheapest form of ore reduction invented by man.

The consumption of fuel is a phase of the subject on which light was badly needed;[1] and, when the discussion developed, valuable data were forthcoming, as the pages of this little volume testify. Is it obviously necessary, in order to achieve maximum economy, to calculate the relative consumption of carbon when adding fuel to the charge in the furnace, consuming carbon in heating the blast, or consuming carbon when increasing the volume and raising the pressure of the blast? In other words, is it cheaper to expend fuel on the blast before it enters the furnace or to employ fuel direct in the smelting operation? Obviously, to a large extent, this, like the tariff, will be a local question, and will depend upon the cost of fuel and the relative amount required by different ores; but, however put, this is the main question at issue between the metallurgists who

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[1] *The Engineering and Mining Journal*, May 12, 1904.

employ pyrite smelting; as Mr. Ingalls has put it: "What is the cheapest way of getting rid of your sulphur?"

In this connection it is a question whether the extremely small proportion of carbonaceous fuel used in pyrite smelting does indeed take any physical part in keeping the charge open to the passage of furnace gases; on the other hand, it is certain that the minute you increase the percentage of coke in raw smelting, you are apt to fuse the charge too quickly and prevent oxidation of the iron, the result being a liability to an extremely silicious slag or else a silicious skeleton in the furnace, which means a freeze-up.

To prevent this evil effect of the coke, to obtain its mechanical aid in keeping crusts off the jackets, and to prevent the too sudden fusion of the sulphides, it has been found best to feed the coke to the sides of the furnace. This is the Mt. Lyell practice, also, recently adopted at the Tennessee Copper plant.

The doubts thrown upon the value of heating the blast provoked a lively controversy, which became quieted by the impressive effect of the facts brought out during the progress of the discussion. This important problem has been clarified. Beside the testimony given by Mr. Walter E. Koch and Mr. S. E. Bretherton, we can state that, at a well-known establishment treating raw nickel-copper pyrrhotite, under varying conditions of charge, with roasted and raw ore, and from 8 per cent down to 3 per cent coke, it was found that the grade of the matte was not appreciably higher than when the cold blast was used. The ores contained 3 per cent copper and 3 per cent nickel, and a concentration only of from 4.5 to 6 into one was desired. The blast temperature was 500° C. The metallurgist in charge came to the opinion that hot blast was advisable, but that direct smelting was extremely hazardous because the furnace "was likely to quit without notice," in other words, freeze up. He concluded that heap-roasting and rapid smelting were preferable to double smelting with hot blast. But he had an unusually bad ore to treat, and his experience was exceptional. Processes were made after ores were fashioned.

During the progress of the discussion a practical and forcible comment upon it has been offered by the decision, after proper trials, of the Tennessee Copper Company to discard the practice of heap-roasting preliminary to smelting, and to smelt the raw ores. Until lately the low costs achieved by Mr. Randolph Adams and Mr. W. A. Heywood had enabled the advocates of preliminary roasting as against straight pyrite smelting to quote the costs at Ducktown against those at Mt. Lyell, because Mr. Heywood's costs were $1.8296 per ton less than those of Mr. Sticht, who omitted roasting and employed double smelting with hot blast. But this comparison is no longer effective.

The value of the practical details given by our contributors will be appreciated by the reader, but two incidents bearing upon this aspect of the discussion are worth mention. At one of the largest establishments using the process of pyrite smelting, the capacity of the furnaces decreased to an alarming degree, and, despite unusual skill possessed by the metallurgist in charge, this decrease was wholly unaccountable. Just at this time, when the problem had become of serious moment to the commercial side of the operations, one of our contributors published an article, which appears in this volume, referring to the fact that analyses for alumina often fail to distinguish between the alumina which is already tied up to silica and satisfied in that direction, and the alumina which is available in slag-making. This hint saved the situation, for the decrease in capacity at the smelter referred to was due wholly to the increased proportion of a schistose rock encasing the ores. Another episode worthy of record was a visit by the editor of this little volume to a large copper smelter in Michigan, where he was introduced to the superintendent, who at that moment was watching the 'dipping of a charge,' but who found time to pull out a notebook from a pocket in his working clothes, and to exhibit the fact that it was full of the data contributed in the course of the discussion on pyrite smelting, the very material which the reader will find on the pages that follow.

Other discussions on this subject will doubtless arise, and each later one will afford knowledge more accurate than its predecessor, until finally that metallurgical millennium is reached where theory merges perfectly into practice and there is nothing more to be said. However, that date belongs to the Greek Kalends. In the meantime, the close of the present discussion marks a stage of progress which is epitomized by the experience of the technical men at the Tennessee Copper Company's smelter as expressed by their administrative chief, Mr. Channing. During the period of two years covered by the interchange of views on pyrite smelting, Mr. Channing and his associates have been applying the ideas distilled from this discussion, and at their smelter they have endeavored to work out some of the problems incidental to the development of a successful metallurgical enterprise. It speaks well not for them only, but for the new spirit which has come over the profession, that information of so detailed a kind should be given for publication where it will prove of most direct service to the industry. We shall be much surprised if the generous spirit thus evinced is not met half-way by those in charge of similar metallurgical enterprises, who, possessing data valuable to the officers of the Tennessee Copper Company, will give with equal generosity from the store of their experience also.

It cannot be said that pyrite smelting has yet reached its culmination; on the contrary, it is obvious that many questions remain unanswered and that the best men are strong in the belief that radical development is probable in the near future. The question of hot *versus* cold-blast has been well ventilated, the part played by the coke has been ascertained, the influence of zinc is recognized, and so forth; but the main point—the possibility of concentration, of making a matte of high percentage from low-grade ore in one operation, eliminating the cost and the loss involved in repetition of treatment—this is a problem upon the successful solution of which the future of this branch of copper metallurgy mainly depends.

T. A. Rickard,
Editor of *The Engineering and Mining Journal.*

June 29, 1905.

CONTRIBUTION BY L. S. AUSTIN.

The Editor:

Sir—I take pleasure in answering the questions you have submitted.

1. *What types of ores are suited to the process?*

Sulphide ores of iron and copper (pyrite, chalcopyrite, etc.) in quantity sufficient to give the necessary matte-fall. Ores containing lead or zinc are not desirable, though they may be added sparingly. In such cases part of the lead and the zinc is volatilized, the remainder going into the matte and slag. Silicious ores containing the precious metals can also be added to the charge in quantity sufficient to form, with the iron of the sulphide and other ore, a suitable slag. Any oxidized ores containing iron and copper can also be run with the addition of limestone in quantity sufficient to form a fluxing mixture.

2. *Is hot blast advisable?*

Hot or even warm blast is of assistance in raw smelting, but in the past, where a hot blast has been used, it has been, in several cases, abandoned. It is probable, however, that where the hot-blast apparatus can be carefully installed and properly looked after, it would be of great advantage to use it, especially in undertaking to smelt a very silicious or comparatively infusible charge. Such is the practice at the Mansfeld works in Germany.

3. *To what extent can fuel be eliminated?*

In the older methods of matte smelting 10 to 15 per cent of fuel is used. In smelting with an abundance of sulphide ore, the amount has been cut to 6 per cent, and at times to even a less quantity. With a rapid-running charge the quantity of fuel may be less than where a slower-running, silicious charge is being put through.

4. *What amount of copper is needed for the collection of the precious metals?*

A charge containing sulphide ores, but quite free from copper, will not give a clean slag, but with as little as 0.5 per cent of copper, this object can be attained. This is especially the case where the other conditions for making a clean slag are present, viz., a charge free from zinc, the use of sufficient fuel and the formation of a fusible slag, as well as a maximum difference between the specific gravities of the slag and matte.

5. *What percentage of lime is necessary to a clean slag?*

Limestone is added to the charge in quantity sufficient, with the iron available, to form a fusible slag. It has not been the aim to use it as a cleaner; but, where present, it has been thought to act in this manner. Slags containing as little as 5 per cent CaO have proved clean.

6. *What percentage of zinc in the charge can be treated profitably?*

Zinc up to 10 per cent of the charge has been treated successfully. As blende it goes into solution in the slag, and in oxidized form, as contended by some metallurgists, it still exists in solution, and not as an active base. Certainly the slag becomes stiffer as the percentage of zinc increases. The specific gravity of the slag becomes higher and its separation from the matte is less perfect. Some of the zinc also goes into the matte, rendering it lighter, and thus increasing the difficulties of separation. By increasing the proportion of silicious ore we produce more slag to dissolve and carry off the zinc, thus promoting the ease of separation in the furnace. In presence of much alumina, zinc acts badly, and makes continual trouble, so that such combinations are to be avoided.

7. *What is the degree of desulphurization attainable?*

In raw sulphide smelting, where pyrite is used, its first equivalent of sulphur is loosely held, and is largely driven off in the upper part of the furnace. It has been found in practice that at least 66 per cent (two-thirds) of the sulphur is driven off, or, at most, only 34 per cent is left to enter the matte and slag. The metallurgist should, as soon as possible, determine the quantity of sulphur both in his matte and in the slag, as a basis of calculation for his charges. He can calculate the quantity of slag from the known silica contents of his charge, and he must keep account of the weight of the matte-fall as compared with the tonnage put through.

8. *What are the possibilities as to capacity of furnace?*

The speed at which a given matting furnace can be driven depends, other things being equal, upon the nature of the charge. A silicious charge drives slowly. Zinc and alumina also cause slow running. On the other hand the mono-sulphides, as pyrrhotite, conduce to rapid running, since there is no first equivalent of sulphur to drive off at the surface of the charge, and the sulphur gives off its heat lower down in the furnace, where it is most useful. Above all, a fusible, somewhat basic slag, produces rapid running. An open charge, where there is no excess of fines, is also an advantage. Where all these favorable conditions exist together, as in the Boundary district of British Columbia, as high as 450 tons per day of 24 hours have been put through a single furnace.

9. *What are the limitations of the process?*

Where there are no lead ores, matte smelting is often the only method which can be used for reducing ores. Sometimes, as shown at Leadville, it can be used in competition with lead smelting. In this case, the omission of roasting, the rapidity of the process, the low fuel consumption, all assisted in giving it the advantage. There is the drawback that much flue-dust is made; this should be bricked before it is re-smelted. Practically, by the time it has been treated over again, the losses approach the percentage loss in flue-dust. In other words, to collect, brick and re-smelt has cost nearly what the flue-dust contained. As before intimated, zinc and alumina are detrimental to steady running. It is the uncertainty of operation which tells so against raw matte smelting, and it may be said that success in it depends on skill in furnacing. Those who are so skilled keep these various matters almost as trade secrets, and indeed the operation of a matte furnace requires experience, and the ability to judge, from the appearance of the furnace, what remedies to apply or what changes to make. The writer, for example, has seen where merely changing the height of the smelting column has caused a badly operating furnace to work smoothly. As has already been indicated, matte smelting depends on having a supply of sulphide ore and upon there being copper available in the charge.

L. S. AUSTIN.

New York, Oct. 10, 1903.

CONTRIBUTION BY E. P. MATHEWSON.

The Editor:

Sir—Herewith please find replies to questions.

1. Types of ore suited to the process. Any ore carrying sulphur in the form of iron pyrite, copper pyrite or pyrrhotite.

2. Hot blast is advisable where it is necessary to increase the grade of matte produced, provided sulphur on charge is not excessive and where coke is expensive and coal or oil cheap.

3. Fuel can be eliminated entirely, provided there is sufficient pyrite in the ore charged. This condition is rarely encountered, the lowest practical elimination in the United States being probably down to 4 per cent.

4. Copper is not necessary for the collection of the precious metals; they will collect readily in an iron matte.

5. Lime can be used successfully up to 31 per cent of slag, as was proved at Anaconda, and I believe higher percentages have been used elsewhere. It is a question of relative economy.

6. Zinc has not been found in large quantities in the ores of this vicinity, so that the limit of the amount that can be treated profitably has not been reached here.

7. The degree of desulphurization attainable on pyrite is 80 per cent with the Butte ores; we readily get 75 per cent. This can be varied with the depth of the charge and heat of blast.

8. Capacity of the furnace is best regulated by the means of getting rid of the slag, and the facilities for charging the furnaces. We consider 600 tons in 24 hours the capacity of our 56-in. by 180-in. at the tuyeres.

To put more than this through the furnaces would require another crew of feeders, which would not be economy.

9. Limitations of the process. Ore too fine, say less than quarter mesh, or too clayey. To have good work it is necessary to have fairly open charge.

The percentage of sulphur may vary greatly. In most cases the grade of matte produced fixes the limit of the process. Where the matte is too low grade by the raw smelting process, roasting has to be resorted to. The aim, in most copper smelters, is to produce 50 per cent at the first smelting, though some have to smelt twice to get it to this point, roasting the first matte between the smeltings. Matte charged again to a furnace raw loses hardly any sulphur in the smelting process, as it merely smelts and runs through the charge.

10. Relative economy. Where this process can be used at all, it is practically the cheapest known.

E. P. MATHEWSON.

Anaconda, Montana.

Contribution by Walter E. Koch.

The Editor:

Sir—I beg to inclose answers to your questions on pyritic smelting. We have two 48-in. water-jackets, with twelve 3.5-in. tuyeres. The distance from charging floor to tuyeres is 57 in. and from tuyeres to taphole 18 in. The capacity of each furnace is 50 tons per 24 hours. The blast is heated entirely by waste gases from the furnaces and boilers (and gas-engines later on).

The analysis of our slag is:

Silica	41 to 45 per cent.
Iron	27 to 31 (as protoxide).
Alumina	5 to 7, seldom above 5.
Lime	5 to 15, seldom above 10.
Magnesia	2 to 5 per cent.

The color is green, a dark green silicate of protoxides.

Pyritic smelting, so-called, is purely a roasting process, and what we cannot burn off goes to matte, so that it is really the exact opposite of ordinary smelting.

1. *What types of ores are suited to the process?*

All kinds of ores are suitable so long as you have sufficient sulphur and iron. We find after two years' continuous practice that 20 per cent of FeS_2 in the charge gives good results.

We have three well-marked classes of ores:

(1) Quartz-pyrite = 70 per cent quartz and 30 per cent pyrite.

(2) Spar ores = 35 per cent quartz, 10 per cent pyrite and 55 per cent calcite.

(3) Red ores = 40 per cent quartz, 25 per cent iron oxides, 20 per cent calcite and 10 per cent alumina. Also pure pyrite and some brown iron ores. These classes of course shade off into one another.

A general analysis of the ore from the mine made three years ago gave: Si, 44 per cent; Al_2O_3, 5 per cent;

$CaCO_3$, 32 per cent; $MgCO_3$, 2 per cent; FeS_2, 15 per cent; $FeCO_3$, 3 per cent; Cu, 0.25 per cent; Mn, 0.20 per cent; Au, 0.5 oz.; Ag, 0.25 oz. per ton. The copper now runs about one-half of one per cent and seems to increase with depth. We have also more iron and less lime now. This sample gave poor results, both with barrel and vat chlorination, and cyaniding, both raw and roasted; and so we were led to try pyritic smelting.

2. *Is hot blast advisable?*

A warm blast of 200° C. is a *sine qua non* with us; it spelled success, cold blast meant failure. I have not as yet found any advantage in heating above 200° C. Our hot blast costs us nothing but the stove, designed at Pittsburg, Pa., by Mr. John Macdonald. The furnace itself and the waste heat from boilers and gas-engines will give all the heat necessary if intelligently applied. Furthermore, with a hot blast you can run clean slags without using barren fluxes to the extent required in cold-blast plants; for example, lime may be cut down to 5 per cent without danger.

The man who has never used hot blast can form no opinion on its use, as the results are so different from cold-blast practice. It not only saves fuel, but gives the smelter a wider and more economical range of clean fluid slags, and makes work easier and pleasanter.

3. *To what extent can fuel be eliminated?*

We find five per cent of coke to be the lowest percentage giving good results. We generally use 7 per cent of the ore charged, or 6 per cent of total charge.

4. *What amount of copper is needed for the collection of the precious metals?*

We use half of one per cent of copper in the charge to collect the gold and silver; it is quite sufficient and the slags run clean. We concentrate fifteen to one, *i. e.*, 15 tons of ore to one ton of matte; and the copper runs from 7 to 8 per cent in the matte which we ship to the smelter. We have had mattes containing only one per cent of copper carrying 200 grams of gold and 60 grams silver with a perfectly clean slag.

5. *What percentage of lime is necessary to a clean slag?*

Seven per cent of lime is enough with hot blast.

6. *What percentage of zinc in the charge can be treated profitably?*

Zinc sulphide is a very good fuel, but I do not know how much we could profitably use.

7. *What is the degree of desulphurization attainable?*

At present about 80 per cent of the sulphur in the charge is burned off, but I think we can do better.

8. *What are the possibilities as to capacity of furnace?*

This is quite unknown to us at present. We have two 48-in. diam. furnaces running continuously. The capacity is 50 tons each per 24 hours. Our new furnace is 126 by 42 in. and 7 ft. high from base-plate to charging floor.

9. *What are the limitations of the process?*

A lack of sulphur and iron. You must have FeS_2.

10. *What is the relative economy as compared to rival processes?*

In our case, after failing with freemilling, concentration, chlorination and cyanidation, both separately and combined, we succeeded in saving practically all our values by pyritic smelting. To me it seems that cyaniding is the only possible rival, but it did not save the silver and copper, and only about 70 per cent of the gold in our case.

We take the ore as it comes from the mine and dump it straight into the furnace, merely spalling the big chunks. The ores required reducing, in rolls, and roasting for both chlorination and cyaniding in our case, and you can smelt pyritically about as cheaply as by milling and roasting. A milling and roasting plant costs more than a pyritic smelter and takes more power to drive it, and then you have to add cost of cyanide or chlorination plant. Again, and worst of all, you have divided superintendence, the millman, roaster-man (concentrator-man if needed) and cyanide expert, all have to be considered; whereas by taking your ore straight from mine to furnace, this friction is avoided. The copper and silver lost in the cyanide and chlorination methods now yield us over $100 per day and the sulphur is utilized as it should be, *one* pound of sulphur being equal to *half* a pound of coke in fuel value. As far as I know, there is no single process which can save practically all the values of gold, silver and copper in ores such as ours, so simply and economically as the hot blast pyritic method.

WALTER E. KOCH.

Santa Maria del Oro, Mexico, June 8, 1903.

CONTRIBUTION BY S. E. BRETHERTON.

The Editor:

Sir—In reply to your ten questions in regard to raw smelting, "that is, the smelting of sulphide ores without previous roasting," I shall endeavor to answer them in the order in which they are given, and trust my replies will be as interesting to the readers of your valuable paper as the replies of others to the same questions will be to me.

1. *What types of ores are suited to the process?*

Types of ore containing gold, silver, copper and lead, if the value of the lead is less than the additional cost to roast and then smelt in a lead furnace, which requires more fuel than the so-called pyritic process.

We put all ores containing less than 10 per cent lead, unless a carbonate, into the matting furnace, and sometimes ores that contain more than 10 per cent lead, if they are refractory—that is, containing a high percentage of zinc and arsenic.

While answering No. 1, I deem it advisable to answer No. 9 at the same time:

9. *What are the limitations of the process?*

This depends a great deal on the local conditions. Where a company is running a custom smelting plant, and can dilute the zinciferous ores and concentrates from one mine with the clean silicious ores from another by judicious mixing, zinc ores may be worked off, no matter how much zinc they contain.

Again, if base ores are plentiful and contain sufficient values to allow what is usually accepted as a fair treatment, the smelter may treat silicious gold and silver ores very cheaply, so that when competing with small milling and concentrating plants, where the cost of milling or concentrating is at least $2 per ton, add this cost of milling to the cost of marketing the concentrates, wear and tear of the plant, and the loss in the slime and tailing (which is generally 15 to 40 per cent of the silver, from 10 to 25 per cent of the gold, and from 15 to 50 per cent of the copper, if the ore contains any copper), and it will come to an amount equal to the cost of smelting, that is, if the mine is conveniently located for shipping to a local smelter, by rail. Of course, the richer the silicious ores are in precious metals, the greater the loss in dollars and cents, and the easier it is for the smelter to compete with the concentrating or milling plant.

But, in the San Juan region of Colorado, in Yavapai county, Arizona, and other sections of the country, the concentrating mill is a great benefit to the smelter, since crude ores as they are produced from the mines, even if they contain values to stand smelting direct, are often too silicious, whereas by wet concentration, the mills, by their treatment of the lowest grade of ores, are not only putting from 65 to 85 per cent of the values of anywhere from two to fifteen tons of low-grade ores into one ton of concentrates, *i. e.*, if the ore is at all suitable for concentration, but at the same time are eliminating the silica, and producing a basic concentrate to be shipped to the smelter, which it in turn can utilize as a base to flux the richer but still silicious ores that are shipped from the mines direct without concentration.

2. *Is hot blast advisable?*

To this question I must emphatically say Yes, for the following reasons:

Increased capacity of the furnace. (Of course the increased tonnage is partly offset by having a decreased amount of ore to smelt by preliminary roasting, on account of the loss of weight in roasting.)

Saving of fuel consumption in the blast furnace.

Saving the cost of preliminary roasting, and the saving of preparing the ore for roasting.

Brighter tuyeres, and less trouble in keeping them open.

Less trouble with ores containing zinc (as zinc seems to be more easily desulphurized in the blast furnace with forced blast, under intense heat, than it is in the ordinary calcining furnace) and the elimination of arsenic, which, so far, has not been successfully accomplished with cold blast.

The advantages in the use of hot blast just enumerated, reduce the cost of smelting nearly half on the heavy sulphide ores, and make success possible, when success would be practically impossible with ore containing much arsenic and zinc, and small percentages of lead and antimony, which are so heavily penalized by the refineries when reduced into matte with cold blast.

3. *To what extent can fuel be eliminated?*

I do not think it advisable to reduce the fuel below 4 per cent on the ore and flux, although it is possible to run for days at a time as low as 3 per cent on the burden, that is, not including the weight of the fuel itself.

4. *What amount of copper is needed for the collection of the precious metals?*

This depends somewhat on the character of the ore to be treated, especially where there is much zinc. I do not think it advisable to smelt with less than 1 per cent copper on the burden, and prefer 3 per cent. Of course, ores averaging as low as one, one-half or one-quarter per cent copper can be smelted by re-smelting a small portion of the matte containing the accumulated copper over and over again with the charge, in order that it may continue to gather the values. We have an excellent illustration here of the advisability of using sufficient copper. Even when making matte as high as 15 to 25 oz. gold, and containing from 300 to 400 oz. silver, with plenty of copper, by using the enriched matte over and over again in order to concentrate the copper up to the shipping point, our slags are as low in gold and silver contents as when making a matte containing only 4 or 5 oz. gold and 100 oz. silver with 15 lb. and less copper on the 1,500 lb. charge burden, the slag running from trace to .03 oz. gold and from .25 to 1 oz. silver to the ton.

5. *What percentage of lime is necessary to a clean slag?*

This depends a great deal upon the amount of alumina and magnesia contained in the ore to be smelted, as it is necessary to have a fluid slag of light specific gravity, to avoid mechanical losses in the slag, yet not so light that the fluidity of the slag is sacrificed. For these reasons I like a little alumina and magnesia in the ore, especially when we have plenty of iron and do not care to use too much lime. Of course, where parties are anxious to smelt as much silica as possible with as little flux as possible, and magnesium lime rock is available, one would naturally think it would be a very desirable flux, as magnesia has nearly twice the combining power with silica that iron has, and nearly half as much again the combining power of lime, that is for a bisilicate where 1 per cent CaO is equal to 1.07 per cent SiO_2, 1 per cent MgO would be equal to 1.5 per cent SiO_2; but we well know what an infusible slag we would get from a bisilicate of magnesia. As a rule, I have found a bisilicate slag, classing the alumina as an acid containing anywhere from 22 per cent to 30 per cent Fe, that is, 28.3 per cent to 38.6 per cent FeO respectively, to give excellent results, the balance being CaO, ZnO, etc.

On several occasions I have endeavored to make a slag high in silica and alumina, in order to force the alumina to act as a base, and found that some of the silica would soon show on the surface of the slag-pots not fused, and the slag would scarcely run. At other times I have been able to make slags extremely low in silica, on account of its alumina contents. Classing alumina as an acid, I have successfully made slags extremely clean, containing over 20 per cent alumina.

Of course we cannot compare sulphide smelting in shallow furnaces, where both time and reducing agents are limited, with the deep iron furnaces, where they use so much fuel, causing a strong reducing action and an intense heat, especially with hot blast.

I do not think it advisable to attempt to make a slag containing more than 12 per cent zinc; that is, 15 per cent ZnO, and the FeO must be reduced in proportion. So that for practical running I would put the maximum amount of ZnO at 11 per cent, and the minimum amount of CaO at 10 per cent, assuming that there are no magnesia or alkaline bases present.

Of course, in matte smelting it is not practical or necessary to figure out, as we do in lead smelting, the exact combining power of the lime, magnesia, iron and zinc contents with the silica, and know that the slag will contain to within 0.5 per cent of the silica, iron and lime, before putting on the charge, as in sulphide smelting we have to depend on the feeding and proper oxidation of the furnace for all our iron, so that a little carelessness on the part of the feeder, or an increased amount of fuel, will produce more matte and decrease the iron in the slag.

6. *What percentage of zinc in the charge can be treated profitably?*

This question is answered in my reply to No. 5.

7. *What is the degree of desulphurization attainable?*

This is a question I cannot answer definitely, for the reason that in making up a charge, the first thing to be considered is to use sufficient silica to combine with the bases, by adding enough lime or magnesia to make the slag of the proper specific gravity. My experience has been that by the time this is done the percentage of sulphur is so reduced that a person can concentrate anywhere from 8 to 12 tons of ore into one ton of matte, which is all that is advisable, and have a proper matte-fall for fast running and keeping the crucible hot. I have made much higher concentration than 12 to 1, but found it was not practicable nor successful, and for this reason I cannot answer definitely the question as to the degree of desulphurization obtainable, but I will say that I think a concentration of 12 tons of ore into one ton of matte in one operation, the maximum.

8. *What are the possibilities as to capacity of furnace?*

This, as in cold blast smelting, depends a great deal upon the character of the ore, matte-fall and percentage of copper in the charge. Plenty of copper and large matte-fall means faster running and less fuel consumption than a limited amount of copper and higher concentration, but I would say that the capacity of the furnace with hot blast, as compared with cold, would be fully one-third more.

9. *What are the limitations of the process?*

This is answered in the reply to No. 1.

10. *What is the relative economy as compared to rival processes?*

This is answered in the reply to No. 2.

S. E. BRETHERTON.

Val Verde, Arizona, Oct. 10, 1903.

CONTRIBUTION BY F. R. CARPENTER.

The Editor:

Sir—I take pleasure in answering your questions.

1. *What types of ores are suited to the process?*

Any gold, silver or copper ores not carrying lead in paying quantities may be treated by raw smelting. If the ores carry no sulphide material, as at Deadwood, it must be added in quantities at least sufficient to make a matte. If they carry no silica, it must be added to slag the iron. If iron is scarce, limestone may be added for flux. If sufficient copper is in the charge to bind the sulphur, satisfactory slags carrying not more than 4 per cent iron, as at Mansfeld, can be made. If iron is plentiful, lime may be omitted.

2. *Is hot blast advisable?*

It may be advisable, but I am sure that it is not an absolute necessity. It probably lowers the zone of fusion to a point nearer the tuyeres, but there is a question as to the desirability of having it there. If much iron pyrite is to be oxidized, the 'noses' formed in front of the tuyeres may become very desirable things to have; in any case, one does not burn much iron until after they are formed. If the hot-air stoves could be heated by waste gases, as in iron smelting, it would pay. It reduces the fuel to the extent of heat so added; but the best hot-air stove made is a very wasteful machine when heated with extraneous fuel, as a large percentage of the heat-value of the fuel is lost. I have never found the magic in mere hot air that others have professed to find.

3. *To what extent can fuel be eliminated?*

All carbonaceous fuel can be eliminated. and in ideal or true pyritic smelting it is highly desirable that it should be. First, the process is an oxidizing process, and not reducing. Second, the ignition of the coke tends to fuse the unoxidized pyrite, causing it to run away in the form of low-grade matte, thus losing to the charge both its fuel and flux values.

4. *What amount of copper is needed for the collection of the precious metals?*

It depends somewhat upon the degree of concentration attempted. If very little matte is made, it should carry perhaps 10 per cent copper. If a very large percentage of matte is made, 2 or 3 per cent of copper is sufficient. This would mean that the charge should carry only traces of copper. If the ores carry both gold and silver, and a fair quantity of matte is made, copper may be entirely avoided. We ran the Deadwood plant for four years without other than mere traces of copper in the matte, certainly less than one per cent. The slags carried from fifty cents to one dollar and a half in gold, and invariably ran up and down in proportion to the quantity of matte made. I do not think that the addition of copper in after years made much difference in the saving of the silver, but it did help in the saving of the gold. I ran all grades of copper matte from mere traces to 30 to 40 per cent copper. Beyond 10 per cent there was no gain. Our matte at Golden averages 6 per cent copper.

5. *What percentage of lime is necessary to a clean slag?*

In ideal pyrite smelting, lime is a nuisance to be avoided if possible. It costs money to purchase it, labor to handle it, fuel to smelt it, and it adds to the quantity of slag made, thereby increasing the quantity of slag losses. In the formation of the slag it takes the place of iron, thereby lessening the degree of concentration.

Upon the other hand, it tends to lessen the specific gravity of the slag, and, within limits, lowers its fusing point. I have not succeeded in using lime to an advantage where I sought to do genuine pyritic smelting. I think, all things considered, especially if the ores carry alumina, lime had best be avoided, particularly in the first smelting. The following slag is made at Golden with a cold blast and 5 per cent coke:

Silica	33.50	per cent.
Iron	32.26	" "
Lime	11.42	" "
Alumina	12.00	" "

with small quantities of magnesia, alkalies, etc., to balance. It is practically free from gold, silver, and copper.

6. *What percentage of zinc in the charge can be treated profitably?*

I do not know. A little zinc seems to make no trouble. It goes partly into the matte, partly into the slag, and partly up the chimney. The smelter at Buena Vista has treated ores heavy in zinc and the experience obtained there would doubtless prove of value in elucidating this branch of the inquiry.

7. *What is the degree of desulphurization attainable?*

It depends entirely upon the charge. With pyrite, in the worst cases 50 per cent of the sulphur is lost at once by the volatilization of one atom of sulphur. At Deadwood in former days, with no copper and but little pyrite in the charge, we often failed to make any matte at all, the whole of the pyrite being oxidized and slagged. At Golden, Colo., and at Florence, I so arranged the slag elements as to call for about 16 per cent FeO. In a lime-alumina-silica slag, such as I make, the furnace can be depended upon to take this amount of iron when fed in the form of pyrite. There is a selective action in the furnace upon which I have heretofore insisted; *i. e.*, with a given ore and a definite amount of limestone the slag will take about so much iron and no more, the excess remaining over for matte. It is well to study the requirements of the furnace and supply them. It would almost seem that the furnace at times acted with intelligence, and could be trusted to do the right thing.

8. *What are the possibilities as to capacity of furnace?*

With two furnaces of the same size, one doing pyritic smelting proper and the other running upon ores prepared by roasting, the latter will smelt twice as many tons as the former; but when we consider the time lost in roasting the ores and the cost, the advantage will be found with the first furnace. One furnace which I have used, that is three by sixteen feet at the tuyeres, running upon a charge consisting of silicious pyrite and limestone, and making the following slag, with either hot or cold blast, will average 250 tons of charge per day:

SiO_2	38 to 40	per	cent.
CaO	27 to 30	"	"
FeO	16 to 18	"	"
Al_2O_3	10 to 12	"	"

If this furnace were 50 per cent wider, as it should be, I believe it would smelt 50 per cent more ore. There is no iron going into the furnace that is not in the sulphide form. All the iron, therefore, in the slag is obtained from the burning of pyrite within the furnace.

9. *What are the limitations of the process?*

As I understand the question, there are practically no limitations if the ores do not carry lead. In smelting for gold and silver, if the ores are very high grade, it would probably be best to sell them to lead smelters. With these exceptions I think the question is answered in my reply to the first question.

10. *What is the relative economy as compared to rival processes?*

The roasting of sulphide ores is avoided, and the interest upon the large amount of capital tied up in the roasting heaps saved. In the actual smelting, fuel is economized. In many places where water concentration is now employed, pyritic or raw smelting, by reason of its closer saving, is more economical. At Deadwood, quantities of silica were slagged off and ores treated that would not have been profitable in lead smelting.

Without meaning to criticise smelting as now carried on at Butte, Mont., I have often thought that raw smelting along the lines which I believe I was first to employ at Deadwood, and later at Golden, would be well suited for Montana ores, and would result in the saving of much of the concentrating and roasting machinery now employed there. It is hardly a mere opinion, as I have smelted many thousands of tons of Butte ores.

Let us suppose that by mixing the more silicious with the more pyritiferous ores an average of 5 per cent copper, 55 per cent silica and alumina could be maintained. I do not know the cost of coke, coal and limestone at Butte, but upon a Denver basis the cost for limestone, coke and hot blast sufficient to smelt ores of the above grade would be $1.71 per ton. This charge is now ready for smelting raw, and to it must be added the cost for administration, labor, etc., against which have been saved the loss by concentration, the cost of concentration and the cost of roasting. There are slag losses in the smelting of the above charge, and there are also slag losses in the present method of smelting calcines, but the above lime-silica slags are wonderfully free from copper, gold and silver, but, of course, far greater in quantity. I cannot say that the total amount of copper produced would upon the whole be greater, but I think it would. The resulting matte in the above charge should be high enough for immediate bessemerizing; but if lower grade ores were used, a second or concentration smelting might become necessary, which costs but very little per ton of original ore.

I am informed that the loss by water concentration at Butte is 19 per cent. If it is, this would go far toward offsetting the extra slag loss, if any, in the raw smelting.

F. R. CARPENTER.

Denver, Colo., Oct. 10, 1903.

CONTRIBUTION BY W. R. INGALLS.

The Editor:

Sir—In reply to your request to participate in the discussion on raw sulphide smelting, allow me first to raise the question as to the descriptive phrase, which is substituted for what we have heretofore known as 'matte' smelting or 'pyritic' smelting. Fault can be found with any one of these phrases, yet it seems to me that the old description of 'matte' smelting is least open to criticism. It refers to the product that is aimed at, whether the crude ore be smelted raw, or roasted, and, as I shall attempt to show, the two methods merge into each other so that there is no sharp dividing line, and no one misunderstands what is meant. On the other hand, the term 'raw sulphide' smelting may be applied to other processes than the smelting of a pyritous ore to a matte. For example, galena is smelted directly, both by the precipitation method in the blast furnace and by the roast reaction method in reverberatory furnaces and Scotch hearths, and those certainly are forms of raw sulphide smelting.

In the matte concentration of pyritous ores, every one, who has attempted a roasting in heaps or stalls, has observed the formation of stall matte, and the idea of effecting the concentration of the ore by one smelting operation, utilizing the fuel value of its sulphur contents, must have occurred early to metallurgists. This was developed into the process of pyritic smelting, which excited much discussion a few years ago. The great feature of that process was to dispense entirely with the use of carbonaceous fuel. I believe that has indeed been done at Tilt Cove, Newfoundland, and at Mt. Lyell, Tasmania, but in general it has not proved to be a practicable process. The smelting of raw pyrite with a comparatively small proportion of fuel has become, however, a recognized metallurgical method, which is employed successfully at numerous places. The distinction between this and smelting after a preliminary roast is only one of degree, and the choice of a particular method must be governed in each case by the conditions.

In most blast furnaces employed for the smelting of cupriferous pyrite there is, beside the reducing action, a more or less powerful oxidizing effect, which serves to burn off part of the sulphur in the charge. The same is true, even in the lead blast furnace, and it is now the common practice to put rich galenas on the charge raw, some of their sulphur going into the matte and some being burned off. All of these furnaces have a factor of oxidation, which must be known approximately before the charge can be calculated. This is true, whether we are considering the smelting of raw pyrite or pyritous ore that has previously been burned down to 7 or 8 per cent sulphur. The chief question is, What is the cheapest way of getting rid of the sulphur?

In this connection it is to be remarked that the roasting of pyritous ores is not necessarily an expensive process. Lump ore can be roasted in stalls for about 50c. per ton, including transportation to the furnace, and the first cost of plant is moderate. Fines can be roasted with the lumps to the extent of, say, 10 per cent. Fine ore alone is roasted at Butte, Mont., in the McDougal type of furnace for as low as 33c. per ton, not including transportation to and from the furnace. In order to determine the best method of smelting, such figures as these have to be combined with the cost of the blast-furnace work, taking into account the quantity of fuel consumed in the latter, the rate of smelting, the matte-fall, etc. The conditions are affected, of course, by the character of the original ore, which may be all fine, or composed of lumps which crumble and are unsuitable for stall roasting. In large operations also, the process has to be considered in connection with the converting of the matte into blister copper, which final step is, indeed, pyritic smelting.

One of the most interesting fields of matte smelting is the concentration of values at mines in remote locations, as, for example, in Mexico. Cases can be found there where ore worth $50 per ton is produced at localities whence freight and treatment charges come to $70 per ton, obviously an impossible proposition, without an intermediate step. By effecting a concentration of 10 to 1, a ton of matte, worth, say, $400 to $450, may be produced from 10 tons of ore, and business can be done. In such undertakings small furnaces are likely to be employed, and the conditions are widely different from those which obtain at Salt Lake City, in British Columbia and other places, where big tonnage is to be handled; charcoal may be the only available fuel, a high degree of concentration is desirable, and comparatively easy-running slags are likely to be a necessity. I am disposed to think that these results can be more easily obtained by smelting partially roasted ore than by smelting raw ore.

In one undertaking of that kind, we aimed to burn the ore in stalls as completely as possible. There was a surplus of fines, and these we jigged, putting the concentrates on the stalls as covering. The furnace was a small one, and, with too small a matte-fall, gave trouble. This could be remedied by putting on the charge some raw concentrates, of which there was always a large supply, and thereby increasing the matte-fall. If there is too little sulphur in a charge it is always possible to tone it up to any desired degree, without any trouble at all.

We had an ore which contained about 50 oz. silver and 1 oz. gold per ton, with generally about 1 per cent copper. There was a high recovery of gold and silver, and, apparently, there need be no fear on that score, when there is 1 per cent copper in the ore, or even less. An important consideration, however, is the relative specific gravity of the matte and slag, and the temperature at which they are discharged into, and maintained in, the settler. The loss of value in the slag is due largely to particles of entrained matte, and there will be obviously the best separation, at a certain temperature, when the slag is silicious or calcareous (of low specific gravity) and the matte is high in copper (of high specific gravity). The most unfavorable combination occurs in smelting a zinky ore, when the matte is bound to be mushy and light, on account of the presence of zinc, and the slag must be high in iron, therefore heavy, in order to carry off zinc. The difference in specific gravity between matte and slag is then small, and both matte and slag are rather infusible.

The pyritic smelters pronounce zinc-blende a good fuel, but I have not found it to be a desirable constituent of the charge for a matte-smelting furnace, at least not for a small furnace. Blende will burn fairly well in properly designed kilns; in stalls the combustion is very feeble and soon ceases. In the quick descent in a blast furnace a good deal must escape oxidation, and difficulties will ensue. There will be less trouble if the blende be in small pieces than if it be in large ones; but I should prefer to have the blende as completely oxidized outside of the furnace as possible, since then it will be easier to get rid of in the slag. There is a difference of opinion as to whether zinc oxide is carried chemically or mechanically in slag. It may be carried in both ways, but at all events there is no doubt that a slag, to be high in zinc, must be also high in iron and low in silica and lime. I have known slags containing 12 to 15 per cent ZnO to be successfully run in matte smelting, and 20 per cent ZnO is regularly run in lead smelting at certain places; but such slags must be high in iron.

The main difference in the matte-smelting process, as now carried out, lies in the percentage of fuel used. In the roasting-smelting process the proportion of coke may be something like 10 per cent of the charge; in raw-smelting it may be about 5 per cent. With coke costing $10 per ton, there is then a difference of 50c. per ton of charge on fuel account, which will come to somewhat more per ton of ore, according to the percentage of ore in the charge, to offset the cost of roasting. There is, on the one hand, a greater quantity of coke to be handled, but it is also to be remembered that the ore will have lost approximately 20 per cent of its weight in roasting. The difference in coke consumption measures substantially what is gained through utilization of the maximum fuel-value of the ore. The matte from the raw-smelting will probably be of lower grade, and the cost of converting it comes also into the consideration.

The behavior of lead in the matte-smelting furnace is incidentally of some interest. I once put some galena on the charge as an experiment, and found that the lead was mostly volatilized, as was to be expected. The ore smelted contained, however, regularly a little lead, 0.5 to 1 per cent, some of which went into the matte, and a little was reduced, although there was a strong oxidizing effect in the furnace. The metallic lead would work down into the brickwork of the hearth, and was extraordinarily rich in gold and silver, assaying about 7 per cent silver and from 0.5 to 2.5 per cent gold. Also, we found that the fire-brick would absorb gold and silver in a remarkable way, old bottoms assaying from 150 to 250 oz. silver and 8 to 20 oz. gold. The brick was colored a light purple and was visibly permeated with metallic specks and filaments, probably of speiss. The use of brasque bottoms prevented seepage of the rich lead, and they did not absorb gold and silver to anything like the extent the brick did.

W. R. INGALLS.

Boston, Oct. 29, 1903.

CONTRIBUTION BY R. L. LLOYD.

The Editor:

Sir—In reply to the ten questions propounded by the *Journal* concerning raw sulphide smelting, I would answer as follows:

1. *What types of ores are suited to the process?*

Any ore, if the metals to be recovered are gold, silver or copper, or all three, and the location with regard to sulphide favorable. I have used the process in the treatment of dry gold ores, not amenable at that time to amalgamation or cyanide treatment, using barren pyrite as a sulphur-carrying material for a part of the time and for another part of the time using imported copper ores, sometimes in connection with the barren pyrite and sometimes without.

2. *Is hot blast advisable?*

I should say, emphatically, yes, for nearly every reason. It would almost seem to go without question that any heat that can be imparted to the blast by any fuel cheaper than coke, or by utilizing waste heat, would mean of itself a clear gain without taking into account other advantages that are claimed for hot air, such as fast driving, an increased desulphurization, etc. The above remark applies, of course, to conditions where fuel is a matter of necessity; that is to say, on ores which have not a sufficient fuel value of their own to be called 'pyritic' ores. With ores containing a sufficient fuel value of their own, to nearly or quite eliminate the question of fuel even under cold-blast conditions, it would seem to go without saying that heating the blast is a useless expense.

3. *To what extent can fuel be eliminated?*

Entirely dependent upon the character of the ore and fuel used. I believe 3 per cent is as low as has been reported by any hot-blast furnace running continually. With cold blast and suitable ores—that is, ores containing sufficient heavy sulphides—I have run successfully for considerable periods with a fuel charge as low as 6 per cent of the total burden of the furnace; that is, the furnace ran on a charge composed of 94 per cent ore and fluxes and 6 per cent fuel, a good Pennsylvania coke. With Butte ores, and using a considerable amount of coarse concentrate, it is common to run on a charge consisting of 92 per cent ore and fluxes and 8 per cent coke, wet weights.

4. *What amount of copper is needed for the collection of the precious metals?*

A very reasonably close gold collection can be made with no copper. A very good collection can be made with a charge containing 0.3 per cent copper. With very high-grade silver-gold mattes it might be necessary to increase the copper tenor in the charge to 1.5 or 2 per cent. But if the matte is under 100 oz. silver and 30 oz. gold, I know that a charge having a copper tenor of 0.5 per cent, or less, will make close savings. At one time, while using iron matte as a collector for gold and silver—gold being the main value to be saved—and the matte containing traces only of copper, our savings were 97 per cent of the gold. When we commenced to make copper matte, using a charge the copper tenor of which was about 0.3 per cent, and making a very low-grade copper matte carrying about 11 to 15 oz. in gold, our savings increased to 100.5 per cent. This can be accounted for by inaccuracies in assaying, the gold existing as telluride, and partially because the fluxes used carried small gold values. Without changing the copper tenor of the charge, we very often ran as high as 40 oz. gold per ton of matte, without in the least affecting the cleanness of the slags. In the case above noted, the silver and gold were about equal in quantity; that is, ounce for ounce. As a matter of interest, I will say that we made, also, at this time some very clean slags in reverberatory furnaces, smelting flue-dust from the blast furnaces, which flue-dust was made mainly by concentrate from a large gold mill, which was at one time our main source of sulphur. The resultant matte from the reverberatories went about 6 oz. in gold, and the slags from this would run from traces to 20 c. per ton in gold. The matte from these reverberatories went about 1 per cent copper.

5. *What percentage of lime is necessary to a clean slag?* This is dependent entirely upon the presence of other bases, notably magnesia. Magnesia, to a certain point, is not objectionable, and, where metallic bases are not to be had, it is very desirable rather than to carry an excessively limy slag. We used magnesia at one time as a very important part of our slag, metallic bases being unavailable, and dolomite being on the premises. We ran from 10 to 14 per cent magnesia in our slags, which were, counting alumina as an acid, bisilicates. There seemed to be a distinct relation between the percentages of FeO and MgO in the slag, and with the FeO available we were not able to carry over 14 per cent magnesia successfully. I am inclined, however, to believe that if the FeO could have been increased, the MgO would have stood a corresponding increase. I have made clean slags containing 5 per cent CaO. These slags were quite aluminous. I have also seen some very reasonably clean slags containing very little, or no, lime; in fact, in the early history of blast-furnace smelting at Butte, for a number of years our blast-furnace charges were low enough in silica and high enough in metallic bases to require no outside flux, and while our slags

were not as clean then as slags are made at this time, they were reasonably so, and could have been made cleaner, if we had used large forehearths, such as are now commonly used.

6. *What percentage of zinc in the charge can be treated profitably?*

A proportion of 10 per cent in the slag is usually considered the maximum ZnO for fast driving; but this, and the percentage of other objectionable acids or bases, can be varied materially, according to conditions. For instance, a coarse charge giving a high matte-fall can be driven faster on a slag containing more ZnO, MgO, or other undesirable acid or base, than a charge which is fine and giving a low matte-fall. My personal experience with zinky slags has been more through reverberatory practice than blast-furnace practice, and I have noted that zinky slag in a reverberatory, that is, a slag exceeding 10 per cent ZnO, while it never becomes as liquid as a slag containing an equivalent amount of silica, but with FeO as practically its own base, still smelts at quite as low a temperature, and, as far as copper is concerned, is quite as clean. In fact, I am acquainted with the practice at one particular smelter that habitually makes zinky slags, which though higher in SiO_2 than the slags of the neighboring smelters, smelts at a lower temperature than the slags which have FeO as their only base, calling Al_2O_3 an acid in each case. I never was able to get pyrometer readings to check this observation, but there was a very apparent difference in the heats used at these works and those used by its neighbors. The coal per ton of material smelted was also lower at these works than at neighboring smelters. From the observation above noted, I should say that ZnO has not so much the effect of lessening the fusing point as of thickening the slag.

7. *What is the degree of desulphurization attainable?*

This is dependent upon the original sulphur tenor of the ore. With Butte ores and cold blast, 70 to 80 per cent desulphurization is readily obtainable.

8. *What are the possibilities as to capacity of furnace?*

This is dependent upon the condition of the charge with regard to coarseness, fineness and analysis, percentage of matte-fall, quality of fuel, etc. The large furnaces, up to date, with very few exceptions, do not seem to drive as fast per square foot of hearth area as the smaller furnaces. This also is true of reverberatory furnaces. The larger furnaces make their savings by large tonnages, decreased labor handling successfully more difficult slags and with less fuel. Seven to 7.5 tons of burden smelted per square foot of hearth area I consider very reasonable work, but a tonnage considerably in excess of this has been at times reported.

9. *What are the limitations of the process?*

Where gold and silver are the metals aimed at, the process, of course, would be limited to the point where chlorination, amalgamation, cyaniding or other wet processes could compete, after the differences between tailing losses and smelter losses had been charged against milling. Where copper is the metal to be won, the process is limited only by location and costs, which, in turn, depend greatly upon the analyses of the ores.

10. *What is the relative economy as compared to rival processes?*

Where the process is suitable it has no successful rival.

I may add here that, in the above, the word "we" may refer to any of the several smelters with which I have been connected in the past twelve years.

R. L. Lloyd.

Cananea, Mexico, Nov. 13, 1903.

CONTRIBUTION BY W. T. KEITH.

The Editor:

Sir—In the recent articles on 'Raw Sulphide Smelting,' published in *The Engineering and Mining Journal*, the writers have dealt with the percentage of copper on the charge required to collect the gold and silver values of the ore. Though the writers differ somewhat as to the requisite quantity, the experiences of each are interesting reading, but I consider that they would be more instructive if they had given some idea of the grade of matte produced.

Does Mr. Lloyd wish to convey the idea that with 0.5 per cent copper on the charge he can produce, say, 50 per cent matte in one concentration and collect 97 per cent of his gold values, or is he going to produce a 5, 10 or 20 per cent matte?

I would like to ask some of the contributors to the discussion what percentage of loss in their gold value they would figure on making if they were concentrating an ore carrying, say, 1 per cent copper, 0.5 oz. silver and 0.12 oz. gold into a 50 per cent matte in one concentration?

W. T. KEITH.

Greenwood, B. C., Dec. 14, 1903.

CONTRIBUTION BY HERBERT LANG.

The Editor:

Sir—It is with pleasure that I add my mite to the discussion on sulphide smelting that is proceeding in your columns, and which I have followed with interest and profit. Let me say at first that I am in accord with your endeavors to find a more suitable designation for this art than the somewhat unfortunate one under which it now labors, and if the usage of the last ten years has not fixed it too firmly in the popular mind to allow the substitution of a better, I consider that your attempts are well-timed and worthy of support. The name 'sulphide' smelting has always seemed to me to be the most suitable designation, it having been used first, as far as I know, by Mr. W. L. Austin some ten years ago. A year or two later I used it in my little book, 'Matte Smelting,' but expressed a fear then that the other name 'pyritic' smelting was too firmly fixed to permit of a change.

While dealing with the terminology of the science, I desire to suggest the use, among copper- and lead-smelters, of the word 'stock,' so much in vogue among iron metallurgists. We are in the habit of using such expressions as furnace mixture, furnace charge, or ore mixture, to express the aggregate of what is put into the furnace. The one word 'stock,' which to me has a fine old English ring, covers the ground much better. I have never noticed its use in this connection, in other than iron metallurgy; so, considering the scope of the word and the accuracy of its meaning, I think it would be a useful addition to this branch of the science.

1. *What types of ores are suited to the process?*

If I were to be permitted, I would modify this question to read, What kind of ore mixture is best suited to the process? For it makes no difference what the ores are, provided that their values are in gold, silver or copper, and that they do not contain lead which it is an object to save. One ore is as well suited to sulphide smelting as another, provided that it be smelted along with other ores or fluxes which make up its deficiencies as a builder of slag or matte and a furnisher of heat. It is, therefore, a question, not of the composition of individual ores, but of the mixture of ores and fluxes which we get together and throw into the furnace. While an unlimited number of ores and mixtures can be very well smelted in the sulphide furnace, the mixture best suited to the process would consist of iron pyrite, copper in some form, it matters not what; silica, at least partly in the free state, and lime in the form of calcite. Each of these substances may be replaced by many others without much detriment to the practical work of the smelter, but there are reasons why these particular minerals are the most favorable. Thus, iron pyrite may be replaced by pyrrhotite, the magnetic iron pyrite; but the former, while doubtless containing less iron, which is the combustible *par excellence* of this process, discharges its sulphur with more ease, thus getting rid of a good proportion of its weight with but small expense to the smelter for fuel and blast. Calcite, again, while containing much lime, which is extremely useful inside the furnace in a variety of ways, still contains much carbonic acid, which, as every one knows, is easily gotten rid of by merely heating, and, like sulphur, has no noticeable effect in increasing the expenditure of fuel. The volatile ingredients of the smelting mixture, which pass out of the smokestack instead of entering the slag and requiring an expenditure of fuel to fuse and of labor to remove, are in one sense a help to a process instead of a detriment. Among these is water, and particularly arsenic, both of which are evolved with ease. Aside from its tendency to cause losses of the precious metals, arsenic, so long a bugbear to other processes, may be in time regarded as a favorite for the sulphide process, on account of this easy volatility. I am not certain that in the list of desirable substances for this method I should not put mispickel at the head, in place of iron pyrite, if it be a question merely of the economical elimination of ore constituents.

Without detriment to the results, there may be oxidized iron and manganese compounds in the stock, as well as any kind of silicate minerals that do not contain undesirable bases. It is not necessary, as some think, that the stock should contain a preponderating amount of sulphides with merely enough silica to flux the iron that may be burned. We do not, indeed, feel the advantages of the process until we have occasion to smelt stock which, containing but a moderate amount of pyrite, is so constituted that a high concentration is practicable. In such a case we may perhaps gain more by the high concentration than we lose by having to apply more coke fuel. This feature was first brought out by Mr. Austin in the early days of the process, but does not seem, even now, to be fully appreciated by those who practise it.

Under all circumstances (as I believe) the sulphide-burning furnace makes a slag in which all the lime, excepting that small part that may become dissolved in the slag as sulphide, combines with the silica as bi-silicate. The lime, apparently, is first taken up by the silica to the exclusion of the metallic bases—iron, zinc and manganese oxides—leaving these to be thereafter satisfied so far as the silica goes. When lime exists in stock containing iron only as sulphide, the proportionate quantity of silica requisite to form the bi-silicate is first united with the lime, when the remainder of the silica, aided by heat and oxygen, decomposes enough of the iron sulphide to produce the right quantity of iron oxide to flux the remaining silica and no more. The residue of iron sulphide will remain unaltered, and will enter the matte. It does not appear that any of the native silicate minerals which we find in ores affect the iron sulphide during the smelting, and it would probably be useless to add them to the stock in the expectation of their assisting the vital reaction of decomposing the melted sulphide. They may, of course, be smelted incidentally to the process without assisting in its characteristic reactions. This is true, I believe, of garnet and hornblende, and of the alumina silicates, many of which are so common in ores, particularly in those of copper.

2. *Is hot blast advisable?*

My later experience is altogether favorable to the hot blast. I believe that every matting blast-furnace plant should be, and in time will be, supplied with this useful auxiliary. It is true that hot blast complicates the plant, increasing its initial cost and the expense of running; but I believe that it invariably decreases the cost of treating the ore. At least, I do not know of a single plant where it has not done so when used. When the pyritic process was first advocated, many mistaken ideas were prevalent as to the functions of the hot blast. Some maintained that the process could not go on without it, holding that the previous heating of the air was a *sine qua non.* It is said that I was the first to demonstrate that there was such a thing as cold-blast pyritic smelting, which is something that we see going on every day now. This unimportant fact, which I hope to be excused for citing, does not blind me to the advantages of heating the blast, which I have long been in the habit of advocating. As I see it, the heating of the blast assists in different ways. First, it saves fuel (question 3); this cannot be denied by even the most hidebound of our opponents—if we have any opponents at this late day. Second, it makes the furnace drive faster and smelt more. This may be, and has been, disputed, but I maintain it in the face of the assertions to the contrary, which come from high sources. In my personal experience such has always been the case. I have studied the process closely for a dozen years and more, and have never missed an opportunity to investigate its peculiarities; and I repeat that, under all circumstances with which I am familiar, the hot blast has increased the rate of driving very materially. It is natural that it should do so, for, as less fuel is required, the lessened amount will be consumed quicker, and will permit the descent of the charges much faster, this resulting directly in increasing the capacity of the furnace. It takes a certain amount of time and of air to burn coke, and the more coke the more time and air. This is demonstration enough for the ordinary man, and when I add to this the observed fact that the furnaces that I have known actually ran faster with hot blast than with cold, I feel that I have had my say.

At a time when metallurgists had at command no other apparatus for blast heating than the cast-iron pipe-stoves of a past era, hot blasts were not so often advisable. The stoves of which I speak were adopted bodily from the iron regions of the East, and had three defects, which rendered them ineffective to a great degree. They were extravagant in first cost; they impeded the blast; and, finally, they were very inefficient in the matter of heat transference. Stoves of this pattern are known to impede the blast in its passage to such an extent that the pressure at the blower commonly registers from 6 to 16 oz. more than at the tuyeres, showing a great loss in power and pressure. The cause is that the passages are much too tortuous and constricted. The apparatus is usually gotten up on the original plans of some extinct iron works, without any alteration to meet the much lower pressures used in copper smelting. I do not know that any one has ever executed tests showing the heat applied to and returned by such stoves, so I cannot speak from accurate knowledge; but I do not imagine that any such stove has an efficiency of more than 50 per cent, if as much. With such apparatus as this, it is no wonder that operators have sometimes felt that the hot blast was not always advisable. I notice from the lately published reports of the operations at Mt. Lyell that the stoves there, when heating the blast to 500° F., had an efficiency of 49 per cent, according to my own estimates, based on their data, and assuming an average calorific power for the wood with which they are heated; and that they retarded the blast to such an extent that the pressure at the furnaces was 6 oz. per sq. in. below that at the blowers, thus consuming about one-sixth of the power applied to the latter. Surely, there is no reason for the tone of gratulation in which interested parties speak of the performances of these particular stoves.

3. *To what extent can fuel be eliminated?*

The saving of fuel in the process under consideration varies much with different charges and conditions. Unless we use the hot blast, there is no saving of fuel in particular. With it, we save from one-half to two-thirds of what would be used without it. Thus, if the charge required 10 per cent of coke to smelt with the cold blast, it would probably require from 3 to 5 per cent with the heated blast. This would depend somewhat upon the composition of the charge, and especially as to its contents in sulphides, which furnish the principal fuel. One smelter that I know of is regularly treating sulphide mixtures with 6 per cent of coke with cold blast. While its stock is well adapted to hot-blast smelting, it is not possible to save much fuel by its aid, the present consumption being so small. In all such cases where, with cold blast, a low fuel consumption is claimed, we must expect to find a quite low degree of concentration. Such is the case at the works of which I speak. Cases are known where no carbonaceous fuel at all is used, the whole supply of heat arising from the combustion of sulphides and from the hot air blown in. The latter source of heat may also be eliminated, it is claimed, and the sulphides relied upon solely. It is said that at Mt. Lyell, where for years they heated the blast to some 500° and a little more, the tendency now is to blow harder and heat the blast less, and rely almost entirely on the combustion of the sulphides for the heat. We cannot tell to what these new experiments may lead; but it is hardly in the line of economy to spend much money on a heavier blast without first inquiring if the useful effects of a hotter and lighter one may not be preferable and had at a cheaper rate.

4. *What amount of copper is needed for the collection of the precious metals?*

This question refers, I suppose, to the quality of the matte which is produced. The aggregate amount of copper in the ore would not be a pertinent matter. It is the richness of the matte that tells the story. I have always found a matte of from 30 to 50 per cent of copper to give the best results in a high percentage of extraction of gold and silver. Others have had a different experience, and some even claim that copper is unnecessary in the matte. My impression is that, if the slag accompanying the matte be largely composed of lime or magnesia, the iron matte without copper will serve as a collector; but that with a heavy iron slag, such as falls when the charge is high in pyrite, a coppery matte is essential. It really requires but a very little matte to extract the values if it be of the right sort; nor have I found a large proportion of matte, of whatever composition, to have any effect in reducing the slag losses. A singular notion is prevalent among several metallurgists who have addressed you of late. They appear to think that successful smelting cannot be done unless a certain proportion of matte be made, some putting the quantity as high as 10 or 15 per cent. One says that the tuyeres get dark; another, that the hearth fills up, etc., etc. The small proportion of matte, indeed, is not without its influence on the working of the furnace, but even if the matte-fall ceases entirely the smelting will proceed, perhaps unchecked. Let me ask such operators if they can smelt charges of slag alone, where no matte is produced; or if they have ever tried to handle mixtures giving a concentration as high as 20 and even 40 to one.

Question 5, which asks what percentage of lime is necessary to a clean slag, leads naturally to the wider question of the influence of lime on slag losses. Lime is exceedingly beneficial in the furnace for three reasons at least, which I may dwell upon. In the first place, the lime-slags of the composition indicated before, work extremely well in the blast-furnace. They run very nicely, with a smooth, oily flow, do not sputter, and keep the furnace and its accessories in first-rate condition. They are light and reasonably fluid, and allow the matte globules to settle out most perfectly. Any, even the lightest, iron matte will settle from high lime slags. It seems to me, when I consider the behavior of these admirable substances, that a pure bi-silicate of lime would have no action at all upon entangled sulphides. The expression "clean slag" is a relative one. There is no such thing as a slag entirely free from gold, silver and copper; but I think that the high-lime slags come nearest to absolute cleanness. Of all such slags, those containing nearly an equal amount of bi-silicate of lime and singulo-silicate of iron behave the best, all things considered, and will be as clean as we can get. Lime is not often a constituent of sulphide ores, so that we can hardly expect it to enter largely into slag composition in this form of smelting; nor is it entirely necessary that it should. Quite acceptable slags are made with iron oxide as almost the sole base. I have made such, and, regarding their freedom from valuable metals, I recall that the average proportion of copper carried off by them at one works was nearly 0.5 per cent, which is probably twice as much as a well-constituted lime-iron slag would have shown. However, under the existing circumstances, this result was good. It would not have paid to add lime to such a slag unless it could have been secured in the form of a paying ore, which at that locality was impossible. Speaking generally, and without reference to particular localities, I should say that it would never pay to add lime to pyritic charges when iron in any form can be had at less cost. On the other hand, it would never pay to add iron oxide as flux when lime ores can be had at less cost, provided that the percentage of lime does not surpass 30 or thereabouts; for a slag containing much above this percentage will not melt properly in the cupola.

The degree of desulphurization in the furnace, depending as it does upon a large number of factors, is very variable. When the stock consists mainly of heavy pyritous ore, where it is not practicable to flux highly with silica, the proportion of sulphur expelled will be small, perhaps not more than 60 or 70 per cent. When the ores contain any other bases than iron, and especially when they contain much lime (and probably magnesia), mingled proportionally with silica, the combustion of sulphides is more extensive, and frequently ascends to 80 per cent and more. When treating charges of mixed ores, oxides and sulphides with a sulphur content of 8 or 10, I noticed that 75 per cent of the sulphur was expelled. Mixtures carrying 26 to 28 per cent gave the same result, but would have lost more had the fluxes been suitable for the work. On another occasion, when smelting a mixture containing some 11 per cent sulphur, the percentage driven off reached the extraordinary figure of 96 per cent. The result in this case was due, in my opinion, to the two concurrent facts that the ore carried nearly as much arsenic as sulphur, while the flux consisted of a large proportion of lime, which is observed to influence the expulsion of sulphur —and hence concentration of the product—in a very favorable manner. The arsenic, as I observed, was largely sublimed in the form of a sulphide, and to this I mainly ascribe the good results obtained during the smelting. Others have noticed that at certain times, depending on the condition of the furnace, all the sulphur may be burned off from charges low in that element.

10. *What is the relative economy as compared to rival processes?*

This question I find most difficult to answer. In fact, I do not think that it can be satisfactorily answered in general terms. It would be necessary to study the conditions that surround each individual case in order to form a judgment as to the applicability or economy of this or that process. As I have heretofore said, there are no ores whatever of silver, gold or copper which may not be treated by sulphide smelting if the stock be proportioned right, but there are thousands of cases where some other process may prove more economical. In the matter of general availability, there is no doubt that sulphide smelting has a great advantage over all others. Again, it is advantageous in that no process requires so little preliminary preparation of the stock. This fact makes, of course, for economy in running, and particularly for the general usefulness of the process. The plant itself is cheap and simple, and if well constructed is at least as reliable in its work as the lead process or any other well known method. I have noticed, however, a tendency to put up cheap and trivial plants, which is much to be regretted, as it will inevitably tend to retard the development of the process.

In considering the merits of a process, we must not blind ourselves to its disadvantages. The principal drawback of sulphide smelting, by which its economy is much lessened as compared with other methods, lies in its production of a crude and comparatively unfinished end-product, which is not only heavy and burdensome to handle and transport, but requires costly and complex operations to bring into useful form. The same is true of the ordinary plain matting processes. As compared with these, such methods as chlorination and amalgamation, whose end-products are saleable at the mine, have an advantage which must not be overlooked. As regards the cost of treatment, now one, now another process has the advantage; but, as a matter of fact, the improvements of late years have brought down the cost in all the standard processes to so low a pitch that there is often not enough difference between them to incline the judgment either way. It would seem that nobody is building plants nowadays with the expectation of their costing over $3 or so per ton for treating the ore or stock. I speak now of the larger establishments which are springing up so numerously on our western hillsides. In forming our ideas in the matter of costs of treatment, we must not omit to discriminate between the amount and character of the preliminary preparation required to fit the ore to the process to be applied, and which frequently makes quite an item to be added to the total of expenses, although in this comparison I have not considered it as belonging strictly to the process. Thus roasting is in no sense a part of the lead-smelting process, but is sometimes a necessary preliminary thereto, as it is to chlorination in most of, if not quite all, cases, and to cyanidation in many.

Numerous instances have been cited in print where the cost of treatment has been brought as low as $2 per ton, and in the case of matting and stamp amalgamation to $1. What wonderful progress these facts indicate! I well remember the time when men went broke trying to smelt for $50 and $100 per ton.

HERBERT LANG.

Oakland, Cal., Dec. 7, 1903.

CONTRIBUTION BY L. D. GODSHALL.

The Editor:

Sir—The discussion of this subject by different metallurgists cannot help but be of importance equal to, if not greater than, the recent interesting and valuable discussions on mine sampling and ore deposition.

The subject at the present time seems to be in a condition somewhat similar to that of the cyanide process at the time of its early introduction into this country. During the last ten years much has been written upon the subject of pyritic smelting. Many plants have been put in operation and a record of failures has resulted, fully equal to that of the early attempts at cyanidation. Until quite recently it has been very difficult to obtain reliable information of actual results secured in practical work. Most extravagant claims frequently have been made, and are still forthcoming, concerning the superiority of this method of smelting over any other. On the other hand, important improvements and discoveries also have been made during recent years.

We have, therefore, on account of these advances and in spite of many failures, every reason to hope that in the near future matte smelting, together with its modification, pyritic smelting, will become as important in fire metallurgy as cyanidation is now in the wet treatment of ores.

In order to bring out as full a discussion on this subject as possible I shall criticize rather freely such statements and claims as in my opinion are either not borne out by experience or else not sufficiently clearly presented to be correctly understood. I shall also call attention to some statements and results obtained which, in my opinion, are of the greatest importance. Certain statements and data have been taken from a publication or catalogue issued by the Colorado Iron Works, of Denver, the accuracy of which has, however, been vouched for by Mr. J. W. Nesmith, president of the company.

In this connection I wish to say that I am seeking for reliable information, no matter whether obtained from practical furnace foremen, who have never seen the inside of a technical institution; reliable manufacturers of smelting machinery, perhaps equally devoid of technical training, but constantly in touch with metallurgists and smelting men in many different localities and working under as many different conditions; highly educated but practically inexperienced professors of metallurgy or practically experienced and technically trained metallurgists.

Your editorial in the issue of October 10 defines the above subject as follows: "Raw sulphide smelting may be defined as the treatment of unroasted sulphide ores with a view to the formation of a matte." And "Raw sulphide smelting is based upon the utilization of the heat derived from the combustion of sulphide ores in a blast furnace *in contradistinction* to the heat obtained directly from carbonaceous fuel." (The italics are mine.)

You appear to object to the well-known terms pyritic smelting and matte smelting, claiming that these terms do not cover the ground. The former is said to omit pyrrhotite ores and the latter to include roasted ores. These objections, it seems to me, are due altogether to the view one takes of the scope of the terms used. As I understand the term 'pyritic' smelting, pyrrhotite ores are not omitted, while the fact that matte smelting does include the treatment of roasted ores should be in its favor, as being more comprehensive.

Mr. Ingalls notes some objections to the term 'raw sulphide' smelting. These I consider well taken. In my opinion, however, he has omitted to state some of the strongest objections. Among these are the conditions governing the smelting of various ores where the contents in sulphur are so low that it becomes necessary to maintain a reducing atmosphere in smelting, in order to prevent the small amount of sulphur present from being oxidized and to obtain the copper in a matte form, and at the same time to produce enough of the latter to collect the gold and silver values present in the ore.

If the definition and basis of utilization of heat be accepted as above given, we can at this time have only a modification of 'raw sulphide' smelting in America, because at no place, so far as I know, is carbonaceous fuel entirely eliminated.

In order to induce further discussion on this point, therefore, I make the following suggestions: To stick to our old terms of 'lead smelting,' 'copper smelting,' and 'matte smelting.' We are all familiar with the many different variations in the processes of the two former, and let us accept and define those of the latter. It is true that copper smelting gradually merges into matte smelting, but the latter covers more cases universally regarded as distinct from copper smelting than either 'pyritic' or 'raw sulphide' smelting.

I would suggest that the definition and basis be somewhat changed to read as follows: *Matte smelting may be defined as the treatment of various ores with a view to collecting the different values by means of a matte produced;* and, *Matte smelting is based upon the utilization of the heat derived from combustion of sulphide ores in a blast furnace, in addition to the heat obtained directly from carbonaceous fuel.* The basis, as here stated, therefore, includes such variations as will result from the minimum of heat derived from the combustion of sulphide ores in the case of roasted ores, and raw ores very low in sulphur contents, to the maximum of heat derived from such combustion in the case of heavy raw sulphide ores, as smelted at Mt. Lyell, Salt Lake, and other places.

Pyritic smelting now becomes a branch or part of matte smelting, and may be defined as follows: *Pyritic smelting is a form of matte smelting whereby the maximum amount of heat is derived from the combustion of heavy raw sulphide ore in a blast furnace, in addition to the minimum amount of necessary heat obtained directly from carbonaceous fuel.*

All types of ore, suitable for copper smelting, are equally so for matte smelting, with the possible exception of high-grade oxidized ores of copper under conditions suitable for direct smelting to black copper. In addition to the above, all kinds of dry ores, together with ores carrying a limited amount of lead and zinc, the limit of the former being a question of commercial value of the lead lost as compared with the cheaper reduction of the other metals contained in the ore.

The limit of the zinc ought to be such that, when mixed with the charge, the normal amount of ZnO contained in the slag shall be about 8 per cent, with a maximum of 10 per cent. Higher zinc slags have been run, according to Mr. Ingalls, but in the opinion of the writer any increase on the above percentages will bring many difficulties in connection with the operation of the furnace, in addition to a largely decreased capacity resulting therefrom.

The amount of sulphur on the charge may vary from the maximum amount present with the iron as FeS_2 to a minimum of 3 per cent, provided the percentage of copper remains slightly less.

So far as I know there are to-day no slags made by matte or pyritic smelters which are not closely or exactly duplicated by the copper smelters. This kind of smelting is therefore not distinct from copper smelting on account of any difference in the slags to be obtained, but it is rather a process using a greater variety of ores and obtaining similar slags with different methods.

The question of the advisability of using a hot blast is a most important one; and, in addition to its importance, there is at the present time also the greatest difference of opinion as to its economy among metallurgists engaged in this branch of smelting. In order to discuss this question intelligently, it is necessary to start from a common understanding of the word "advisable." In my opinion in this connection it must mean that it is more *economical* and therefore more *profitable* to use a hot than a cold blast in smelting.

With the claims and evidence for and against this proposition, it becomes necessary to study closely the conditions present as well as the results obtained in connection with both cases.

Very strong claims have been made by hot-blast advocates concerning its superiority over cold blast. Koch[1] claims that a blast of 200° C. produced cleaner slags with wider range and smoother work. Mathewson[2] claims that hot blast will produce higher grade matte, or, in other words, that it will oxidize more sulphur than its equivalent of cold blast. Austin[3] claims it is advantageous in smelting a silicious or infusible charge. Bretherton[4] cites many advantages, among them, increased capacity of furnace, saving of fuel, brighter tuyeres, smoother running of furnace, largely increased limit of zinc and arsenic on charge by reason of its stronger oxidizing action, and concludes with the statement that hot blast reduces the cost of smelting nearly half on heavy sulphide ore. Lloyd[5] claims increased capacity, stronger oxidizing action, and that it is cheaper to heat the blast before it enters the furnace than to heat it in the furnace with coke. J. W. Nes-

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[1] See page 24. [2] See page 21. [3] See page 17. [4] See page 28. [5] See page 45.

mith, as a manufacturer of blast furnaces, claims two points in favor of the hot blast: A saving of at least 10 per cent in the cost of the fuel used per ton of ore smelted, and an increased smelting capacity of at least 25 per cent.

The claims above enumerated in favor of the hot blast cannot be ignored. If only a part of them can be substantiated, the value of the hot blast must be admitted.

The evidence on the opposite side is mainly of a negative character, and consists principally in a failure on the part of metallurgists of note to find the merits claimed for the hot blast. Carpenter[1] says he "has never found the magic in mere hot air that others have professed to find." Pre-eminently the greatest exponent of the hot blast is Sticht in his practical experience at Mt. Lyell, where he is using it with such unqualified success. In contrast with his experience, we have that of Nutting, at the Bingham smelter at Salt Lake, and Carpenter at Deadwood. The former is working under conditions as nearly identical with those at Mt. Lyell as it is possible to find; the latter, at Deadwood, under conditions as widely different. Both use cold blast, and the scope and commercial success of Nutting apparently is on an equality with that at Mt. Lyell. In addition to the practical experience of these two prominent operators, we have that of Butte, where hot blast has been tried without apparent success. An examination of the actual results obtained at different plants using the hot and cold blast now becomes interesting.

In all cases, so far as I know, where heavy raw sulphide ores, low in copper, are smelted without a preliminary roasting, a double smelting is required in order to obtain a matte sufficiently high in copper, say from 45 to 50 per cent, for converter purposes. This applies

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[1] See page 83.

alike to hot- and cold-blast smelting. At Mt. Lyell the grade of the first matte is said not to exceed 15 per cent copper, being a concentration of about seven tons of original ore into one ton of matte. Higher grade first mattes are being produced at other places, but with a lower percentage of sulphur on the charge. At the Bingham plant at Salt Lake, where the character of the charge is similar, but cold blast is used, I have been informed from unofficial sources that the grade of the first matte is also about 15 per cent copper.

The products of the hot and cold blast do not, therefore, seem to show any great difference, from which we may conclude that on certain classes of ore the oxidizing action of the hot blast does not seem to exceed that of the cold blast. On ores containing appreciable quantities of zinc, lead, arsenic or antimony, or on first mattes, where the sulphur is less easily oxidized, more information is necessary before any definite decision on the relative efficiency in oxidizing action between the hot and cold blast can be reached. Bretherton claims to have obtained a higher oxidizing action with the hot blast in connection with zinc sulphide, but gives no figures. He also states that the elimination of arsenic so far has not been successfully accomplished with cold blast.

The last statement requires modification, as it is well known how simple and easy it is to volatilize the arsenic when present as arsenical pyrite. The writer has even added speiss in large quantity and in large pieces to matte smelting charges, using cold blast under oxidizing conditions, without discovering any evidence of the presence of arsenic in the matte produced. The speiss in question had been produced in lead smelting. Practical results obtained in the concentration of first matte by hot- and cold-blast smelting would shed considerable light on the subject. Until such information is forthcoming, it seems as if the greater oxidizing action claimed for the hot blast is yet unproved.

The comparative furnace capacity is difficult to determine. We have so many cases of phenomenal capacity obtained with the cold blast, both in copper and in matte smelting, that it is difficult to imagine any greater efficiency with the aid of the hot blast. Mr. Paul Johnson, the pioneer in copper and matte smelting in British Columbia, obtained, at Nelson and in the Boundary district, results in capacity with the cold blast not excelled at any place with the hot blast. Mr. Robert Hedley, who succeeded Mr. Johnson at Nelson, has been equally successful with cold-blast work. The slags made there were, moreover, what are usually termed highly silicious, containing about 42 per cent silica, and from 15 to 20 per cent alumina, the charge containing but from 2 to 5 per cent copper, besides some silver and gold. The matte produced in one smelting assayed from 45 to 50 per cent copper, the slags from 0.2 to 0.3 per cent in copper, and as clean from silver and gold as in lower concentration. The capacity of the furnace was frequently in excess of 300 tons per day and seldom below 250 tons.

At Butte and Great Falls, as well as at other places, capacities with the cold blast are being obtained which are not exceeded anywhere by the hot blast.

According to recent reports, Mr. Johnson is now putting in hot blast in a new smelter he is erecting on the northwest coast, where the conditions are somewhat different from those where he formerly operated in British Columbia.

Mr. J. W. Nesmith gives the results of a test run made at Silverton, Colo., at the matte smelter formerly owned by Thomas F. Walsh, the metallurgist in charge being E. W. Walter.

The following are the results:

Cold blast.			Hot blast, 700° F.
90 hours.	Duration of test		72 hours.
104 tons..	Charges per 24 hours		150 tons.
300 lb....	Coke used per charge		150 lb.
7 into 1	Degree of concentration		7 into 1.
$109.20....	Cost of coke	$78.75	
	Cost of heating hot blast	34.50	
			$113.25

The above figures are of great value as illustrating three important points: Capacity, fuel required in furnace, and oxidation of sulphur. The smelting capacity is so clearly in favor of the hot blast for this particular charge that the inference naturally follows that there may be many other cases where such capacity must be equally favored by the aid of it, but it does not necessarily follow that all ores will show such a variation between hot- and cold-blast smelting.

Economy in fuel is as important a factor as capacity in smelting. The saving in coke brought about by the external heating of the blast must exceed the actual expense of such external heating in order to become an economical advantage. Some writers speak of such saving of coke as self-evident, but it must be remembered that in heating the blast inside of the furnace by means of coke alone, all of the heat contained in the latter is utilized. In the case of the external application of heat to the blast, cheaper fuel is available; but, on the other hand, heavy losses in heat occur by radiation and otherwise.

Where fuel cheap in comparison with coke can be obtained, and the mechanical appliances are so perfected as to utilize the greatest possible amount of heat contained in such fuel, there ought to be no question of the economy of the hot blast so far as the fuel question is concerned. On the other hand, where the difference between the cost of coke and other fuels is not so marked, the economy, in fuel, of the externally heated blast is very materially lessened, and may in some cases become doubtful.

The experience at the Silverton smelter, as shown by the above data, leaves no doubt in that case as to the economy of the hot blast. In that test, using cold blast, 390 charges were smelted at a total cost for fuel of $109.20, while with the hot blast, 450 charges were smelted at a cost of $113.25. The weight of the charge is not given, but, judging from the weight of charges given in connection with other data, it probably was from 2,000 to 2,500 lb. It will therefore be readily seen that in this case the total fuel required for the hot-blast test cost but $4 more than that for the cold blast, and for this extra expenditure of $4 more than 60 tons of additional charge were smelted. The importance of the proved advantage of the hot blast in some instances should make the subject of sufficient interest to all metallurgists to give it their most careful consideration and study; for that which to-day is a success, under certain limited conditions, may to-morrow be improved and its use widely extended.

The consumption of coke in hot- or cold-blast smelting varies according to the amount of sulphur present in the charge. The following figures are given as representing the approximate percentages used under different conditions:

	Per cent.
Mt. Lyell, with hot blast and double smelting; coke used per ton ore smelted	6
Bingham, Utah, with cold blast and double smelting; estimated coke consumption on ore smelted	10
Butte, Mont., single smelting, with cold blast	10 to 11
British Columbia, single smelting, with cold blast and raw ore low in copper and sulphur	12 to 13

In all of the above cases, the grade of the matte is estimated to be on a basis of 45 to 50 per cent copper, and the percentage of coke is based on ore smelted.

In an article entitled 'Smelting at Mt. Lyell, Tasmania,' Mr. Alfred Miller makes the statement that coke had recently been entirely eliminated in the raw ore smelting at Mt. Lyell, and in addition to the above innovation the hot blast, which had been found in previous experience to be "obligatory and not to be circumvented," had been lowered in temperature, and that in No. 1 plant both ore and matte were now being smelted with cold blast. This marks such a change and improvement over past experience that additional information on the subject would prove of very great interest.

The relative economy of the different methods of matte smelting at present in use is a most difficult question to determine.

The competing methods adapted to the smelting of sulphide ores carrying a small percentage of copper are as follows:

1. The hot blast, double-smelting process for raw ores.
2. The cold blast, double-smelting process for raw ores.
3. The cold blast, single-smelting process, requiring a preliminary roasting of all or part of the heavy sulphide ore treated.
4. The reverberatory process, requiring a preliminary roasting.

According to some of the literature on pyritic smelting there is only one way to smelt heavy pyritic ores, and the expense of that method is so extremely low that no other method can possibly compare with it. To show the fallacy of such claims without further argument it is merely necessary to point to two notable instances: The Highland Boy smelter at Salt Lake, treating heavy pyritic ore, containing a low percentage of copper and very small quantities of gold and silver; and the Argo smelter at Denver, treating heavy pyritic ores and concentrates low in copper, but frequently high in gold and silver. The fact that the latter stands to-day as the result of development, from the old Welsh process which was crude and primitive when first introduced into this country by Mr. Richard Pearce, while the former is a modern reverberatory furnace plant erected but a few years and designed by a noted copper metallurgist, denotes that reverberatory practice is still a healthy rival of the blast furnace methods.

The long-continued success of both also gives weight to the argument. The important discoveries and developments made in connection with, and bearing upon, the different methods of matte smelting may be summed up as follows:

1. Direct smelting of heavy raw sulphide ores, with either hot or cold blast, and oxidizing a sufficient amount of sulphur to provide the necessary iron oxide for fluxing purposes.

2. Concentration of copper in matte so produced, by a second direct smelting to approximately 50 per cent copper, thereby rendering it suitable for converting without any previous roasting.

3. Important improvements and developments in mechanical roasting furnaces, whereby the cost of roasting has been greatly reduced, thereby lessening the cost of copper smelting and matte smelting as distinguished from pyritic smelting.

4. Important improvements and developments in water concentration, greatly reducing the losses formerly sustained, thereby lessening the advantages of direct smelting of the crude ore.

5. The successful application of the hot blast with its decreased amount of coke required for smelting.

It is not likely that the accuracy of the above statements will be questioned by any of the metallurgists, however widely they may differ on many other points. The first four statements are self-evident; the fifth, without any further explanation, might be objected to as not stating the whole proposition, because it sets aside, for the moment, the question of the relative economy of hot, as against cold, blast. Those who deny the economy of the hot blast are, however, forced to admit that it is practical. For this reason the question of economy is omitted from the statement.

In determining the question of the relative economy of the different methods, it is always necessary to take into consideration the character of the ore, both chemical and physical, the cost of supplies and the cost and character of fuel and labor.

Some of the writers on this subject make a great point of the advantage of direct smelting without roasting, but neglect to take into consideration the cost and disadvantage of a second smelting of the first matte produced. They also overlook the improvements recently made in modern mechanical roasting furnaces, whereby the heat from the burning sulphur is utilized and fuel almost eliminated, reducing the cost of roasting to a minimum. In many instances there are very distinct advantages to be derived in the single smelting preceded by the roasting of heavy sulphide ores. Where copper ores are scarce, but necessary for making clean slag, a preliminary roasting will produce a much higher concentration and therefore higher grade copper matte, which means a correspondingly smaller amount of copper on the charge. Where copper ores are brought from a distance and smelted at a loss merely for the sake of the copper, to produce clean slags, such difference might change failure into success. Where appreciable quantities of zinc sulphide accompany the ore a preliminary roasting will be of the very highest importance, as pointed out by Mr. Ingalls. Zinc-blende is one of the most difficult minerals to oxidize, and if put in the furnace raw, in large quantities, it will check the smelting capacity, form crusts of sublimated zinc sulphide along the furnace walls and derange the operation of the furnace generally. Some will be oxidized and enter the slag as ZnO, and the balance will be found distributed partly in the slag and partly in the matte, in both cases in a dissolved state, and still another portion will be found floating away in the slag like mushy ice or snow in water. Under such conditions clean slags are next to impossible to obtain, while with a preliminary roasting most of the zinc will enter the slag as ZnO, producing cleaner slags and avoiding the mechanical troubles of the blast furnace. In addition, high zinc slags invariably require a heavy percentage of iron, and render therefore the usually profitable and always desirable high silica slags an impossibility.

Messrs. Carpenter and Lang both advocate the direct smelting, without concentration, of Butte ores, but if we take into consideration that these ores by analysis show the following composition: SiO_2, 58 per cent; Fe, 12 per cent; S, 17 per cent; Cu, 4 per cent, it becomes apparent at once that such smelting would be exceedingly costly by reason of the excessive amount of fluxes and fuel required, and it seems to the writer that it would be better to first demonstrate the advantage, if there be any, of smelting the present Butte charge raw or without any previous roasting. If it cannot be demonstrated that the low silica and high sulphur, under such conditions, can be smelted more advantageously raw than it is now being done, it would seem to be still more difficult to demonstrate the advantage of smelting, raw, the low-grade mill ore, which is much higher in silica and correspondingly lower in sulphur.

In conclusion the writer wishes to suggest that there is one field not yet entered by matte smelting which, in his opinion, deserves attention; that of smelting dry ores by matte smelting as against lead smelting, in central localities like Denver and Pueblo in Colorado, and the Salt Lake valley in Utah. Matte smelting under such conditions should be carried on by one smelting to produce 45 to 50 per cent copper matte, and all heavy sulphide ores should be roasted. Under such conditions we would have the following advantages as compared with lead smelting:

Slags comparatively as clean as in lead smelting.

High, instead of low, silica slags.

Capacity of matte furnace nearly twice that of lead furnace.

Result, very much lower cost per ton of ore smelted.

L. D. GODSHALL.

Denver, Colo., Dec. 20, 1903.

Contribution by William A. Heywood.

The Editor:

Sir—As a contribution to the discussion on raw sulphide smelting, I give the results of a series of experiments recently made at the plant of the Tennessee Copper Company, using cold blast. The ore treated is a pyrrhotite mixed with schist containing approximately 2.5 per cent Cu, 40 per cent Fe, 25 per cent S, and 25 per cent insoluble. The furnace used was 56 by 180 in. at the tuyeres and 72 by 180 in. at the top, water jackets 15 ft. high. Air was blown through 28 tuyeres of 4 in. diameter. For supplying the blast a Nordberg cross-compound blowing engine was used, and the quantity of air consumed in each experiment was carefully noted. The experiments were made to determine the results of different volumes of air. Only the air factor was changed in each experiment; probably by varying other factors more favorable results, especially a higher rate of concentration, may be obtained:

	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.
Tons ore smelted per day	401	228	267	276	468
Tons quartz smelted per day	50	19	22	23	39
Tons other material per day	74	..	..	..	..
Total charge per day.	525	247	289	299	507
Tons coke used per day	56	5.7	6.7	6.9	11.7
Per cent coke on charge	10.3	2.3	2.3	2.3	2.3
Per cent coke on ore smelted	13.9	2.5	2.5	2.5	2.5
Per cent copper in matte made	40.0	7.7	11.0	8.4	8.1
Per cent copper in slag made	0.43	0.14	0.11	0.12	0.11
Per cent insol. in slag	32.0	41.0	39.2	39.0	39 0
Per cent FeO in slag.	51.0	37.2	40.6	40.0	41.0
Cubic feet free air per minute	16,800	14,400	16,800	19,200	22.080
Height of charge above tuyeres	11 ft.	6.5 ft.	6.5 ft.	6.5 ft.	6.5 ft.

No. 1 is an average daily run, calculated from six months' smelting of heap-roasted ore with quartz, converter slag and some custom matte. No. 2, 3, 4 and 5 are the experimental runs on raw sulphides with quartz and 2.5 per cent coke. With the exception of the blast, all the conditions were kept as nearly the same as possible. The quantity of ore smelted seems to be in proportion to amount of air blown into the furnace. In these experiments the rate of concentration appears to bear no relation to the rate of smelting.

The low-grade matte made in these experiments was re-smelted with converter slag, quartz and 6 per cent coke on the charge, producing a 40 per cent copper matte, which was converted, and a slag containing 28.4 per cent insoluble, 54 per cent FeO, and 0.46 per cent copper.

The furnace ran smoothly throughout the experiments. No difficulty or delay was experienced in changing from the usual roasted ore charge to raw ore or vice versa. During experiment No. 5 the slag and matte were so extremely hot that it was apparent that the coke could be either materially reduced or entirely eliminated.

A great deal has been written on the subject of hot blast, but I have never found any definite comparison published on the treatment of any particular sulphides or class of ores by both hot and cold blast. I am sure it would be of great interest if some metallurgist would give the comparative results, both metallurgical and financial, of both methods of smelting; in other words, to add to the volumes that have been written on what *may be* done with hot blast, a few words on what *has been* done.

WILLIAM A. HEYWOOD.

Copperhill, Tenn., Jan 29, 1904.

Since the above was written the experiments have been continued. Matte containing 12 to 16 per cent Cu has been produced in our later campaigns with a slag containing 40 per cent insol., 40 per cent FeO and 0.15 per cent Cu. The mattes produced show an excess of Fe amounting to 11 to 13 per cent over the Fe required for the formula Cu_2S, FeS. This iron is present as an oxide; both chemical and microscopic examinations of the matte showed no metallic iron.

WILLIAM A. HEYWOOD.

July 28, 1904.

CONTRIBUTION BY G. F. BEARDSLEY.

The Editor:

Sir—'Raw sulphide' smelting, 'matte' smelting, 'pyritic' smelting; I have no wish to quarrel with terms, any of which are not exactly descriptive of the fusion of iron- or copper-bearing sulphides by their rapid oxidation in the blast-furnace under suitable conditions; that is, hot blast and small quantities of coke. To the term pyritic smelting, however, the above definition has become attached, and, I am afraid, cannot be shifted. I will use it as indicated above. To save repetition I will also use iron 'sulphide minerals' as including pyrite, pyrrhotite and chalcopyrite, and 'pyrite' or 'pure pyrite' as a pyritic ore containing 90 per cent $FeS_2 + nCuFeS_2$, with 10 per cent of impurities, consisting of silica 3 to 5 per cent, barite 1 to 2 per cent, alumina 2 to 3 per cent, other constituents 0.5 to 1.5 per cent.

1. *What types of ores are suited to the process?*

Pure pyrite and pure pyrrhotite, pre-eminently. The furnace charge at Mount Lyell averaged 21 per cent silica, 30 per cent iron, 35 per cent sulphur, and 10 per cent of earthy bases, including about 5 per cent alumina. This may be taken as a type, and any composition of ore between this and pure pyrite can be smelted with 3 per cent of coke or less, provided the physical condition of the ore is not against it, that is, fine ore, large lumps of pure silica, or pure schist, or an even blending of the above constituents in the pieces of ore. It is necessary that a certain proportion of the ore be of sufficient purity to start running as a matte at a comparatively low heat in the upper part of the charge. This filtering through the coarse material, oxidizing with the ascending blast, generates heat, at the same time dissolving and fluxing the more infusible portions of the charge. An even blending of 20 per cent silica plus 10 per cent of bases, and equal to about one-third of the bulk or weight of the ore-lumps, will seriously retard this effect.

2. *Is hot blast advisable?*

I think it advisable, and in some cases necessary. It is advisable when the blast can be heated by waste gases. It is necessary where it is desirable to force as much silica into the slag as possible. At Mount Lyell, as the charge column was gradually raised and the blast increased, the stove-heat was taken off, until finally stoves were not used at all. Hot-blast pipe-stoves are certainly a most wasteful apparatus for heating air. From 300° to 500° F. should be picked up from the waste-gases of the furnace with a proper arrangement, especially if there is sufficient volatilized sulphur to burn above the surface of the furnace charge.

3. *To what extent can fuel be eliminated?*

This depends upon two conditions: the quantity of iron sulphide minerals in the ore, which are the heat producers, and the relation of those minerals to the gangue. To go back to the approximate furnace-charge analysis under the first question: with a mixture of pyrite and a silicious aluminous schist containing 5 per cent iron, and 4 to 8 per cent copper in the form of bornite, but with the sulphide mineral in concentrated and isolated patches in the gangue, 2.5 to 3 per cent of coke was required, and the campaigns were short. With the same approximate analysis, but with a laminated schist containing 8 per cent of iron and 1.5 per cent of copper, with the sulphide mineral scattered somewhat evenly through it in small grains, practically the same furnace-charge is smelted with 0.3 to 0.4 per cent of coke, and the campaigns last from 60 to 80 days. There is no ore, carrying an appreciable quantity of iron sulphide minerals, upon which some pyritic effect cannot be obtained in the way of reduction of fuel in the smelting.

4. *What amount of copper is needed for the collection of the precious metals?*

I note that some metallurgists maintain that copper is not an absolute necessity, and that clean slags can be made with iron matte alone. My own experiences in pyritic smelting have been limited to 1.5 per cent and over of copper in the charge; with the lowest copper I have always had clean slags.

5. *What percentage of lime is necessary to a clean slag?*

The copper matte is the slag cleaner, not the lime. Enough lime or bases, including alumina, to make the most liquid slag obtainable, is all that is required. This allows the free settling of the matte particles. With alumina and barite present, I usually aim at a quarter slag containing about 13 per cent of bases other than iron, and vary the lime to suit that requirement. One-fifth slags work well also; but, going below that, the deficiency of the lighter bases raises the specific gravity of the slag, and is liable to cause a poorer settlement of the fine shots of matte.

6. *What percentage of zinc in the charge can be treated profitably?*

I have had no special experience; but very small amounts in the ores I have been treating. I have noticed, however, on one or two occasions, in the side accretions, more particularly in the furnace corners, small quantities of the zinc-lead sulphide which usually forms the collar accretions in lead furnaces. If there were no lead in a pyritic ore it is possible that some zinc might be carried through, but I believe it would give trouble in the forehearth.

7. *What is the degree of desulphurization attainable?*

I have been in the habit of figuring on the oxidation attainable; one is the converse of the other to a great extent. The degree of oxidation determines the matte-fall. The higher the oxidation, the smaller the matte-fall and the higher the grade of the matte. Oxidizing 80 per cent of the Fe present in 90 per cent pyrite, which is high in practice, will give a desulphurization of 90 per cent, and oxidizing 60 per cent of the iron, which is low, will give an 80 per cent desulphurization.

8. *What are the possibilities as to capacity of furnace?*

The table below gives some dimensions of furnaces:

		2,000 lb. tons of charge per 24 hours.		Tons per sq. ft. of hearth area.	
Size of Furnace.	Area of Hearth in sq. ft.	Blast 20 to 30 oz.	Blast 40 to 45 oz.	25 to 30 oz.	40 to 45 oz.
126 by 36	31.50	199.08 tons	284.57 tons	6.31 tons	9.03 tons.
168 by 40	46.16	217.43 "	402.94 "	4.65 "	8.60 "
210 by 42	61.25	271.43 "	530.76 "	4.42 "	8.65 "

On a pure pyritic ore, under the most favorable circumstances and with 50 oz. of blast, I have no doubt that the quantities could still be increased by a fifth. The larger furnaces give a smaller hearth-efficiency, but the labor-efficiency is largely increased, that is, the number of men to run the furnace is not increased in proportion to the additional tonnage treated per day.

9. *What are the limitations of the process?*

The answer to this may be deduced from the answer to question No. 3. It is an oxidation process, and naturally shades off from pure pyritic ore, where the oxidation must be the highest, to those oxidized ores that do not require it at all; from those ores of sufficient fuel-value to furnish their own heat for fusion, through those that must be assisted by varying quantities of extraneous fuel, to those that have to be smelted entirely by it. It also may be limited by excessive quantities of zinc and lead.

10. *What is the relative economy as compared to rival processes?*

It has no rival, where suitable. As against the kiln or heap-roasting, previous to blast-smelting, there is a large saving in the expensive fuel, coke; a complete elimination of the cheaper fuel used for starting the roast-heaps, namely, wood; a saving in the labor of handling the ore to and from the roast-yard; a saving in the interest on a large quantity of dead stock locked up in the roast-heaps, and a great saving in the loss of copper by leaching, especially in wet countries. With 8 to 12 ft. of rain per year, as is the case on the west coast of Tasmania, open roast-heaps are simply out of the question. To cover the heaps required to furnish 1,000 tons of ore per day would mean a heavy first cost and a heavy maintenance cost.

One of the most striking advantages is the rapidity of the work. Eliminating the periods for cooling the matte between the several operations, the actual time from the feeding of the ore into the furnace to the pouring of the blister copper into the moulds from the converter vessels is about 5.5 hours. This includes the smelting of the ore and the production of a first matte of 9 to 15 per cent copper, the resmelting of the first matte, producing converter matte of 40 to 50 per cent copper, and, finally, the melting and blowing the converter matte to blister copper. If converter matte is made direct from mixed ores, then the time of treatment is but 3.5 hours. I might here add a note that so far as the furnace manipulation is concerned, pyritic smelting is the most delicate of all blast-furnace work. With little or no coke in the furnace there is no latitude for any application of the remedies common to other work, and although, when absolutely necessary, a sudden shut-down of an hour may be made, it will take seven or eight hours to recover from it, and often the foundation is laid for a trouble that does not manifest itself for a day or two. Short campaigns are the rule, and the frequent shutting down of a furnace must be considered as part of the process.

G. F. Beardsley.

San Francisco, Jan. 9, 1904.

CONTRIBUTION BY THOMAS T. READ.

The Editor:

Sir—A good example of the smelting of raw sulphides of iron and copper, utilizing the heat of combustion of the sulphur content to replace a portion of the coke usually required, is seen at the plant of the United States Smelting Company at Bingham Junction, about 15 miles from Salt Lake City, Utah.

This plant is at present smelting about 700 tons daily, but the work of enlarging it to 1,000 tons capacity is already under way. The larger part of this tonnage is derived from the company's mines at Bingham cañon, but ores from the Boston Consolidated are also being smelted. The ore is of two classes, a silicious ore that presents no especial difficulties, and a sulphide ore that breaks up into fine particles and gives rise to the difficulties which are always met in the smelting of fine ore. The average content of the two ores is about 2 per cent copper and nearly equal values of gold and silver.

The plant has only been in operation about a year and is essentially modern and up to date in every particular. Being situated on the eastern side of the valley of the river Jordan it has the sloping site necessary for the ellipsoidal arrangement of track and ore-bins which characterizes the late designs of smelting plants. The smelter and converter houses, or house, for they are under one roof, are of structural steel and sheet iron; the power-house is of brick and the ore-bins are of the usual timber construction.

The ore is delivered at the smelter from the mines in 50-ton cars and goes first to the sampling bin and then to the storage bins. These are provided with hopper bottoms and Fairbanks track-scales beneath. These scales are set at the desired weights in the usual way and the charging cars, drawn in a train by an electric locomotive, are weighed directly as they are being filled. From the bins the trains go to the charging floor, where the cars are dumped on a shoveling-plate. As designed, it was intended to charge the furnaces directly from the cars, but this was found unsatisfactory and charging by shovel was substituted. A slight change has been made in the design of one of the furnaces and automatic charging is to be again tried.

The furnaces are five in number (four in blast), 42 in. by 180 in., water-jacketed with round brick settlers. Apparently the width is a little too great under the circumstances—that is, with a charge containing a good deal of fine and a highly silicious slag—for it renders necessary a high blast pressure (30 oz.), to secure penetration which, in conjunction with the finely divided condition of the ore, makes an excessive quantity of flue-dust, so much indeed that it is difficult to handle. Having to put all this back through the furnaces, after briquetting, tends to increase the trouble. Apparently narrower furnaces with larger tuyeres and a lower blast pressure would give better satisfaction. With such a high blast-pressure the end tuyeres on the sides would cause a chill if allowed to remain open, and are therefore kept plugged.

The heat of combustion of the sulphur allows the coke to be cut to 6 or 7 per cent. A hot blast has been used on two of the furnaces with apparent success, but the hot-blast stove has required closing down for repairs so often that no results that are really conclusive have as yet been obtained. No attempt is made to utilize the heat of the flue-gases. The flue is constructed of brick, with sheet steel downtakes. These latter were at first made of 6 ft. diameter, but, being the cause of a very gassy charging floor, were replaced by others 7 ft. in diameter with a marked improvement.

The first matte from three of the furnaces averages about 15 per cent copper and is cast in the usual way. It then goes back to the fourth furnace, together with a nearly equal percentage of ore. The matte from this furnace runs from 25 to 30 per cent copper and goes directly to the converters, where it is blown up to blister copper. This is then cast into pigs and shipped to an eastern refinery. The casting of anode plates is not attempted at any plant in the district.

A large economy in the handling of the slag is effected by granulating it. The granulated slag is led into two settling tanks, alternating from one to the other, so that while the slag is being removed from one tank the other is filling. These tanks are fitted with hopper bottoms, so that the slag larry can be brought beneath and filled directly. The slag is then trammed to the southern end of the property and dumped. There is sufficient fall to allow of wasting the slag directly from the end of the flume, but the fear of allowing it to enter the Jordan river has so far prevented this. The water for the granulation of the slag and the general supply of the plant is obtained from a ditch taken out of the river some distance up-stream. As the formation of anchor ice in winter and irrigation in summer are likely to diminish this supply, a pumping system has been installed, which is capable of supplying the entire plant from the Jordan river directly opposite the plant. In addition there are three artesian wells which are pumped by the Poehle system into elevated tanks to supply water under pressure where it is needed and for fire protection.

The matte is tapped directly from the settlers into 10-ton pots which are handled by an electric crane. The first matte is cast in molds on a casting track, the second is delivered directly to the converters, of which there are two, of the cylindrical or 'trough' shape. These hold one potful of matte and have a hydraulic tilting device which is actuated by a small Worthington pump. Silicious ore is ground in a pug-mill with just enough clay to give it the right consistency, and is then used for the converter lining. It is rammed into place by a 1.75 in. Ingersoll rock-drill, which is fitted with a tamping iron in place of the drill bit. The drill is suspended from a small jib-crane, allowing ready adjustment to the work. The converter shells are handled by the electric crane and only about ten minutes is required to remove a shell from its stall and substitute another.

The slag is poured at intervals during the blow, which lasts about two hours, the progress of the skimming being watched by dipping an iron paddle into the issuing stream at frequent intervals, and noting when spots of matte or white metal appear on it. The termination of the blow is known by the appearance and sound of the issuing flame, as in the bessemerizing of steel. The small amount of slag remaining on the surface at this stage is kept back by throwing some of the rough fragments of copper from the previous casting into the mouth of the converter. In this way clean pigs are obtained. Naturally with such a low-grade matte a great deal of patching of the lining is required, and the lining has to be frequently renewed, so that a gang of men are kept constantly busy preparing and applying converter lining. A blast pressure of 13 lb., furnished by a piston blower, is used at the furnaces.

The entire power plant is of Allis-Chalmers design and construction. Cross-compound Corliss engines at 145 lb. steam pressure are used for the blowers. The electric power is furnished by a tandem compound Corliss, which is belted to a direct current, and an alternating current generator, the direct current being used for the tramming system and the electric crane, the A.-C.

being used for all other purposes—as, for instance, the driving of the briquetting machine, the furnishing of power for the machine- and boiler-shops, lighting, etc. The Corliss engine has been running, at 120 revolutions per minute, a year, with only a shut-down of a few minutes each Sunday. A Westinghouse engine is being installed in order that an accident to the Corliss may not cause a shut-down of the whole plant. A Rand compressor supplies the air for the tamping machine and for pumping the artesian wells.

The boiler plant consists of four Sterling units delivering 1,500 engine horse-power. They are fitted with Roney automatic stokers and the ashes are sluiced away along with the slag, so that two men are amply sufficient to take care of the plant. The Roney stokers give good satisfaction, although not an absolutely smokeless combustion.

The briquetting plant is chiefly interesting because of the amount of work it is called upon to perform. Slacked lime is used as the binding material and a fair quality of briquette is obtained. The method of sampling the blister copper is worth a passing note. The assay sample is taken near the middle of the pour. The samples taken during a day are carried to the machine shop the next morning and cut completely in two on a diagonal line by a planer. From the chips thus obtained the assay sample is taken.

A little less than 200,000 lb. of copper per week is the usual product of the plant. It is a great financial success, and thus, under the circumstances, bears eloquent testimony to the excellence of its design and the efficiency of its management.

THOMAS T. READ.

Laramie, Wyo., Feb. 16, 1904.

CONTRIBUTION BY HENRY W. EDWARDS.

The Editor:

Sir—In the current number of the *Journal*[1] I see that Mr. A. McCharles, of Sudbury, refers in a rather complimentary manner to the work done some 12 or more years ago at the smelter belonging to Messrs. H. H. Vivian & Company, at the Murray mine. He is quite right in his main facts; and as this plant was rather a pioneer in a small way, it may be of interest to give a few particulars concerning it.

To the best of my knowledge and belief it was the first plant to tap matte direct from the blast furnace to the converter, and it was there that the first trough-shaped converter was used in North America. Prof. Douglas in a pamphlet entitled 'Bessemerizing Copper Mattes' is good enough to give the undersigned the credit of being the first to operate this style of converter.

The smelter originally consisted of two old-style round Piltz furnaces, 60 in. diam., three converters, a small vertical engine driving a Schicle fan, and a simple horizontal blowing engine. The idea of the fan was to produce a large volume of blast with a view to enriching the matte by oxidation of a part of the sulphide of iron in the ore. However, one of the Piltz furnaces was soon replaced by an oval furnace designed by the writer with a single eye to low first cost. If I remember correctly, it cost about $2,000 erected. Its greatest length was about 90 in., its width 33 in., with a 3-in. water space and vertical sides. It was blown by a Baker blower, No. 6, I think, and its capacity was about 600 to 800 tons per week of roast-heap stuff. The matte

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[1] *The Engineering and Mining Journal*, Feb. 11, 1904, p. 232.

was tapped direct into small trough-shaped converters, each mounted upon a traveling blowing stand; after receiving its charge the converter was moved along the track (by means of hand-cranks operating gear-and-pinion on the driving wheels) to the blowing-hood. The usual charge of matte was about a ton or a ton and a quarter for a converter with a fresh lining; with an old lining they would take three tons and over. Blowing was continued until practically all the iron was oxidized, the product being a copper-nickel matte containing very small amounts of iron sulphide. The converter slags usually assayed 2 per cent nickel, and over, and were returned to the blast furnace.

The plant was simple in the extreme; it cost about $25,000 or $30,000; its capacity being about 600 to 800 tons of burnt ore per week, in addition to taking care of all the converter slags. It cost to operate about $2,000 per week, including roast-heaps. About half, or rather more, of this was for fuel. The cost per ton of ore for the whole series of operations, burning, smelting and converting, was about $3 on an average, including all the ordinary repairs and renewals.

The ore was very low grade, running about 1.5 per cent nickel and about 0.75 per cent copper, so that I had to be very careful not to let the slag assays run too high. I am glad to hear from Mr. McCharles that they have not yet been beaten in the district.

At the time this plant was running the price of nickel was about half the nominal quotation of to-day; this, coupled with the cost of mining, the small scale of operations and the disadvantage of an administration 4,000 miles away, all combined to make the venture rather discouraging; and after some four years of existence it was finally given up. If current prices of nickel could be guaranteed it would pay to reopen the property, as there is still in the mine plenty of ore of the grade then worked. Very little work was ever done underground below 50 ft. and none at all below 100 ft. During the year 1892 I tried there several experimental runs with crude ore, the more modern 'pyritic' smelting, in fact; but the results were not at all satisfactory, chiefly owing to the failure to get a matte of suitable grade for the converters.

The ores of the Sudbury district in general, and of the Murray mine in particular, consist mainly of massive pyrrhotite with a gangue of gabbro or diabase. Both these rocks are comparatively fusible. As a consequence smelting took place so rapidly that there was little opportunity for oxidation beyond that necessary to generate the heat of fusion. The use of silicious material as a retarder was not attempted.

In this instance hot blast would have been a detriment rather than an advantage.

A little coke was used along with the cold blast. The fuel was not added regularly but in occasional doses preceding and following a slag charge, in order to revive the furnace when the pyrite action seemed to wane. The concentration was only about two or two and a half into one, and no attempt was made at re-concentration in the blast furnace before blowing in the converters.

Another question also presented itself: sulphur not being a reducer for nickel oxide, would not nickel enter the slag if oxidation were pushed too far? As the main object of the operation was the recovery of the nickel I was a little timid in running too close to the limit. There is no reaction between nickel sulphide and nickel oxide analogous to that between copper sulphide and copper oxide. There seems, however, to be a partial action between copper sulphide and nickel oxide which demands closer investigation.

HENRY W. EDWARDS.

Grand Junction, Colo., Feb. 17, 1904.

CONTRIBUTION BY CHARLES H. FULTON.

The Editor:

Sir—I take pleasure in replying to the ten questions submitted in regard to pyrite smelting. I have read with much interest the replies given to these questions by the metallurgists taking part in this discussion, noting how their experience compares with that gained by me at the smelting plant at Rapid City, South Dakota.

1. *What types of ores are suited to the process?*

Ores containing sufficient sulphides of iron, either pyrite, pyrrhotite, etc., or copper sulphides are suitable; also, probably, arsenical pyrite. The question is, however, a wide one, for ores partially oxidized and containing but little sulphur can be handled without difficulty. A rather exceptional case is presented by the practice at Rapid City and Deadwood, where the dry silicious ore of the Black Hills is smelted with barren pyrite, limestone and a little copper ore. It might be said that any ore mixture containing sufficient sulphur to form matte with iron and copper, and with sufficient silica to form a rather high silicious slag, is suitable. But little or no lead and but little zinc should be present. Any lead present would be lost, aside from causing heavy silver losses.

2. *Is hot blast advisable?*

Regarding the saving of fuel by the employment of hot blast I cannot speak from experience. At Rapid City, what might be called a warm blast is used, the blast being heated in a U-pipe stove placed in the dust-chamber directly below the downtakes. The temperature of the blast is usually about 130° F., not enough to materially affect the fuel consumption. Hot blast, however, possesses one notable advantage, increasing the oxidizing effect of the furnace, permitting a greater desulphurization and hence a higher concentration. This may become a vital point in the smelting of low-grade ores.

3. *To what extent can fuel be eliminated?*

I have no experience on this point. In the Black Hills the production of highly silicious slags, with the feeding of just enough pyrite to furnish iron for the slag and matte, and sulphur for the matte, calls for considerable fuel. Also, the grade of coke used, for the most part Wyoming coke containing 30 per cent ash, aside from being but loosely coherent, precludes much experimentation on this point. The amount of fuel consumed amounts to about 12 to 14 per cent of good eastern coke.

4. *What amount of copper is needed for the collection of the precious metals?*

That depends upon what amount of slag loss the plant can stand. It is evident that when only the silicious ore going into the furnace is productive, and the limestone and pyrite being added are barren, the slag loss per ton must be very low. At Rapid City it was endeavored for some time to smelt without copper ore, with the result that the slags would invariably run from $1.50 to $2.50 in gold and 0.5 oz. silver. The addition of copper ore immediately remedied this. Copper ores are procured from Montana, and are added in just sufficient quantity to give the desired result. The minimum amount of copper added is 1 lb. for every one ounce of gold present in the charge, and more if available.

The first matte made usually contains from 10 to 14 per cent copper, 4 to 5 oz. gold, and 6 to 8 oz. silver. This is re-smelted twice, until the shipping matte reaches a value of 20 to 22 per cent copper, 17 oz. gold and 22 oz. silver. The re-smelting of the matte also furnishes somewhat more copper for some of the charges.

The loss of silver becomes very noticeable when the copper contents of the matte drop below 8 to 10 per cent. In my opinion few plants can smelt without any copper ore.

5. *What percentage of lime is necessary to a clean slag?*

Lime is necessary for a mechanically clean slag, that is, a slag free from entangled matte. Lime reduces the specific gravity of the slag, letting the matte, low in specific gravity, settle out completely. The amount of lime in the slag depends upon local conditions. At Rapid City the slag made has the following composition: CaO, 28 per cent; FeO, 18 per cent; Al_2O_3, 4 to 5 per cent; SiO_2, 49 to 50 per cent. Some of the lime could be replaced with profit by magnesia up to 8 or 10 per cent. Lime slags of the composition quoted flow very freely and smoothly from the furnace, and allow a perfect separation of the matte. High silicious slags of this type, when produced with copper in the furnace, are very clean. They usually carry 20 to 30c. in gold and 0.2 oz. silver.

6. *What percentage of zinc in the charge can be treated profitably?*

I have no experience with zinc in pyrite smelting.

7. *What is the degree of desulphurization attainable?*

At Rapid City the desulphurization is between 75 and 80 per cent of the total sulphur in the charge. I believe a greater degree of desulphurization could be obtained if necessary. The furnaces used contract but little at the tuyeres and have a strongly oxidizing action. The matte-fall, with this desulphurization, amounts to between 4 and 5 per cent of the total charge. This is found amply sufficient for the collection of the values, giving a clean slag.

8. *What are the possibilities as to capacity of furnace?*

This is less than with furnaces running on partially roasted ores. A 38 by 144-in. furnace will take about 150 tons of burden per 24 hours, with a considerable amount of fine present. With good lump ore and good coke, a furnace of the above size should take 200 tons.

9. *What are the limitations of the process?*

Pyrite smelting is limited to the ores discussed under the first question.

10. *What is the relative economy as compared to rival processes?*

That depends much on local conditions. At present the pyrite smelting plants of the Black Hills could not profitably handle silicious ore of less than $15 per ton, bearing in mind that this is the only productive material going into the furnace. In other districts, where conditions are more favorable as regards fuel and ore supply, pyrite smelting would seem to be a serious rival of other smelting processes, of water concentration, and, indeed, of chlorination and cyanidation, for even comparatively low-grade ores, on account of the high saving made.

Charles H. Fulton.

Rapid City, S. D., Feb. 8, 1904.

CONTRIBUTION BY P. WISEMAN.

The Editor:

Sir—In your issue of February 4th you have an editorial entitled 'Pyrite Smelting,' and Mr. Godshall presents an interesting contribution upon the same subject.[1] You make the suggestion that this term be used when referring to the "oxidizing smelting of pyrite ores in a shaft furnace." Such designation would probably be more satisfactory than 'raw sulphide' smelting, but in reviewing the reasons given it seems to me that even 'pyrite' smelting does not fittingly describe the operation. In the minds of those interested in the smelting of copper sulphide ores there are indelibly associated with the word 'bessemer' the oxidation of sulphur and iron, and the utilization of the heat thus generated. As this is the chemistry of the process which has been variously referred to as 'pyritic,' 'sulphide,' 'raw sulphide,' 'matte' and 'pyrite' smelting, it would seem more appropriate to speak of it as 'bessemer' smelting, and I would therefore suggest the use of that term.

This designation describes the process and at the same time implies the character of the material treated, and likewise the product of that treatment, at least when converting is the last stage of the process. The term would seem to be both comprehensive and distinctive. It is broad enough to cover every type of ore from which enough heat can be generated by the oxidation of its sulphur (and iron) contents to do away entirely, or as far as possible, with the use of carbonaceous fuel in shaft-furnace smelting. If further explanation is desired one can speak of bessemer smelting of pyrite, pyrrhotite, chalcopyrite, etc. Confusion

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[1] *The Engineering and Mining Journal*, **Feb. 4, 1904, pp. 188 and 192.**

can scarcely arise on account of the bessemerizing of matte, as this operation is almost universally spoken of as 'converting,' and as for the product, it is certainly none the less bessemer copper because the same reaction has been employed in each operation throughout the process.

The use of this term would not necessarily imply that carbonaceous fuel was entirely eliminated, but that as full use as possible was made of the heat caused by the oxidation of the sulphur (and iron) contents of the charge. Bessemer smelting would have reference to an oxidizing reaction only. Mr. Godshall calls attention to the fact that the term 'raw sulphide' smelting is particularly inappropriate in cases where the sulphur content is so low that it becomes necessary to maintain a reducing atmosphere in order to obtain a matte product. In such instances the term 'bessemer' smelting would not be proper, and the operation would probably continue to be spoken of as simply 'matte smelting.' If it were desired to make a further distinction between the processes of matte smelting, the expression 'bessemer matte' smelting would answer for the oxidizing method.

Bessemer smelting may, therefore, be defined as follows: Bessemer smelting is the smelting of heavy raw sulphide ores in a blast furnace, by means of an oxidizing atmosphere, and the utilization of the heat generated by the oxidation of the sulphur and iron contents of the charge to reduce to a minimum, or entirely do away with, the use of carbonaceous fuel.

With regard to the use of hot blast, its apparent advantages at the plants where it has been put in use do not seem to have received from those interested in smelting the careful consideration they would seem to deserve. That, under the conditions prevailing at some of the plants, the use of hot blast has been a commercial success will probably not be questioned. It is not just to condemn hot blast unless a test of both hot and cold blast has been made upon the same ore, and under conditions fair to both, and the hot blast has failed to give the desired or expected results. That upon another ore, and possibly under the most favorable conditions, one has been able to smelt with as little coke and with as large a capacity per square foot of hearth area as some one else using hot blast is simply negative evidence. Possibly the use of hot blast might have given even more favorable results. It is unusual for a process to give equally satisfactory results upon all ores; the process must be adapted to the ore and the working conditions. It seems reasonable to suppose that with fuel considerably cheaper than coke it would be less expensive to heat the blast outside, instead of inside, the furnace, and that with the probable reduction in the quantity of coke used, the ore capacity of the furnace would be increased. However, in heating outside the furnace the problem is to avoid excessive losses from radiation and otherwise, but when attention is given to this matter it probably will be satisfactorily solved. I have seen no data as to the test of hot blast at Butte, but I have the impression it was found unsatisfactory because the saving of coke effected was offset, or more than offset, by the expense of heating the blast outside the furnace. At the time the test was conducted were efficient steps taken to prevent excessive losses by radiation in the hot-blast stove? At all smelting plants there is a great amount of heat going to waste, and it would seem that every smelter superintendent should make an effort to use this waste heat in the heating of the blast, or for some other purpose. If the installation of boilers in connection with reverberatory furnaces, at Anaconda, proves satisfactory, it is a distinct advance. While the utilization of escaping furnace gases might not heat the blast to as high a temperature as might be desired, it is probable the gain would be sufficient to more than pay interest upon any expenditure for apparatus installed.

I understand that Mr. J. Parke Channing has been experimenting at Ducktown, with the object of doing away entirely, or nearly so, with the use of carbonaceous fuel, and that he has met with very considerable success. His experiments may have included the use of hot blast. Perhaps Mr. Channing will favor your readers with his results? They would certainly be of very great interest in the consideration of this subject.

I have been informed of one plant where the use of hot blast has reduced the coke consumption, about 25 per cent. A part of the charge consisted of roasted ore, and when raw ore is substituted for this it is expected the coke consumption will be still further reduced. I hope to be able soon to give detailed information as to the results at this plant.

In the latter part of his article Mr. Godshall goes into the relative economy of the different methods of treating ores, and mentions that Messrs. Carpenter and Lang both advocate the direct smelting, without concentration, of the Butte ores. When one hears of the very low smelting costs which are said to prevail at some of the Montana plants the question very naturally arises whether or not the present methods could be advantageously changed. The costs of smelting in recent years have certainly been very materially reduced. Years ago the costs of, and the losses in, concentration were considerable, and if the costs and losses of those days are being figured against the low smelting costs of to-day, the advantage, assuming a favorable smelting mixture, would possibly be with straight smelting. But it is possible, and probable, that the costs and losses in concentrating have likewise been very much decreased, thus lessening any advantage straight smelting might seem to have. Furthermore, the concentrating ores contain such a percentage of silica that a large tonnage of expensive flux would be required, and the amount of first-class ore going into the charge would not greatly alter the composition percentages. It therefore seems unlikely that straight smelting would prove to be any economy over the method in vogue. It has occurred to me, however, that possibly a larger tonnage of the low-grade ores is being concentrated than is absolutely necessary for a desirable smelting charge, but only those who have all the facts which would have to be taken into consideration can determine this.

Mr. Godshall suggests that instead of attempting straight smelting, "it would be better to first demonstrate the advantage, if there be any, of smelting the present Butte charge raw or without any previous roasting." I do not recall that roasted concentrates go to the blast furnace. The expense of briquetting the fine concentrates satisfactorily is so great that they are preferably roasted and treated in the reverberatory furnaces with comparatively slight losses in flue-dust. I think in the present practice the blast-furnace charge is 'raw,' and probably the only changes which can be looked forward to will be the utilization of the heat value of the sulphur in this charge, if the sulphur contents will permit, and the use of a heated blast.

P. WISEMAN.

Los Angeles, Cal., Feb. 12, 1904.

CONTRIBUTION BY W. H. NUTTING.

The Editor:

Sir—The term 'raw' smelting certainly defines itself and covers a great variety of ores, while 'pyrite' smelting has some distinctive features, and is more restricted in its application, so that the latter term can hardly be avoided.

When the ore treated has sufficient iron sulphide to become a decided factor in replacing carbonaceous fuel by the rapid oxidation of these elements in the blast furnace, thereby generating enough heat to do away with all, or a large part, of the carbonaceous fuel which would be necessary in the absence of these elements in excess of the amount required for the composition of matte showing the required concentration, pyrite smelting—as, possibly, the most distinct process—merges into smelting for matte without any great excess of sulphur and iron to be used as fuel, and thence into copper smelting, with so little sulphur present that it does not need to be considered as a factor in the running of the furnace or the composition of the product. I believe that the questions under consideration apply more nearly to pyrite smelting.

1. *What types of ores are suited to the process?*

Iron sulphide ores—the purer the better—combined with silicious ores; and if both are free from fine the conditions are ideal, especially if the charge is free from zinc sulphide. Under these conditions most of the claims made by the most enthusiastic advocates of pyrite smelting could be verified, but, unfortunately, all these favorable conditions seldom exist in any one locality, and the question arises: What percentage of fine and zinc sulphide makes the ore unsuitable?

Or, in other words: At what point would some other method, or *combination* of methods, be competitive? I think that each separate smelting proposition has to be figured out from existing local conditions, such as the physical character of the ore, the cost of coke and fluxes, etc. Ores containing lead are not suitable for various commercial reasons, though this element does not interfere with the process in other respects.

2. *Is hot blast advisable?*

Every metallurgist should be guided by local conditions as to the advantage to him of a heated blast from a commercial standpoint. Metallurgically I believe it is always an advantage; but the question of getting a heated blast at a cost showing any additional profit over the use of the necessary amount of coke, at Utah and Colorado prices, has never been satisfactorily demonstrated to me, either by my own experience or that of others. The conditions affecting the comparative cost are the amount of fine and zinc sulphide in the charge, together with the price of coke.

3. *To what extent can fuel be eliminated?*

With a coarse charge, together with other true pyritic qualifications as to sulphur and iron, coke can be eliminated in proportion to the absence of fine and zinc sulphide, down to the intermittent use of two or three per cent.

4. *What amount of copper is needed for the collection of the precious metals?*

In the absence of zinc sulphide a fairly satisfactory recovery of the precious metals can be made in an iron matte without copper, but the beneficial effect of even 0.5 per cent copper in the charge will be recognized; and if the charge contains a considerable amount of zinc sulphide, 1.5 to 2 per cent copper becomes necessary for a good separation, as a reasonable difference in the specific gravity between the matte and slag must be maintained.

5. *What percentage of lime is necessary to a clean slag?*

The amount of lime necessary is governed by the amount of available iron for the silica in the charge. When the iron is sufficient to satisfy both the matte requirements and the silica, very little lime is necessary, and may be cut down to six or eight per cent. In the case of insufficient iron, lime can be used as a base with which to combine the silica, to a considerable extent. I find that under my present conditions here the following composition of slag works well in every respect: FeO, 28 per cent; CaO, 18 per cent; SiO_2, 44 per cent; Ag, 0.24 oz.; Cu, 0.22 per cent.

6. *What percentage of zinc in the charge can be treated profitably?*

This element, in the form of sulphide, is an unmitigated nuisance. Its bad effects are numerous, and will be noted in the tonnage, recovery and product. Difficulties start with as little as three per cent in the charge, and increase quite in proportion until 12 per cent is reached, when the bad effects become serious, scaffolding the furnace shaft and forming accretions at the bottom of the furnace, sump and fore-hearth. The accumulation on the bottom of the furnace continues, if not corrected, until the tuyeres are slagged, at which point the furnace is likely to go out of business for a few days. If taken in time, and the zinc in the charge reduced, with the addition of scrap iron, the furnace can generally be nursed into good condition again.

Zinc requires additional fuel and iron, reduces the tonnage treated, causes volatilization of silver, both in the furnaces and converters (if blistered copper is the final product), increases the specific gravity of the slag and decreases that of the matte, thereby making the separation less perfect, and reduces the value of the matte, whether sold, re-treated, or converted, and will show a very marked increase in silver losses in the slag.

7. *What is the degree of desulphurization attainable?*

Under conditions which warrant the choice of this process, the smelting is rapid and the elimination of sulphur is from 60 to 70 per cent, with a concentration of 6 or 7 into 1. A much higher degree of desulphurization could be attained, but it would be at the expense of tonnage, either by slower driving, or by a more silicious charge. In either case, the tonnage would be less.

8. *What are the possibilities as to capacity of furnace?*

The capacity of a furnace means the number of tons of *ore* put through in 24 hours, and the possibilities in this direction are governed:

First, by the amount of fluxes necessary to the ore charge.

Second, by the percentage of fine in the charge.

Third, by the amount of zinc sulphide present in the charge; assuming that other conditions are comparative.

I believe that the limit, as regards hearth area, and volume and pressure of blast, have been reached, and that new records will depend upon the character of the charge.

9. *What are the limitations of the process?*

Competitive processes, or combination of methods, showing greater profits, governed to a great extent by fine, zinc sulphide and cost of coke.

10. *What is the relative economy as compared to rival processes?*

Less expensive construction of plant, less handling of material, less fuel required. Absence of roasting, either in mechanical roasters, which require crushing and rolling of the ore, or heap roasting, with its attendant handling and loss; and fuel used in either case. Large tonnage and rapidity of work, which means less operating cost per ton of ore. The resulting slags can be very silicious, if necessary up to 46 per cent, without greatly retarding the speed of the furnace, and are comparatively clean. The sulphur excess in the charge is naturally a safeguard in taking care of the copper, preventing the formation of oxides which might enrich the slag.

In conclusion, I wish to confirm the experience of other operators in regard to the delicacy of the process. The care and continual watchfulness increase as the carbonaceous fuel is decreased, and any stop at all affects the furnace for hours, and long campaigns are the exception.

W. H. Nutting.

Bingham Junction, Utah, Feb. 28, 1904.

RAW SULPHIDE SMELTING AT DUCKTOWN.

BY W. H. FREELAND.

The following facts, figures, and observations are intended to contribute something to the practical side of a subject the literature of which is, unfortunately, still largely that of theory and desultory experiment.

The work described here was done at the works of the Ducktown Sulphur, Copper and Iron Company, Limited, at Isabella, Tennessee, under the writer's personal supervision. It covered a period of several months, the first of which was productive of little else than 6 per cent matte, 'break-outs,' and 'chills'—a repetition of the writer's experience in earlier attempts here and elsewhere. But in the present instance, supported and encouraged by a board of progressive directors, and particularly by Mr. J. G. Gordon, managing director, the work was persisted in, and rewarded by the measure of success now recorded; and it may not be amiss to add here, that it has been suspended only to clean up the stocks of roasted ore on hand before adopting it permanently. The practice consisted of two operations, carried out alternately in the same furnace:

1. The smelting of raw ore to a low-grade matte, in the neighborhood of 20 per cent.

2. The re-concentration of the low-grade, to a 50 per cent matte.

Furnace, etc.—A Herreshoff furnace was used, a general description of which may be found in Peters' 'Modern Copper Smelting.' It has a total depth of 8.5 ft., with a cross-sectional area, at tuyeres, of 21.7 sq. ft. The regular Herreshoff forehearth, proving troublesome, was replaced by a water-cooled, blast-trapping spout, and an ordinary brick-lined settler of 5 ft. by 4 ft. by 18 in. internal dimensions. A No. 6 Connersville blower, driven by a direct-connected engine, supplies the blast.

Duration of Campaigns.—As each locality presents its own conditions, it is useless to recount the preliminary difficulties. Suffice it to say, that these were overcome, when the lessons taught by repeated failures were learned and understood. This stage reached, several campaigns of from two to six weeks were made without stoppage of blast, their duration being limited only by the necessity of shutting down to wash out silt from furnace jacket, spout, etc.

Test Run.—One of these campaigns was a test run, differing from the others, however, only in the care bestowed upon weighing and sampling.

Hourly samples were taken of each constituent of furnace charge, also of slag and matte. These were combined into daily samples, and reduced to laboratory pulps, which in their turn were combined, in proportions corresponding to the daily tonnages. The final samples, thus representing reliable averages of the entire run, were carefully analyzed, in duplicate, by Mr. Thorn Smith, a chemist whose abilities are known to readers of *The Engineering and Mining Journal.* The same care was observed in the weighing of materials, including flue-dust; and allowances were made for material in furnace at start and finish of both operations.

The first operation occupied 16½ days, smelting in that period 1,120 tons ore, 89 tons quartz, 162 tons slag; 1,371 tons total burden; with 38 tons coke, the coke being equivalent to 3.4 per cent of the ore, or 2.77 per cent of total burden.

Ore, etc.—The Ducktown ore, as is well known, is a pyrrhotite, carrying less than 3 per cent copper, and no precious values. Full analyses of the ore, fluxes, and coke used in first operation, and products therefrom, are given in Table A (page 111).

Matte.—The matte produced is represented by 396 hourly samples. Ignoring fractions, or calling anything over 0.5 a unit, and dropping all lesser fractions, the variations in assays are shown as follows:

6	samples	assayed	11	per cent.	24	samples	assayed	24	per cent.
4	"	"	14	"	20	"	"	25	"
16	"	"	15	"	20	"	"	26	"
38	"	"	16	"	9	"	"	27	"
30	"	"	17	"	10	"	"	28	"
36	"	"	18	"	14	"	"	29	"
28	"	"	19	"	7	"	"	30	"
34	"	"	20	"	1	"	"	31	"
39	"	"	21	"	1	"	"	32	"
28	"	"	22	"					
31	"	"	23	"	396	samples	aver'gd	21.18	%

The above numerical average is, of course, only approximate (the true average being 20 per cent). Calling all values with less than 20 occurrences accidental, the matte may be said to have ranged from 16 to 26 per cent. The lower assays are eliminated, as they practically all occurred at blowing in; and the higher ones, because steps were taken to reduce them as quickly as detected.

From the foregoing tonnages, the average charge is readily calculated. Based on this and the tabulated analyses, a synthesis of the charge and its products is constructed, which, if somewhat empirical, is nevertheless interesting and useful. See Table B (page 111).

Reducing the constituents of the 938.24 lb. of the calculated slag to percentage terms, it is seen to compare very closely with the actual slag analysis, viz.:

Comparison of Calculated and Actual Slags—First Operation.

	By Calculation. Lb.	By Calculation. %	By Analysis. %
Cu	3.40	0.36	0.37
Fe	358.22	38.18	38.84
S	15.94	1.70	1.74
SiO_2	303.89	32.39	32 60
CaO	84.20	8.97	8.24
MgO	30.30	3.23	3.44
Zn	14.11	1.50	1.54
Al_2O_3	11.84	1.26	1.50
Mn	8.15	0.87	0.80
O	108.19	11.54	10.88
Totals	938.24	100.	99.95

TABLE A.—Analyses Pertaining to First Operation.

	MATERIALS SMELTED.				PRODUCTS.		
Constituents.	Ore.	Quartz.	Slag (used on charge).	Coke.	Matte.	Flue-Dust.	Slag.
Cu	2.744	....	.73	....	20.00	2.20	.87
Fe	36.519	1.45	39.20	2.30	47.15	30.80	38.84
S	24.848	.82	1.75	1.58	24.00	16.51	1.74
SiO_2	18.548	96.79	30.90	8.41	.44	23.92	32.60
CaO	7.294	.23	8.51	T	.10	4.45	8.24
MgO	2.672	T	2.71	T	T	1.38	3.44
Zn	2.556	....	2.88	none	2.05	2.98	1.54
Al_2O_3	.911	.32	1.90	3.56	.82	1.94	1.50
Mn	.77	T	.85	none	.53	.56	.80
O, etc	....	†.38	†11.37	†1.00	*4.91	*15.26	†10.88
C	....	....	....	83.86	....		....
CO_2, etc.	*3.138	....	....	....	....	....	....
Loss on ignition	....	.39	....	....	....	....	....
	100.	99.88	100.80	100.71	100.	100.	99.95

*By difference.
†By calculation.

TABLE B.—Synthesis of Charge and its Products—First Operation.

Charge.	Lbs.	Cu	Fe	S	SiO_2	CaO	C	MgO	Zn	Al_2O_3	Mn	Dif.*	
Ore	1,000	27.44	365.19	248.48	185.48	72.94	....	26.72	25.56	9.11	7.70	31.38	
Quartz	80	....	1.16	.26	77.43	.18	....	....	....	.26	....	.71	
Slag	145	1.06	56.84	2.54	44.81	12.34	....	3.93	4.18	2.76	1.23	15.31	
Coke	34	....	.78	.54	2.86	....	28.51	....	....	1.21	....	.10	
Totals	1,259	28.50	423.97	251.82	310.58	85.46	28.51	30.65	29.74	13.34	8.93	47.50	
Deductions (as below)	428.95	25.10	65.75	235.88	6.69	1.26	28.51	.35	15.63	1.50	.78	47.50	
Balance (+ O to Fe, Zn and Mn) = Slag..	938.24	3.40	358.22	15.94	303.89	84.20	....	30.30	14.11	11.84	8.15	....	O 108.19
					DEDUCTIONS.								
Products.													
Matte (20% Cu)	122.65	24.53	57.83	29.44	.54	.12	....	....	2.51	1.00	.64	6.04	
Flue-dust recovered	25.71	.57	7.92	4.24	6.15	1.14	....	.35	.77	.50	.14	3.93	
Volatilized	280.59	....	....	202.20	....	....	28.51	....	12.35	....	....	37.53	
Totals (deducted above)	428.95	25.10	65.75	235.88	6.69	1.26	28.51	.35	15.63	1.50	.78	47.50	

*This column covers imperfections in analyses, and undetermined CO_2 and O.

TABLE C.—Analyses Pertaining to Second Operation.

Constituents.	MATERIALS SMELTED.					PRODUCTS.		
	Matte.	Ore.	Laboratory Samplings.	Quartz.	Coke.	Matte.	Flue-Dust.	Slag.
Cu	20.00	2.79	2.45	....	....	49.63	2.49	.60
Fe	47.15	43.26	31.07	1.45	2.30	25.24	24.79	43.99
S	24.00	29.18	14.84	.32	1.58	28.00	8.91	1.19
SiO_2	.44	10.01	22.66	96.79	8.41	.26	31.43	33.72
CaO	.10	6.32	5.71	.23	T	T	3.81	2.03
MgO	T	1.39	2.03	T	T	T	1.18	.57
Zn	2.05	2.56	2.05	....	none	1.53	3.81	2.12
Al_2O_3	.82	1.00	1.15	.32	3.56	T	3.93	2.16
Mn	.53	.69	.75	T	none	.30	.30	.50
O, etc.	*4.91	....	*17.29	†.38	†1.00	....	*19.85	†12.86
C	....	....		....	83.86	....		
CO_2, etc.	....	*2.80	....	....	....	....	....	....
Loss on ignition	....	....	....	.39	....	....	....	....
	100.	100.	100.00	99.88	100.71	100.05	100.	99.74

* By difference.
† By calculation.

TABLE D.—Synthesis of Charge and its Products—Second Operation.

Charge.	Lbs.	Cu	Fe	S	SiO_2	CaO	C	MgO	Zn	Al_2O_3	Mn	Dif.*	
20% matte	1,000	200.00	471.50	240.00	4.40	1.00	....	T	20.50	8.20	5.30	49.10	
Raw ore	170	4.74	73.54	49.60	17.02	10.74	....	2.36	4.35	1.70	1.17	4.78	
Laboratory samplings	84	.83	10.56	5.05	7.70	1.94	....	.69	.70	.39	.26	5.88	
Slag	160	1.17	62.72	2.80	49.44	13.62	....	4.34	4.61	3.04	1.30	16.90	
Quartz	330	....	4.78	1.06	319.41	.76	....	T	....	1.06	T	2.93	
Coke	95	....	2.19	1.50	7.99	T	79.67	T	....	3.38	....	.27	
Totals	1,789	206.74	625.29	300.01	405.96	28.06	79.67	7.39	30.16	17.77	8.09	79.86	
Deductions (as below).	763.36	199.61	104.33	285.86	4.81	.40	79.67	.14	6.60	.47	1.61	79.86	
Balance (+ O to Fe, Zn and Mn) = Slag.	1,182.18	7.13	520.96	14.15	401.15	27.66	....	7.25	23.56	17.30	6.48	....	O 156.54
DEDUCTIONS.													
Matte (49.63%)	401.60	199.31	101.36	92.18	1.04	T	....	T	6.14	...	1.57	....	
Flue-dust recovered	12.00	.30	2.97	1.07	3.77	.40	....	.14	.46	.47	.04	2.38	
Volatilized	349.76	....	....	192.61	....	....	79.67	....	....	....	....	77.48	
Totals	763.86	199.61	104.33	285.86	4.81	.40	79.67	.14	6.60	.47	1.61	79.86	

* This column covers imperfections in analyses, and undetermined CO_2 and O.

Second Operation.—The reconcentration of the 20 per cent matte occupied a few hours less than three days, and smelted, in addition to the matte, 34 tons of raw ore and discarded samplings from laboratory; this ore being added to charge to keep the tenor of final matte from rising too high for comfortable running.

Analyses, etc., of second operation are tabulated as follows. (For analyses and synthesis of charge in second operation see Tables C and D (page 112).

Comparison of Calculated and Actual Slags—Second Operation.

	By Calculation.		By Analysis.
	Lb.	%	%
Cu	7.13	0.60	0.60
Fe	520.96	44.07	43.99
S	14.15	1.20	1.19
SiO_2	401.15	33.93	33.72
CaO	27.66	2.34	2.03
Mgo	7.25	0.61	0.57
Zn	23.56	2.00	2.12
Al_2O_3	17.30	1.46	2.16
Mn	6.48	0.55	0.50
O	156.54	13.24	12.86
Totals	1182.18	100.	99.74

Furnace Capacity.—Including both operations, the furnace's average daily capacity is 60 tons raw ore. Its average on roasted ore may be fairly quoted at 115 tons, hence a loss of efficiency of nearly 48 per cent. But in considering this feature, it must be stated that granulated slag was used throughout the test run, and that a daily average, equivalent to 80 tons raw ore, has been repeatedly attained when lump slag was available, thus reducing the loss of efficiency to 30 per cent. It is hoped that, with time and experience, this loss will be still further reduced. It is highly probable that hot blast would increase the smelting capacity, but with a furnace of the Herreshoff dimensions, the additional tax for fuel, coal-heavers and firemen was found prohibitive.

Concentration.—It will be seen that concentrations effected were 7.3 into 1 by first operation, and 2.5 into 1 by second, but these by no means represent the limits attainable. Contrary to the experience of many, the reconcentration of the first matte presents no difficulties at Ducktown. There is no limit, within the range of matte, to the second operation. A 6 per cent matte may be brought up to a 50 per cent quite as successfully as any higher grade initial matte; nor is this statement based upon odd or occasional samples, but upon more than one car-load of matte so made.

Occasional samples of 70 per cent matte have been assayed from the reconcentration of a 10 per cent initial matte, but such conditions, if permitted to continue, would speedily result in a 'chill,' particularly with the scanty flow of a small furnace.

The degree of concentration, whether in first or second operation, is in the main proportionate to the speed at which the furnace is driven, and is controlled by the proportion of quartz on charge, or manipulation of blast, or both. But on these seemingly simple measures, hinge not only the grade of the matte, but the life of the campaign, and probably the confidence of owners, if disasters occur too often on first attempts.

Furnace Conditions.—The average volume and pressure of blast were 4,500 cu. ft. free air per minute, and 17 oz., respectively. Some incrustation forms around the furnace top, but not enough to be troublesome. At the region of the tuyeres, however, a porous friable accretion bridges the furnace from wall to wall. Light is rarely discernible on punching the tuyeres. It may seem unreasonable, but is nevertheless true, that a bar has been driven through the furnace, entering a tuyere on one side, and withdrawn from the opposing tuyere, by the naked hand.

This condition, alarming as it would seem in ordinary smelting practice, is believed by the writer to be essential to satisfactory concentration. The condition encountered in barring the tuyeres leaving no doubt that, for a certain area surrounding each tuyere, the furnace is bridged from wall to wall, the molten matte and slag must find its passage into the crucible through channels between tuyeres. If visible, a horizontal section of furnace at tuyere level would probably present something like the appearance shown in the accompanying diagram, Fig. 2.

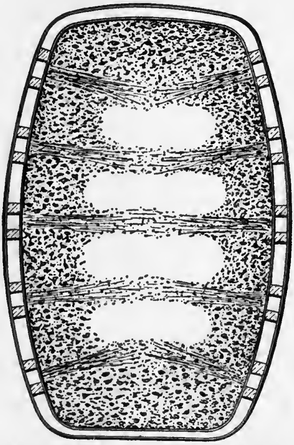

FIG. 2. HORIZONTAL SECTION OF FURNACE.

While the horizontal section is hypothetical, there is no doubt as to the vertical shape assumed by the accretion, which is found to be as shown in Fig. 1.

Granting the condition described, then the effect is: the column of charge resting upon boshes and bridge, undergoing a partial roasting in its descent, and a rapid, fierce oxidation as it reaches and is held in the constricted channels. The charge sinks evenly and uniformly, rarely showing a hot top. The slags run hot and fluid; in fact, the furnace gives less trouble throughout than those alongside smelting roasted ore, a fact testified to by the furnace hands importuning the foreman to "give them a chance" on No. 1.

Coke.—Calculating the percentage of coke on copper-bearing burden only, the regular charge of 1,000 lb.

ore carries 30 lb. coke. Often days will pass without a variation in charge sheet. Occasionally from necessity, but more frequently from the foreman's timidity, the coke is doubled for an hour, or perhaps two, on a shift. It is due to these causes that the average coke percentage on this test run is raised from the nominal 3 to 3.4 per cent of the first operation.

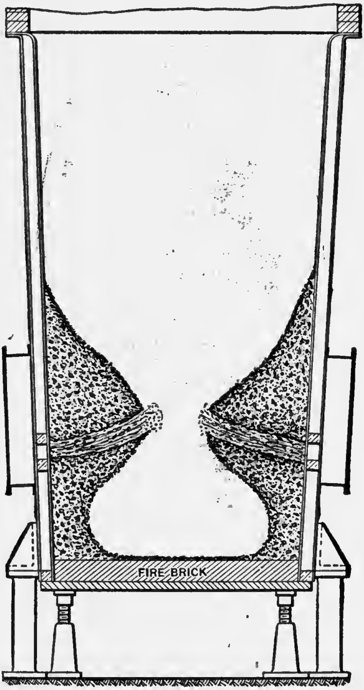

FIG. 1. VERTICAL SECTION OF FURNACE.

On the second operation, the coke averaged 8 per cent on matte, etc. Calculating the coke of both operations back to the original ore, the total coke consumption is 4.4 per cent thereof.

Slag Losses.—Referring to the tables, slag losses are seen to be 0.37 per cent on first, and 0.6 per cent on second operation. Calculating both back to a basis of the slag of the first operation, the equivalent is 0.45 per cent, a loss that should claim the attention of all familiar with the high concentration of a low-grade ore, particularly where settler area is limited by a small matte flow.

Flue-Dust.—The flue-dust recovered from both operations (and almost wholly from the first) was equivalent to 53 lb. per ton of original ore.

Economics.—While it is not within the scope of the present article to go into comparative costs and copper recoveries, it may be added that, despite the greatly reduced tonnage capacity per furnace, the economical result of raw ore smelting is gratifying beyond all anticipation.[1]

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[1] Reprinted from *The Engineering and Mining Journal*, May 2, 1903, p. 664.

COPPER LOSSES IN BLAST FURNACE SLAGS.

By **WILLIAM A. HEYWOOD.**

With a view to presenting in graphic form the ratio of copper in blast furnace slags to the copper contained in the matte produced at the same time, 2,590 assays, representing all the work of two furnaces for a period of two and one-half years, were plotted with the results shown in the accompanying diagrams. The first intention was to plot the slags to show the copper contents only, but it soon became apparent that the amount of silica in the slag had a great influence on the copper contents, the more basic iron slags carrying away much more copper than the more silicious slags. As the ore smelted was heap-roasted pyrrhotite containing only from 10 to 20 per cent of silica, and the remain-

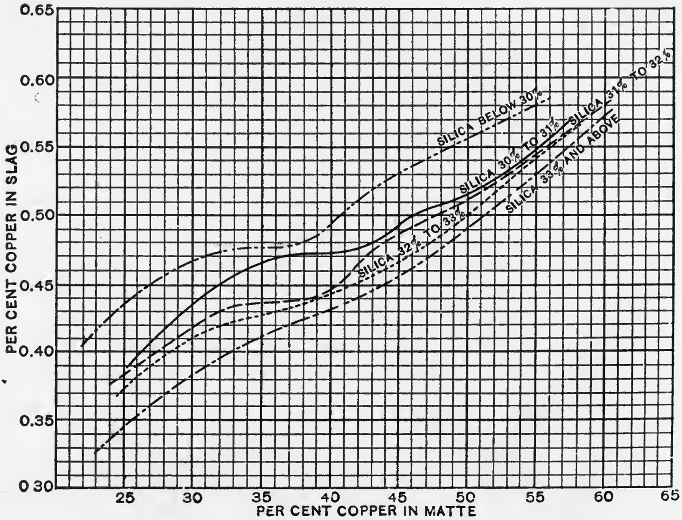

FIG. 1. SLAG CURVES REPRESENTING 2,500 ANALYSES. TENNESSEE COPPER COMPANY, JAN. 25, 1904.

ing silica had to be added in the form of barren quartz, it was an important question to determine what quantity of silica was the most economical, considering the rate of smelting and the cost of converting matte of different grades, as well as the copper losses in the blast furnace slags.

An impression exists at many copper smelters that a furnace producing a slag high in iron runs much faster than one running on a more silicious mixture. Within the limits of our practice this has not been confirmed. Taking our largest runs of over 600 tons per furnace-day, as many have occurred when the furnace was producing a slag with 32 to 34 per cent silica as when the slags contained from 28 to 32 per cent silica.

Each assay represents an average of 12 hours slag from one furnace. The samples were taken hourly, chilled, and the copper determined on a five gram sample by cyanide after precipitating the copper with aluminum. Once a week composite samples were made up from the daily sampies from each furnace and assayed by electrolysis. Part of the weekly composite samples were sent to New York regularly, for check assay. The electrolytic determination is usually 0.02 per cent higher in copper than the cyanide assay. The insoluble in the slag was taken as silica, although the absolute silica determined by fusion is 1.5 to 2 per cent lower.

The following is a complete analysis of a month's average slag:

SiO_2	31.04	per cent.
FeO	51.40	" "
Al_2O_3	4.84	" "
CaO	6.30	" "
MgO	1.37	" "
S	1.36	" "
Zn	2.01	" "
Mn	0.51	" "
Cu	0.45	" "

In Fig. 1 the assays of the different slags and mattes were first plotted separately for the different silica percentages. The curves were then traced on one sheet.

In Fig. 2 these curves were adjusted, and the actual copper losses are shown by allowing for the increased quantity of slag made by the additional quartz necessary to produce the more silicious slag.[1]

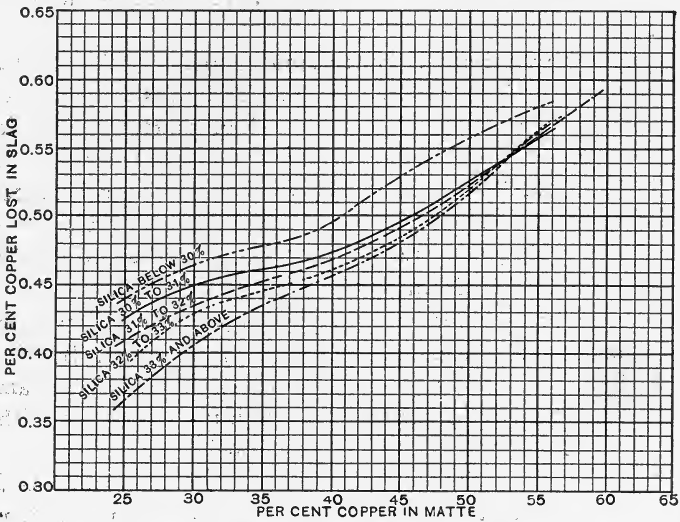

FIG. 2. COPPER LOSS CURVES FOR DIFFERENT SLAGS.

Based on adjusted slag curves and assumption that total amount of slag increases in direct ratio to silica added in form of quartz, as follows:

30% Silica = 10,000 slag.
31% Silica = 10,145 slag.
32% Silica = 10,290 slag.
33% Silica = 10,447 slag.
34% Silica = 10,606 slag.

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[1] Reprinted from *The Engineering and Mining Journal*, Mar. 10, 1904, p. 395.

CONTRIBUTION BY E. A. WEINBERG.

The Editor:

Sir—I venture to contribute a few observations to the discussion on this subject, which was resumed by Mr. Godshall's letter in your issue of February 4, and followed on February 11 by the contributions of Mr. William A. Heywood and Mr. G. F. Beardsley, respectively.

Concerning the designation 'pyritic smelting,' it seems immaterial whether we continue using the term or not, but in my opinion it would serve no purpose to alter a technical term, which as such has found a place in many metallurgical text-books and has become established by common usage. Its definition, according to recent experiences, has been altered to the extent that in future we shall not regard the hot blast as one of the *essential* features in the direct smelting of heavy sulphide ores for matte. Mr. Godshall briefly defines 'pyritic' smelting as a form of matte smelting, whereby the *maximum* amount of heat is derived from the combustion of heavy raw sulphide ores in a blast furnace, in addition to the *minimum* amount of necessary heat obtained directly from carbonaceous fuel, and I believe this definition will be acceptable to most metallurgists.

Whether hot or cold blast should be used will probably remain a question to be determined by the special requirements of any particular proposition, though recent experiences strongly point to the fact that an increased air supply for the rapid oxidation of the sulphur and iron is all that is required to bring about the desired conditions.

With lower blast-pressures and lower charges it may be possible that the hot-air blast has an initiative action on the combustion of the sulphides, but, after all, it is not the temperature of the air, but its volume, which is responsible for the reaction in pyritic smelting, and for the remarkable increase in the tonnage treated. Any metallurgist, who has had practical experience with matte smelting in blast furnaces, will be able to recall many instances when his furnaces would 'drive' to such an extent that often the pots available were insufficient to cope with the slag flow, and the air-supply to the furnaces had to be reduced in consequence. With larger furnaces, better blowers, and a better equipment for the handling of large quantities of material, our aim to-day is to obtain just those conditions which, not many years ago, were accidentally produced, and almost considered objectionable.

When we compare the results of the experimental furnace run No. 5 with those obtained in the run No. 4, Mr. Heywood's figures are remarkable for the enormous increase in the tonnage treated. A few explanatory remarks from him regarding the duration of these experimental runs would no doubt be welcome.

I hope also that Mr. Beardsley, as late metallurgist at Mt. Lyell, will favor us with a more conclusive statement regarding the work done at Mt. Lyell with hot and cold blast respectively. He already informs us that the recent practice at Mt. Lyell has been to discontinue the hot blast, from which it would appear that the management found it profitable to do so. But even if the blast could be heated by waste-gases, as suggested by Mr. Beardsley, and all other conditions remained the same, it would be of great interest to know his experience with regard to the practical gain of using heated air under these circumstances.

Mr. Godshall quotes Mr. J. W. Nesmith's comparative statement giving the results of a test run at Silverton, Colorado. The figures are not conclusive, in so far as no attempt was made to ascertain whether an increased air-supply alone would have brought about an increase in the furnace capacity.

Owing to a long absence from the United States, I am not competent to discuss the question concerning the inducements which central localities like Denver, Pueblo and Salt Lake may offer to a matte smelting establishment to compete with lead smelters in the market for dry ores. The question, in my opinion, is certainly not to be settled by the mere cost of smelting per ton of ore, as in addition, cost of refining and realization charges would have to be taken into consideration.

E. A. WEINBERG.

London, March 2, 1904.

CONTRIBUTION BY WALTER E. KOCH.

The Editor:

Sir—Some time ago I began to remodel our smelter and this involved taking out a small heater and putting in a larger one preparatory to erecting a larger furnace in place of one of my 48-in. diameter furnaces. Meanwhile one 48-in. furnace has been running on hot blast and the other on cold. Here are two furnaces running under exactly the same conditions as to size, air, ores and fuel—two 48-in. diameter water jackets almost exactly alike. The cold-blast furnace has the advantage of a more powerful blower and the blast can be raised to 4 lb., while hot-blast furnace gets less than half that pressure; moreover, the cold blast furnace is newer and has a better arrangement of tuyeres. It seemed to me that there was a good chance to compare hot and cold blast results under similar conditions, with results that should be valuable.

The two furnaces are still running side by side. I send you, herewith, results for last month, March. As far as we are concerned the hot-blast furnace makes profits and the cold-blast comes perilously near making losses, so great is the difference. I can only say that during this run it made our American superintendent hustle to keep the cold-blast furnace going, while ordinary Mexican labor made the hot-blast furnace run comfortably. We had to select both ores and coke for the cold blast, while the hot blast took the leavings without 'kicking'; our superintendent, at the end of the first run, remarked, "If we select the ores and coke and charge and watch it very carefully and give it plenty of coke, we can get along, but if we have trouble in the furnace or stoppage of blast, it takes a long time to come around, nor can we run the charges which the hot blast furnace will take." On one occasion a change of ore during the night made trouble in both furnaces; the hot-blast one was convalescent and working normally in an hour, the cold blast nearly froze up, and it took almost a day to come round. At critical periods the hot blast blows the furnace in and the cold blows it out. You can put your hand into the cold-blast furnace and pull out pieces through the tuyeres, and there are long noses to them often, which is never the case when using hot blast, indicating that the zone of heating effect is larger with the latter. Charges of ore which will work well with hot blast, and slags which run fluid and clean with hot blast, will not work out or run at all with cold blast; in fact, with cold blast you are limited as to charges and slags, and the increase in coke required is a serious item; moreover the coke must be selected and good. We simply could not afford to run cold blast on our low-grade ores, and I can see now how cold blast has spoilt many a good pyritic smelting proposition, especially with dear, and often dirty, coke, and a limited selection of ores. The oxidizing power of cold blast is far less than that of hot even after increasing pressure of blast, and there is a great difference in the power of concentration. We get far better and quicker concentration with hot than with cold blast. In every way the advantage remains with the hot blast.

Last week we had our first freeze-up in over two years, and that was on the cold-blast furnace; the hot-blast furnace went right along, although it ran much slower than usual and the slag was less fluid and the production less. The trouble came, as usual, during the night, with a change of ore in the pile, and although we worked hard, the cold-blast furnace died on our hands. The hot blast also suffered, but the Mexicans pulled it straight in a couple of hours. I cannot for the life of me see why any one should advocate cold blast; it seems to me like 'smelter suicide.' It is certainly like fighting with one hand tied behind your back. It costs nothing beyond the first cost of heater, as in our case the furnace itself supplies the heat and the heater is a most efficient dust-catcher. Our hot blast never runs over 400° F. or 204° C., and seldom reaches that. It should be called "warm blast."

The details of our results for the month of March were as follows:

MARCH 1 TO EASTER FIESTA.

	Furnace A.	Furnace B.	
	Hot blast.	Cold blast.	Hot blast.
Time of run......	22 days	16 days	18 days
Ore charged per 24 hours.......	41 tons	*31 tons	42 tons
Coke charged per cent. of charge.	6¼ to 6½	*8¾ to 9	5¾ to 6
Coke charged per ton of ore	150 lb. or 7½%	220 lb. or 11%	140 lb. or 7%
Percentage of ore in charge.......	84	81	85
Concentration....	up to 15 to 1, good	below 10 to 1, poor	up to 15 to 1, good
Aver. slag, 45%, SiO_2, 37% FeO, 8% CaO and 8% Al_2O_3.....	....	less ore and more flux	previous run before change

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* Coke and ore were *selected* for cold blast; hot blast got the leavings and did not object.

Further details of the furnaces may be useful: Height from charging floor to base plate, 84 in.; height from charging floor to center of tuyeres, 57 in.; height of center of tuyeres above base-plate, 27 in. Each furnace is 48 in. diameter and has twelve 3.5-in. inclined tuyeres (pitched downward). The heater for furnace B was erected for a 42-in. diameter furnace and is too small. A larger one is going up now for a 126-in. by 42-in. furnace to displace B, which in turn will displace A and have a larger heater. A has a MacDonald heater, heated by waste gases from furnace. Both furnaces get about 4,000 to 4,500 cu. ft. of air per minute, at about ½-lb. pressure at tuyeres, while B can be run up to 4 lb. at tuyeres. B is a newer and improved furnace in a few details, notably in arrangement of tuyeres, and with hot blast can beat A by a good margin.

I do not consider our coke consumption as low as it should be, and will be, I hope, later on. It should be nearer 5 per cent than 7 per cent per ton of ore, or 4 to 6 per cent on the total charge, and 10 per cent is quite disgraceful in any furnace matting copper ores.

If loss of time, output, temper and sleep could be put against furnace B on cold blast it would be a large account. No more cold blast for me, although I am still running B on cold blast till the new furnace is ready to erect; while we get tonnage and keep our men together, it cuts into profits; the cost of coke being over $20 gold per ton at smelter. Furnace B uses over 50 per cent more coke on cold than on hot blast, with smaller charges of the same ores.

WALTER E. KOCH.

Santa Maria del Oro, April 29, 1904.

CONTRIBUTION BY W. H. FREELAND.

The Editor:

Sir—In response to your request to supplement my contribution upon this subject, I take pleasure in stating that the experimental work of the Ducktown Sulphur, Copper & Iron Company, Ltd., carried on at intervals, for several years past, was concluded by the test campaign, the detailed data of which appeared in your issue of May 2, 1903. Following that, the last of its stock of roasted ore was cleaned up towards the end of September, 1903, and since then its whole furnace plant has run exclusively and constantly on raw ore. Almost 70,000 tons of raw ore have been smelted in the past seven months, with less trouble and interruption than records show for any similar period and tonnage on roasted ore.

The conditions and results of the commercial practice tally so closely with those of the experimental campaign as to confirm and verify (literally) the data already published. The only important modifications of those data are: A very material gain in furnace's capacity; the use of slag on the charge has been entirely discarded; and a heavier blast is used.

Furnace Capacity.—Referring to this feature in my original article, it will be noticed that a loss of efficiency of 48 per cent was shown on furnace's performance on raw ore, as compared with roasted ore. The same furnace is now averaging 110 tons of raw ore per day; deducting the time necessary to re-concentrate the matte from 'first operation' gives an average capacity (including both operations) of 96 tons raw ore per day, or a loss of efficiency of only 16.5 per cent instead of the 48 per cent originally quoted.

Preparations are being made to further increase the blast, which, when completed, will still further increase the furnace tonnage.

Opinions of the Process.—Considerable interest in the Ducktown Sulphur, Copper & Iron Company's practice having been expressed by others, the following may be quoted from the *Boston News Bureau* of November 16, 1903, as Mr. J. Parke Channing has had numerous opportunities of seeing and studying the operations: "According to President Channing, of the Tennessee Copper Company, Mr. Freeland has admirably succeeded in these experiments, and his plant is now running entirely on raw ore, and making both a technical and commercial success. It is solely as a result of his successful experiments that the Tennessee Copper Company decided to adopt the same methods."

Comparative Economy of the Process.—The plant and equipment of the Ducktown Sulphur, Copper & Iron Company is not of modern design, yet under its conditions, a reduction of 3.018c. in the cost per pound of copper is effected by raw sulphide smelting. The writer is of the opinion that an up-to-date plant should make an even better showing, and it is to be hoped that our immediate neighbors, the Tennessee Copper Company, with their admirable plant, will confirm that opinion in the near future, as that company is now making preparations to adopt the same methods.

W. H. Freeland.

Isabella, Tenn., May 2, 1904.

CONTRIBUTION BY AMADO BUEN.

The Editor:

Sir—In reading the various articles on matte smelting I notice that though hot blast is generally conceded to increase the capacity and to give cleaner slags and better conditions in general, the stoves are of low efficiency and troublesome to keep in repair. As the furnace gases cannot be used for fuel, brick stoves seem not to have been tried at all, as they need a gaseous fuel. It appears, however, that for conditions like southern Arizona, where petroleum is used to heat boilers instead of coal, petroleum might just as easily be used to heat a brick stove, when all the trouble coming from cast-iron stoves would disappear, and a plant might in this way largely increase its capacity and decrease its fuel consumption. From what I have observed at smelters in southern Arizona, success in running a furnace with a top fire depends almost entirely on judgment in feeding the furnace. I have seen at one plant how the blowholes, that always form with high-pressure blast, are fed with slag by hand, with the intention of keeping the furnace open, and the result is that by feeding an easily fusible material into that part of the furnace which is open, the rest is chilled and the furnace is always in a bad condition and full of crusts. Also, at the same plant, the weight of the charges, ore and coke, has been decreased, with the result that the coke does not get a sufficiently thick cover to keep it from burning out before it reaches the fusion zone. As the plant is new and the method of charging is directly from the cars, which dump sideways into the furnace, this arrangement has been blamed for the unsatisfactory running. At a neighboring plant, however, though the material is brought up in big two-wheel barrows by hand, the dumping into the furnace is exactly the same, almost no hand-shoveling being done. Only the charges are larger, and special care is taken to cover the coke up immediately with a large amount of ore, and to put ore and not slag on top of the blow-holes, while the slag is put to those places which show the least heat, and the furnace keeps in good condition all over.

AMADO BUEN.

Douglas, Arizona, May 21, 1904.

PYRITE SMELTING.—A REVIEW.

By EDWARD D. PETERS.

INTRODUCTORY.

At the request of the Editor, I have undertaken to review and comment upon a series of papers on the subject of pyrite smelting which have appeared in the columns of *The Engineering and Mining Journal* between October 10, 1903, and the present time.

Pyrite smelting is a process of very recent development, and its published literature is meagre. A large proportion of the metallurgists best qualified to write authoritatively upon the subject, are actively engaged in practising and developing it, and have no time to write treatises. Messrs. W. L. Austin, F. R. Carpenter and Herbert Lang had previously given us much valuable information as to the results obtained in their own practice; but there were many other metallurgists doing highly original and successful work in this field, from whom it was deemed desirable to hear. The editor of the *Journal*, therefore, sent out last autumn a series of ten questions to a considerable number of the metallurgists of the world who are, or have been, engaged in this branch of the art. The replies to these questions constitute a body of information on the subject of pyrite smelting that I believe to be unique in the metallurgy of copper, both for its comparative unanimity of opinion, and for the authority of the sources from which it comes.

It is the object of this review to present, in a concise and graphic form, the results and opinions reached by these authors; to draw general deductions from

QUESTIONS.	L. S. AUSTIN.	G. F. BEARDSLEY.	S. E. BRETHERTON.	F. R. CARPENTER.	C. H. FULTON.
No. 1. *What types of ore are suited to the process?*	Sulphide ores of Fe and Cu in quantity sufficient to give required matte-fall. Silicious precious-metal ores to form suitable slag. Oxidized ores of Fe and Cu.	Pure pyrite and pyrrhotite pre-eminently, with sufficient SiO_2 and earthy bases to form a suitable slag. Physical condition of ore important.	Types of ore containing Au, Ag, Cu, and Pb, unless there is Pb enough present for lead smelting. Even ores tolerably rich in Pb, if containing As and Sb.	Any Au, Ag, or Cu ores not carrying Pb sufficient for lead smelting. SiO_2 or CaO must be added where needed to make a suitable slag.	Any ore mixture containing sufficient S to form suitable matte with Fe and Cu, and with sufficient SiO_2 to form a rather high silicious slag.
No. 2. *Is hot blast advisable?*	Hot, or even warm, blast is a great aid. If properly installed, probably of great advantage in smelting silicious or comparatively infusible charges.	Advisable, and in some cases necessary, especially with a silicious slag. Hot-blast pipe stoves are a most wasteful apparatus for heating air.	Emphatically, yes, for the following reasons: 1. Increased capacity. 2. Saving of fuel. 3. Less trouble with tuyeres. 4. Better elimination of Zn and As.	Desirable, but not an absolute necessity.	Have not had sufficient experience with a *hot* blast to compute saving of fuel. It increases oxidizing effect and yields a richer matte.
No. 3. *To what extent can fuel be eliminated?*	With plenty of sulphide ores, coke has been cut to 6% and less, Rapid running needs less coke than a slow-running silicious charge.	With heavy pyritous charge, proportion of coke to charge may vary from 0.3% to 3%, being dependent on manner in which the sulphides are distributed through the gangue.	Usually not advisable to reduce coke much below 4%, though have run for days on 3%. (I understand Mr. Bretherton's coke contains about 24% ash.—E. D. P.)	All carbonaceous fuel can be eliminated, and it is desirable that it should be eliminated in ideal, or true, pyritic smelting.	Unanswered.
No. 4. *What amount of copper is required for the collection of the precious metals?*	A charge containing sulphide ores, free from Cu, will not yield a clean slag; but as little as 0.5% Cu will suffice if other conditions are favorable.	My own experience has been limited to a minimum amount of 1.5% Cu in the charge. This gave clean slags.	Depends on character of ore, especially where much Zn is present. I don't like to go below 1% Cu in the charge, and prefer 3%.	Depends somewhat on degree of concentration. Where very little matte is made, it should contain, perhaps 10% Cu. Cu may be entirely unnecessary, with both Au and Ag and fair matte-fall.	At Rapid City, an iron matte alone failed to collect the Au and Ag with entire satisfaction. We now arrange to have the charge contain at least 10 lb. Cu for each oz. of Au present. Matte-fall = 4 to 5% of charge.
No. 5. *What percentage of lime is necessary to a clean slag?*	CaO is used as a base to supplement the FeO, rather than to cleanse the slag, though it is thought to have that effect. Clean slags are made as low as 5% CaO.	The copper matte—*not* the lime—is the slag-cleanser. Enough CaO or other earthy bases, perhaps 13%, should be used to lower the specific gravity of the slag.	The "cleaning" effect of lime mainly due to its lowering specific gravity of slag, which is also equally well effected by other earthy bases.	Is used merely to lower specific gravity of the slag, and, in certain cases, to increase its fusibility.	Lime is necessary for the production of a slag out of which the globules of matte will settle rapidly, especially if the matte also has a low specific gravity.

QUESTIONS.	L. S. AUSTIN.	G. F. BEARDSLEY.	S. E. BRETHERTON.	F. R. CARPENTER.	C. H. FULTON.
No. 6. *What percentage of zinc in the charge can be treated profitably?*	Zn up to 10% of the charge has been successfully treated. In presence of much Al_2O_3, Zn makes continued trouble.	Unanswered.	In practice, I should put the maximum amount of ZnO in slag at 11%, and the FeO must be proportionately reduced. There should be a minimum of 10% of earthy bases.	Unanswered.	Unanswered.
No. 7. *What is the degree of desulphurization attainable?*	In raw sulphide smelting, where pyrite is used, at least 66% of the S is driven off.	In smelting heavy pyrite ores, the desulphurization would be 80 to 90% of the S present.	Unanswered.	Depends upon conditions. At the Deadwood (Au and Ag) smelter I have often oxidized *all* of the pyrite, making no matte.	At Rapid City, 75 to 80% of the S is removed in the smelting, and I think still higher desulphurization could be attained if desired.
No. 8. *What are the possibilities as to capacity of furnace?*	A silicious charge or one containing much Zn or Al_2O_3, runs slowly. A basic, fusible and coarse charge runs rapidly, for instance in the Boundary district, where 450 tons per 24 hours have been smelted.	On heavy pyritous ores, with 40 to 45 oz. blast, the 42 by 210 in. Mt. Lyell furnace has smelted a charge of 530 tons in 24 hours.	Depends on varying conditions. With hot blast, a furnace will smelt fully one-third more than with cold air.	Capacity of pyritic furnace not over half as great as on roasted ore. A 36 by 192 in. furnace, producing a 40% SiO_2 slag from silicious pyrite and limestone, averages 250 tons per 24 hours.	Much less than on roasted ore. A 38 by 144 in. furnace smelts 150 tons daily, there being considerable fine.
No. 9. *What are the limitations of the process?*	Depends on proper supply of sulphide ore, and available Cu in the charge, and on skill in running the furnace; it being much more difficult than smelting roasted ore.	Want of sulphides in the ore, and excessive amounts of Zn and Pb.	See reply to No. 1 question. (No. 1 question was answered very fully by Mr. Bretherton in his original letter.—E.D.P.)	Practically no limitations if the ores do not carry Pb.	See reply to No. 1 question.
No. 10. *What is the relative economy as compared with rival processes?*	Unanswered.	It has no rival where suitable. Great rapidity. From moment of feeding the ore into blast furnace to pouring converted pig copper into moulds is about 5½ hours.	Answered in my replies to the preceding questions.	It saves roasting. It saves fuel in smelting. It permits the production of highly silicious slags.	A serious rival of other smelting processes, of water concentration, and even of chlorination and cyanidation, even for low-grade ores, on account of the high saving made.

QUESTIONS.	W. E. KOCH.	HERBERT LANG.	R. L. LLOYD.	E. P. MATHEWSON.	W. H. NUTTING.
No. 1. *What types of ore are suited to the process?*	All ores are suitable, so long as there is sufficient S and Fe.	Any Au, Ag, or Cu ores free from paying quantities of Pb, and that can be so mixed or fluxed as to form a proper slag and matte, and furnish heat.	Any ores whose values consist in Au, Ag, or Cu, and where sulphide ores can be advantageously obtained.	Any ore carrying S in the shape of pyrite, chalcopyrite, or pyrrhotite.	Iron sulphide ores—the purer the better—combined with silicious ores. ZnS objectionable. PbS does not interfere with the process, but loses the Pb.
No. 2. *Is hot blast advisable?*	In our work, a warm blast (200° C.) meant success; a cold blast failure. No man who has not used a hot blast can appreciate its advantages.	My later experience is altogether favorable to the use of the hot blast. It saves fuel and increases capacity.	I should say emphatically, yes, for nearly every reason: Saving of coke. Increased capacity. Better desulphurization.	Advisable where it is desired to increase the grade of the matte produced in ores not too rich in sulphides. Also where coke is expensive.	Metallurgically advantageous, but might not be worth while where coke is cheap. Presence of ZnS and of fine would also affect decision.
No. 3. *To what extent can fuel be eliminated?*	We have found 5% coke the lowest practicable limit. We generally use 7% of the weight of the ore charged.	Varies with conditions. Hot blast may save ⅓ to ⅔ of the coke. Smelting heavy sulphides with cold blast usually means little coke and a low ratio of concentration.	Varies with conditions. With Butte ores, containing a considerable amount of coarse raw sulphide concentrates, it is common to use a charge of 92% ores and fluxes, and 8% coke.	May be entirely omitted if sufficient pyrite is present. Probably 4% coke is as low as anyone has reached in the United States.	With a coarse charge and suitable pyritic ingredients, coke can be eliminated in proportion to the absence of fine and ZnS, down to the intermittent use of 2 or 3%.
No. 4. *What amount of copper is required for the collection of the precious metals?*	We make clean slags from a charge containing 0.5% Cu, ½ oz. Au and ¼ oz. Ag per ton. Ratio of concentration, 15 into 1. Matte contains 7 to 8% Cu.	A little matte high in Cu gives cleaner slags than a greater amount of matte low in Cu. To use an iron matte, very low in Cu, the slag should contain a considerable proportion of earthy bases.	A reasonably close gold collection can be made with a matte free from Cu. To make a matte high in Au and Ag, it might be necessary to have 1.5 to 2% Cu in the charge.	Copper is not necessary for the collection of the precious metals. They will collect readily in an iron matte.	Charges containing considerable ZnS require 1 to 1.5% Cu to maintain sufficient difference in specific gravity between matte and slag. Otherwise, a fair collection can be made by Fe matte alone.
No. 5. *What percentage of lime is necessary to a clean slag?*	Seven per cent. lime is enough with hot blast.	Perhaps the cleanest slag that could be made would be ½ bisilicate of CaO and ½ singulo-silicate of FeO; but acceptable slags can be made with FeO as almost the sole base.	Dependent upon what other bases may be present. Can make tolerably clean slags with merely metallic bases.	Unanswered.	Amount of CaO necessary is governed by amount of available Fe for slag formation, this depending on amount of Fe carried into the matte.

QUESTIONS.	W. E. KOCH.	HERBERT LANG.	R. L. LLOYD.	E. P. MATHEWSON.	W. H. NUTTING.
No. 6. *What percentage of zinc in the charge can be treated profitably?*	ZnS is a very good fuel, but I do not know how much we could profitably use.	Unanswered.	A proportion of 10% ZnO in the slag is usually considered the maximum for fast driving.	Unanswered.	Difficulties begin at 3%, eventually blocking furnace at 12%. ZnS demands increased fuel, causes imperfect separation of matte and loss of Ag by volatilization.
No. 7. *What is the degree of desulphurization attainable?*	About 80% of the S in the charge is burned off, but I think we can do better.	Pyritous ores with a not very silicious slag may lose 60 to 70% S. With earthy bases and more SiO_2, 80% and more. Have burned off 96% from ore containing 11% S.	On Butte ores, with cold blast, 70 to 80% desulphurization is easily attainable.	75 to 80% on Butte ores.	60 to 70%, which would be increased by a more silicious slag, and slower smelting.
No. 8. *What are the possibilities as to capacity of furnace?*	Have not yet determined. I am averaging 50 tons per 24 hours in a 48-in. round furnace.	Unanswered.	Conditions vary. I consider 7 to 7½ tons smelted per sq. ft. of hearth area as reasonable work.	Capacity best regulated by ability to feed the furnace and handle the slag. We consider 600 tons per 24 hours the capacity of our 56 by 180 in. furnace.	Capacity largely modified by three factors: 1. Amount of flux required. 2. Per cent of fines present. 3. Per cent of ZnS present.
No. 9. *What are the limitations of the process?*	A lack of sulphur and iron.	If constituents of charge be rightly proportioned, there are no *metallurgical* limitations for ores of Au, Ag, or Cu.	Limited solely by economic considerations.	Too fine or too clayey ores. Also, as to whether it can effect a satisfactory ratio of concentration.	Competitive methods, or combination of methods, showing greater economy, governed to a great extent by fines, ZnS, and cost of coke.
No. 10. *What is the relative economy as compared with rival processes?*	No process can save practically all the Au, Ag, and Cu values in our ores so simply and so economically as hot blast pyritic smelting.	(Impossible to give a just abstract of Mr. Lang's full and thoughtful reply to this question. See page 51 of this work.—E. D. P.)	Where conditions are suitable, it has no successful rival.	Where this process can be used at all, it is practically the cheapest known.	Less costly plant. No roasting. Rapidity. Silicious and clean slags. —E. D. P.

them, so far as seems justifiable; and, incidentally, to comment on and discuss such points in them as seem to invite further consideration. Ten of the replies answer all, or nearly all, of the questions categorically, and thus lend themselves to the construction of a table, where a bird's-eye view may be had of the opinion of each metallurgist on each separate question. It has been difficult to express in a few words an opinion that may, perhaps, have occupied an entire column in the original reply, and I must ask the indulgence of the authors.

The results of this tabulation are striking, and it is interesting to note how little difference of opinion there is in the replies. Indeed, where differences of opinion exist at all, it is nearly always due to the fact that some of the individual questions cover so much ground that they admit of a good deal of variation in the answer, without there being any actual conflict. It is simply that they have been answered from a different standpoint, and that each author tends, naturally, to base his reply upon the particular class of ore that he is working, and upon which much of his experience is based; for pyrite smelting, in its modern form, is of too recent origin to admit of a very wide or varied experience on the part of any one person.

Thus, as a typical instance where conditions compel two differing answers to the same question: Mr. G. F. Beardsley has for years been smelting the unusually heavy pyrite ores of the Mt. Lyell mine, in Tasmania, using enormous furnaces, and a high blast-pressure, and, owing to the great preponderance of basic, ferruginous ores, has been compelled to produce a slag as high in iron and as low in silica as was compatible with the desired ratio of concentration. Consequently, his replies to certain of Mr. Rickard's questions must, necessarily, be widely different from Mr. S. E. Bretherton's, who, at his smelter at Val Verde, Arizona, is almost always short of pyritous material; whose coke costs $13 per ton; whose profit arises almost entirely from the smelting-tariff on silicious gold and silver ores, containing more or less arsenic, antimony, lead and zinc, all of which he has to completely remove in the pyritic fusion, so that his matte may not be penalized by the refiners for these substances; and who has no copper in his charge, excepting the little that he is forced to add to collect the precious metals, and which he has to bring by rail from a distance of some 500 miles.

Here are certainly conditions as diametrically opposed to each other as one can well imagine; yet a careful perusal of the two vertical columns of answers in the table, headed by the names of Beardsley and Bretherton, will show that these two highly experienced metallurgists have arrived at conclusions that are almost identically the same in nearly every respect. There are some points in each set of answers that are not touched upon in the other. But there is no common point on which the two differ materially. In no instance do their replies contradict each other; they merely supplement each other. This same unanimity of conclusions will be found to prevail pretty generally throughout the entire one hundred answers.

A similar agreement in experience and deductions will be found in the letters of Messrs. Ingalls, Weinberg, Godshall, Heywood, Read and others, which form a part of this same discussion, and of which I shall equally avail myself in this review, although it was impossible for me to include them in the table without taking greater liberties with the text than would have been justifiable.

When a dozen or more practical and experienced metallurgists agree upon a technical point, we may feel safe in accepting it as a fact—at least, until it is modified by new evidence. In these replies, there *is* such an agreement to a very remarkable extent, and we have here the material for the construction of a more solid basis of information in regard to pyrite smelting than we have hitherto possessed, though many of the views enunciated in these letters have been foreshadowed. and, in some instances, distinctly stated from time to time in articles by Messrs. W. L. Austin, Carpenter, Lang and others.

I. Types of Ore Suited to the Process.

In the series of questions sent out by *The Engineering and Mining Journal*, the first in order is: 1. What types of ore are suited to the process? So much of the same ground is also covered by the ninth question, "What are the limitations of the process?" that the two can be advantageously considered together.

The fact that the questions refer to pyrite smelting implies the presence of a certain proportion of sulphides in the charge; else it would not be pyrite smelting. Consequently there is little variation in the replies to these two questions.

Pre-supposing the presence of sulphides, the general, or composite, answer might be: "Any mixture of gold, silver, or copper ores that will yield a fusible, and rather silicious, slag." Certain of the replies indicate some limitations, and it is to these that our attention must be mainly directed.

The presence of a considerable proportion of fines in the charge is mentioned by several of the contributors as being objectionable; as, indeed, it certainly is in ordinary blast-furnace smelting. In pyrite smelting it may be particularly dangerous in cases where the silicious portion of the charge is comparatively coarse, while the raw sulphides exist in the shape of fines. Under such circumstances, the sulphide fine will trickle down through the interstices of the charge, and, owing to its fusibility, melt into matte without remaining long enough in the oxidizing atmosphere above the tuyeres. This circumstance produces three serious evils:

1. An excessive formation of low-grade matte.
2. A dearth of iron with which to flux the silica.
3. A waste of fuel; the sulphides going into the matte instead of burning so as to supply much of the heat required for smelting the ore.

There are two obvious remedies for this trouble, both of which I have, at various times, employed, or seen employed. One is the wrong one; the other, the right one. The wrong remedy is to attempt to supply the deficiency in heat by adding more coke. This is the step usually first taken by men who are inexperienced, and is quickly fatal. The trouble that already exists in the furnace is, that not enough pyrite is being oxidized to furnish ferrous oxide with which to flux the silica. The addition of more coke fills the furnace shaft with powerfully reducing gases, and prevents the oxidation of even that small proportion of pyrite which was being changed into ferrous oxide. The result is, of course, that, under the new conditions, even this little oxidation ceases, practically *all* of the sulphides go into the matte, and the slag becomes too silicious to melt at all.

The right remedy is to change the physical condition of the sulphide fine, so that it shall not trickle down through the interstices of the charge. This can be accomplished in two different ways, either of which is reasonable, though only local conditions can determine which is the more economical.

1. Mixing the fine with lime, and briquetting it under pressure, which is perfectly feasible for raw sulphides.
2. Fusing into an iron matte in a large reverberatory furnace. The capacity of such a furnace running on raw sulphide is enormous, while the fuel consumption is very small, and, as the main result of this operation is

simply to change the FeS_2 into FeS, there is comparatively little loss in heat-producing constituents. The weight of the raw sulphide is considerably reduced, and the resulting iron matte forms a valuable fuel and flux in lump form. The fact that it makes no flue-dust, when added to the blast furnace charge, also tends to reduce the cost of this reverberatory fusion.

A large percentage of zinc-blende in the charge is referred to by several of the contributors as being particularly objectionable in pyrite smelting. As the sixth question deals exclusively with this unwelcome mineral, I will omit all discussion of it under the present heading.

None of the replies object seriously to the presence of lead sulphide, except to point out that it is wasted in the pyritic furnace, the lead being mostly volatilized or slagged.

Mr. L. D. Godshall calls attention to two conditions where the ordinary smelting of roasted ore might prove more advantageous than pyrite smelting. These are: (1) Where the smelter will not, in a single fusion, give the desired degree of concentration; and (2) where there is a large amount of zinc-blende in the charge.

To these two conditions I will add still another: (3) Where sufficient silicious ore cannot be obtained, and it becomes necessary to smelt with a rather basic slag.

Mr. Godshall's first condition really includes mine, and I have only made this apparently superfluous addition in order to bring out separately and emphatically what I think may be called the most important maxim in the art of pyrite smelting; namely, *that, other things being equal, the degree of concentration increases with the percentage of silica in the charge.* (This point will be taken up when discussing the seventh question.) Consequently, if we have heavy sulphide ores, and cannot profitably obtain sufficient quartz (and it must be, at least in part, *free silicc*, and not feldspar or clay or garnet) to form a tolerable silicious slag; and if these ores are not rich enough in copper to give us, even with our probable low rate of concentration, a 45 to 50 per cent matte on the first smelting, we shall be confronted with the necessity of putting this matte through a second pyritic fusion, in order to bring it up to a suitable grade for converting.

A concentrating smelting of the raw matte is not at all a serious matter where our ratio of concentration has been high; because, under such circumstances, the cost per ton of original ore will be very small. Indeed, in a silicious district where there is a high smelting tariff on quartzose ores, this cost may be in great part offset by the profit obtained from the silicious ores that we smelt in connection with the raw matte, and which it is absolutely necessary to smelt with it unless we desire to see it come out of the furnace in pretty much the same condition as when it went in.

On the other hand, when the ratio of concentration on the first smelting has been low, this second fusion of the raw matte constitutes a serious charge against each ton of the original ore, and it is in these very cases that we also usually fail to obtain any profit from the silicious ores that it is necessary to use in the concentration-smelting of the matte. This arises from the very simple reason that, if it had been a district where there *was* a good smelting profit on silicious ores, we should have used enough of them in the original ore smelting to have formed a silicious slag that would have given us a satisfactory rate of concentration in the first place.

I therefore agree entirely with Mr. Godshall that there may be certain conditions where sulphide ores can be more economically treated by roasting and smelting than by pyrite smelting. I believe, however, that these conditions are mainly limited to districts where silicious ores are scarce.

Mr. Godshall's second limiting condition refers to ores containing too much zinc-blende. It is pointed out both by Mr. Godshall and Mr. Ingalls that zinc-blende is much more easily smelted if it has been oxidized outside of the blast furnace, than if both oxidizing and smelting are conducted in the same operation within the furnace. No doubt all metallurgists will agree with this statement, and the main point to determine for each individual case would be, at just what percentage of blende it would be more advantageous to give up pyrite smelting and submit the ore to a preliminary roasting. A fuller discussion of this matter will be deferred until we reach the sixth question, which deals exclusively with zinc.

It seems to me that, in replying to the first question, Mr. Lang goes to the root of the matter. He says he should like to modify the question so that it might read: "What kind of *ore mixture* is best suited to the process? One ore is as well suited to pyrite smelting as another, provided its values are in gold, silver, or copper, and that it be smelted with other ores and fluxes that make up its deficiencies as a builder of slag and matte, and a furnisher of heat."

In speaking of the necessity of always having enough silica (in excess of that needed to form a bi-silicate of lime and a singulo-silicate of iron) to decompose the iron sulphides, and thus effect the required degree of concentration, Mr. Lang emphasizes the important point that the native silicate minerals, that we find so commonly in ores, probably do not exert this decomposing influence upon the sulphides; but that *free silica* is required for this purpose. That, therefore, these native silicates—such as garnet, hornblende, and the alumina silicates—may, of course, be smelted incidentally to the process, and will aid in rendering the slag silicious, but do not directly assist in the decomposition of the sulphides.

I will mention another class of ores that seems to be peculiarly adapted to this process; not because the pyrite smelter has any particular affection for them, but because they are very much less objectionable in the pyritic furnace than in the ordinary blast furnace with coke. I refer to ores containing a large proportion of barite, which in ordinary blast-furnace work forms an undue amount of matte of low specific gravity, and tends to reduce greatly the capacity of the furnace, and to increase the losses in the slag. In the pyritic furnace, when using a reasonably acid slag, barite is mostly decomposed, the BaO entering the slag as a valuable base, whilst the sulphur trioxide escapes through the stack.

It seems to me that we may justly draw the following deductions from the replies to Questions 1 and 9: All ores, or mixtures of ores, containing copper, gold or silver are suitable for pyrite smelting, provided they contain the constituents for the formation of a suitable matte and slag, and do not contain too large a proportion of zinc-blende. (See also Question 6.)

The conditions where roasting and smelting might be more economical than straight pyrite smelting are:

1. With heavy pyrite, and insufficient profitable silicious ores.
2. Where too much zinc-blende is present.

2. USE OF HEATED BLAST.

Nine of the ten tabulated replies to the question, 'Is hot blast advisable?' are in the affirmative. Five of them are emphatic. The remaining four speak of hot blast as advantageous rather than essential. The tenth reply regards it as more or less advantageous, but attaches much less importance to it than do the others.

The trouble with this question is that it covers too much ground. Where there is a great abundance of iron sulphides, and a consequent ample generation of heat from their oxidation, the advantages of a hot blast are, of course, much less apparent than where there is an earthy and silicious charge, with a moderate amount of pyrite, and yet where a high degree of concentration is required. (This condition occurs most often in the dry smelting of gold and silver ores, and where even unprofitable copper is added to collect the precious metals.)

In the former case, there is an ample exccss of heat to warm a cold blast of air up to the temperature of the smelting zone, and the harder one blows, the more heat there will be, for the more rapidly will the pyrite be burned; the rate of concentration being mainly controlled by the amount of silica added. In the latter case, however, the conditions are quite different. The moderate quantity of pyrite in the charge does not furnish sufficient heat for fusion; and yet the 3 or 4 per cent of coke that is being used to make up the deficiency cannot be increased, or some of the FeS will go into the matte instead of being oxidized; the ratio of concentration will suffer, and the slag will at once become too silicious from the loss of its FeO, which has gone into the matte as FeS. Thus the metallurgist is "between the devil and the deep sea." If he does not add coke, his furnace will chill for want of heat. If he does add coke, his matte will be low grade, and his furnace will chill from too silicious a slag. It is in such cases as this, where the furnace is continuously on the ragged edge of freezing-up, that the hot, or even warm, blast becomes not only advantageous, but indispensable.

Dr. Carpenter comes the nearest of any one to denying the economic advantage of hot air. He has had great experience, and every one will attach much weight to his opinion. After stating that "It reduces fuel to the extent of heat so added, but that the best hot-air stove is a very wasteful machine when heated with extraneous fuel," he adds that he "never has found the magic in mere hot air that others have professed to find." This is not easy to reconcile with his opinion expressed in Vol. X of *The Mineral Industry*, page 698, that "At Golden, Colorado, the Deadwood practice was materially improved by the addition of hot air, without which pyritic smelting should not be attempted."

Mr. Fulton says that he cannot speak from experience, but that he believes that the use of a hot blast notably increases the oxidizing effect of the smelting, and, consequently, improves the degree of concentration.

Mr. Mathewson also refers to the effect of a hot blast in improving the degree of concentration, but, in a private letter to myself, desires to limit this statement to cases where there is not a high percentage of sulphur in the charge, and where the ore column in the furnace is lowered. The wonderfully rapid smelting that Mr. Mathewson is now doing at Anaconda is of just the kind that does *not* require a hot blast, there being a large proportion of heavy sulphide ore in the charge, and the ratio of concentration being unusually low, owing to the high percentage of copper in the material smelted.

Mr. Nutting believes that "metallurgically, the hot blast is always advantageous." He doubts, however, whether it is always economically profitable where coke is comparatively cheap. Here, again, we have conditions where a hot blast is not indispensable. At the Bingham smelter, they have rather cheap coke, ample sulphide ores, and a moderate ratio of concentration.

Mr. Beardsley speaks strongly in favor of hot blast where it is necessary to make a silicious slag, and equally strongly regarding the unsatisfactory duty of U-pipe stoves. It has been found possible at Mt. Lyell to do without them at all, and, by heightening the ore column, and greatly increasing the blast, the furnaces are putting through a very large tonnage, and still oxidizing the sulphides sufficiently to maintain a fair degree of concentration. Here is a typical case of smelting a massive pyrite ore, with the addition of the least amount of silica necessary to produce a suitable slag. The charge is highly fusible, and contains an unusual proportion of heat-producing constituents.

The foregoing are the types of pyrite smelting that obtain the least advantage from the use of the heated blast.

Mr. L. S. Austin considers a hot, or even warm, blast of assistance in raw smelting, but emphasizes its value particularly in smelting a very silicious, or infusible, charge. He also refers to the difficulty so often experienced with the apparatus for heating the blast.

Mr. Bretherton speaks emphatically as to the necessity of a heated blast in pyrite smelting. I have already alluded to the conditions under which he is working, which are diametrically opposite to those of Beardsley or Nutting or Mathewson, and which, I am quite convinced from personal observation, render a heated blast indispensable to his existence at Val Verde as a successful smelter. He seems to have no difficulty with the stove of his invention, which, on my late visit to Val Verde, was consuming, per 24 hours, 2.5 cords of old railway ties to heat the blast for his 160-ton furnace.

Mr. Koch, of La Lustre smelter at Sta. Maria del Oro, speaks strongly in regard to the advantages of the hot blast. He says: "A warm blast of 200° C. is a *sine qua non* with us; it spelled success; cold blast meant failure." I can well understand that, in smelting the silicious ores of the Magistral vein with 7 per cent coke, and making a concentration of 15 into 1, Mr. Koch may, indeed, depend upon the heated blast. It is a typical case for its employment. Since Mr. Koch's first communication on the subject he has carried out a test[1] between two furnaces operating under like conditions except as to blast, and the results thoroughly confirm the opinion previously expressed by him.

Mr. Lloyd also speaks strongly in favor of heated blast.

Mr. Herbert Lang, who is credited with being the first person to demonstrate that pyrite smelting without a heated blast was possible at all, throws the weight of his experience entirely on the side of the hot blast. He says: "I believe that it invariably decreases the cost of treating the ore. . . . 1. It saves fuel. 2. It makes the furnace drive faster and smelt more in a day." He also adds: "The furnaces I have known run faster with hot blast than with cold." He speaks strongly of the defects and wastefulness of U-pipe stoves.

From the testimony on which this review is based, as well as from a number of private communications, it seems doubtful if the pre-heating of the blast effects any improvement in the degree of concentration when smelting a heavy sulphide charge. As this matter of concentration is a very important one, and as there is a large class of metallurgists who have as yet had no experience in this kind of smelting, and feel in doubt as to what types of ores are most benefited by pre-heating the blast, I shall take the liberty of going into the subject in some detail.

I suppose that, in the pyritic furnace, *any* degree of desulphurization can be obtained, by using ample blast, and provided there is sufficient free silica to retard the smelting operation and combine with the ferrous oxide formed by the burning of the sulphides. Theoretically, *all* of the sulphur could be burned off, and even the copper scorified and carried into the slag, if desired. Practically, such a complete oxidation as this would be

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[1] See page 124.

very difficult to accomplish, owing to the lack of heat after the combustible portions of the charge have been consumed.

If, for experimental purposes, we should endeavor to accomplish such an unprofitable feat as this, we would find that the addition of carbonaceous fuel, beyond a certain moderate limit, would nullify the result desired, as it would simply cause the still remaining sulphides to melt down without decomposition, thus producing the very matte that we were trying to *avoid* producing, and also robbing the free silica of the ferrous oxide which is necessary for a flux. We can use a small amount of carbonaceous fuel—say 2 to 6 per cent coke—and still blow in enough air to maintain the powerfully oxidizing atmosphere that must be kept up if we are to effect the thorough scorification of each and every constituent of the charge that is capable of being scorified; but, as the slow-sinking charge approaches the zone of fusion, it becomes more and more helpless as a source of heat. There has been long-continued and powerful oxidation going on for several feet above the smelting zone, and, although all the constituents of the charge have been heated to so high a temperature that the small proportion of sulphides that still remains undecomposed is almost ready to melt, while the silica requires only a few hundred degrees more heat to fit it to combine with the ferrous oxide that will result from the burning of these remaining sulphides, we shall find that there is not quite enough heat left to complete the process. Most of the oxidizable constituents of the charge have already been oxidized, and, as soon as a substance has been ozidized, it ceases to exist as a source of heat. It is inert and dead, and absorbs, instead of producing, heat.

The furnace is in the same condition that a converter is in when we start to blow white metal up to blister copper without having an excess of heat stored up in its walls. We have no proper fuel left in the charge;

the 20 per cent of sulphur combined with the copper is but a feeble source of heat, and our reliable fuel—the iron content of the matte—has been removed in a previous operation. To prevent chilling, extraneous heat must be procured from some source or another, and, in practice, this is usually effected by adding a quantity of lower-grade matte, the combustion of which furnishes not only the heat required to melt the white metal thoroughly, but also to warm up the converter lining to a point sufficient to carry the copper through the finishing stages of converting, where it cannot itself supply heat enough to keep the charge thoroughly melted. The converting of copper matte to blister copper without extraneous heat would be impossible, and, as has just been shown, this extraneous heat is obtained by storing up caloric in the converter walls during the stage of the process where we are generating an excess of heat, and drawing upon this source during that stage in which insufficient heat is being produced.

In pyrite smelting in the blast furnace, it is impossible to store up excess heat, to be drawn upon when needed later; we lack sufficient undecomposed sulphides in the charge to augment the heat required at the moment of fusion; and we are debarred from the use of sufficient carbonaceous fuel to supply what is wanting. The balance seems to be struck just about at the point of fusion, and the addition or the subtraction of a few thousand units of heat just at this critical moment will make the difference between successfully melting the charge, or freezing up the furnace. It is under such conditions as these that the pre-heating of the blast, even to a very moderate degree, will turn the scale; and it is because they are dealing with (approximately) such conditions that Koch and Bretherton insist upon the heated blast, not as *advantageous*, but as *indispensable*, while those gentlemen who are smelting heavy sulphides regard it as a luxury rather than a necessity.

3. SAVING OF COKE.

The question, "To what extent can fuel be eliminated?" cannot be answered categorically, if it is to convey accurate information. It can be said in a general way that, other things being equal, the percentage of coke to be used in any given case stands in inverse proportion to the oxidizable constituents of the charge. Or, in plain language, when a charge contains its own fuel in the shape of iron sulphides or arsenides, we need not add so much outside fuel to melt it. And it is not only that we are not *obliged* to use as much extraneous fuel; we can not use it, if we expect to get any reasonable degree of concentration.

If an ore is full of sulphides, and we desire to obtain a fairly concentrated matte, the only way that it can be done is to burn the sulphur to sulphur dioxide and send it out of the stack; and burn the ferrous sulphide to ferrous oxide and send it into the slag; these objects cannot be accomplished unless two conditions are present: 1, an oxidizing atmosphere; and 2, an excess of free silica.

To obtain this oxidizing atmosphere, we must reduce the coke until, when blowing a reasonable blast into the furnace, there will be enough oxygen not only to burn the coke, but also to produce a highly oxidizing atmosphere with which to burn the sulphides; for, when an atom of oxygen has to choose between an atom of glowing coke and an atom of glowing sulphide, it prefers to combine with the coke, and the sulphide simply melts as ferrous sulphide and goes into the matte as such, and it is only the *excess* of oxygen that acts upon the sulphides. Consequently, when smelting a heavy sulphide ore, with the determination of obtaining a considerable degree of concentration, the question, "How much can I cut down the coke?" is an entirely inappropriate one. It actually seems to me sometimes that it would be more correct to ask: "How much coke do I dare to use?" For the experimenter will soon find that it is the degree of concentration required that will answer these questions. Assuming that other conditions are favorable, and that he has ample free silica in his ore, he must keep carefully lessening his coke until the required degree of concentration is attained.

It is nervous work when one first tries it; but, if the charge is suitably made up for genuine pyritic work, it will be found that, although the behavior and the interior appearance of the furnace will exhibit marked changes as it shifts its fuel from coke to sulphides, the smelting will proceed satisfactorily, though slowly, and the supply of heat will be ample. I will indicate, however, one danger to be avoided as it has been the cause of trouble in several experiments that have come to my knowledge, namely, as the coke is lessened, the limestone (if such is being used) must also be lessened, or the silica increased—which is virtually the same thing. Reference to this point will appear too elementary to the contributors to this discussion, but it will be useful in many instances. This necessity of lessening the limestone, or increasing the silica, arises, of course, from the fact that, as the proportion of coke is lessened, the sulphides are burned more vigorously, and the resulting ferrous oxide requires silica to slag it. The charge, therefore, is now more basic than it was when coke was being burned as a fuel, and, unless the ferrous oxide is offered the necessary free silica to combine with, troubles will arise, and the degree of concentration will drop again.

An oxidizing atmosphere is the distinguishing characteristic of pyrite smelting, and heat is one of its principal products. Consequently, an ore that contains considerable quantities of suitable oxidizable constituents, such as pyrite, is capable of yielding considerable quantities of heat, and thus requires but little coke for its smelting; and *vice versa.*

Austin speaks very conservatively as to the extent to which carbonaceous fuel may be reduced, and states that, with an abundance of sulphide ore, the amount of coke has been cut to 6 per cent and less. No doubt he is referring solely to cases where there is a tolerably high ratio of concentration (and, consequently, a silicious charge), for much so-called pyritic smelting has been done for years with one-third as much fuel as this.

Beardsley comes direct from the great Mount Lyell smelter, where there is a maximum amount of sulphides, and where the ratio of concentration is moderate, say 7 to 1, and tells us that, when running with a portion of the charge consisting of certain ores, 2½ to 3 per cent coke is required, while, with the same proportion of the charge consisting of certain other ores of an almost identical *chemical* composition, the charge is smelted with from 0.3 to 0.4 per cent coke. This result is, as far as I am aware, the lowest point to which the proportion of coke to charge has been reduced, with anything like this ratio of concentration, and with campaigns of from 60 to 80 days. The result is also attained with an unheated blast.

Beardsley adds: "At Mount Lyell, as the charge column was gradually raised, and the blast increased, the stove heat was taken off, until, finally, stoves were not used at all." This shows positively that, with a high ore-column, a powerful blast, and heavy pyritous ores, the furnace may make long campaigns on a coke charge of less than one-half of one per cent, providing the physical make-up of the ore is suitable.

Let us see what physical condition of the ore is *unsuitable,* as this is an important point, and one that we have but little information on, in connection with pyritic smelting. Every one will agree that the presence of a considerable proportion of fines is quite disadvantageous in any kind of smelting in the blast furnace. It is doubly harmful in pyrite smelting, and is especially emphasized by Nutting, who suffers from having most of his sulphide ores in a finely divided condition. These fine, heavy, fusible sulphides run down between the coarser lumps of ore until they reach the smelting zone, while an infusible silicious skeleton is left behind. Beardsley, however, calls attention to a still different circumstance, and one which appears to have a much greater influence upon the process than we should anticipate, especially considering the enormous furnaces and powerful blast at the Mount Lyell plant. His ore consists of a massive pyrite, to which must be added sufficient silica and earthy bases to form a suitable slag. He describes two different kinds of silicious ores which are employed as a flux. The first kind consists of a silicious aluminous schist, containing from 4 to 8 per cent copper as bornite, and about 5 per cent iron as pyrite. The sulphide minerals in this schist occur in concentrated and isolated patches throughout the gangue, and when this class of ore is used as a silicious flux, the campaigns are short, and the charge requires 2½ to 3 per cent coke. The second variety of silicious flux is also a schist, having approximately the same chemical composition as the preceding ore, except that it contains about 8 per cent iron, and only 1½ per cent copper. But in this schist, the sulphides are disseminated in little grains pretty evenly throughout the gangue. When using this second variety of schist, the campaigns last from 60 to 80 days, and only 0.3 to 0.4 per cent coke is required.

I think that I am fairly interpreting the replies to this question when I say that, in pyrite smelting, the proportion of coke may be reduced to almost nothing in the case of heavy sulphide ore of favorable physical make-up, and where the ratio of concentration is not very high, and that the amount of coke increases as the proportion of sulphide diminishes, and also increases (within narrow limits) as the degree of concentration increases; but that, even with high concentration, and with quite a low proportion of sulphide, a marked saving in coke is effected by running the furnace pyritically.

4. Copper a Collector of the Precious Metals.

The question of the amount of copper required for the collection of the precious metals is of much interest to those who are engaged in the smelting of copper ores by any process, and is doubly important where pyrite smelting is practised. Copper ores frequently contain values in gold and silver, and the ordinary blast furnace smelter often finds it advantageous to buy outside gold and silver ores containing little or no copper, *provided they are not too silicious for his charge*; but this italicized limitation cuts him off from a great part of all the precious metal ores on this continent.

Nor is this all. He is also cut off from another great class of ores, which are not much wanted by either lead or copper smelters, and on which there is, accordingly, a quite profitable treatment charge. I refer to dry ores of only moderate richness in gold and silver, and containing, perhaps, 1 or 2 per cent copper, 3 or 4 per cent lead, a little arsenic and antimony, several per cent of zinc-blende, and enough pyrite to bring the total sulphur contents up to 10 or 12 per cent. Such ores contain too much sulphur to smelt raw either in the copper or the lead furnace, and yet scarcely enough to pay to roast them, and being usually silicious as well, are not regarded with favor by the average metallurgist.

Yet these very ores that I have been describing are among the most advantageous that the pyrite smelter can desire. With the aid of the hot blast, he may expect to completely volatilize the lead, arsenic, and antimony, and to either slag or volatilize the zinc. He can obtain almost any degree of concentration that he desires, and the heat produced by the oxidation of even this very moderate proportion of sulphides will effect quite a saving in coke. There is so little copper in these ores that, although promptly giving up their gold and silver values to the matte, they add very little to its weight. Hence, the refining charges per ton of original ore will be small, and the conditions for close saving of values are so favorable, that the 1.3 per cent copper deducted in accounting for what little of that metal may be present, will generally pay for the limestone that must be added to flux the excess of silica that these ores usually carry.

The smelting of these dry silicious, precious metal ores, which will, of course, be mixed with more massive pyritous material, seems to me to be the especial province of pyrite smelting. A typical instance of this kind may be found in Bretherton's excellent work at the Val Verde smelter, in Arizona.

Eight of the correspondents have replied to this question so simply and specifically that it is possible to make a little table of their figures.

COPPER REQUIRED TO COLLECT GOLD AND SILVER.

Austin........	0.5	
Beardsley.....	1.5	Has never had experience with less.
Carpenter.....	1.0	and less.
Fulton........	0.5	
Koch.........	0.5	
Lloyd.........	0.5	and less.
Mathewson....	0.0	
Nutting.......	0.5	

There is a remarkable uniformity in these replies. An average, or composite, answer to a question of this kind is, of course, almost worthless where the replies vary much; but, in the present case, they are so nearly identical that I will give the average result of the replies, omitting the answers of Messrs. Beardsley and Mathewson: the former, because he says that he has not had occasion to try the minimum limit, the latter, because he contends that no copper at all is necessary, as, indeed, do one or two of the other gentlemen, under certain conditions. The average of the remaining replies is 0.643 per cent—or, say two-thirds of one per cent—as the amount of copper in the furnace charge that is sufficient to make a satisfactory collection of the gold and silver *under favorable conditions.*

Let us see what the various correspondents regard as "favorable conditions." Austin says: "A charge containing sulphide ores, but quite free from copper, will not give a clean slag; but, with as little as 0.5 per cent copper, this object can be attained." He mentions various conditions as favoring the production of a clean slag; such as, absence of zinc, fusibility, reasonable difference in specific gravity of matte and slag.

Beardsley has never had occasion to go below 1.5 per cent copper in the charge, and, down to that point, has always had clean slags. This has been mostly with a concentration of about 7 to 1, and with a ferruginous slag, rather low in silica for pyrite smelting, and containing only about 13 per cent of earthy bases.

Bretherton, who is smelting a silicious charge rather high in earthy bases, tolerably rich in precious metals, and making a concentration of anywhere from 12 to 20 into 1, requires 1 per cent copper, and prefers 3 per cent.

Carpenter bases his reply upon the percentage of copper the resulting matte should contain, rather than upon the proportion of copper that should be in the charge before smelting. He also lays stress upon the *quantity* of matte produced, and intimates that a large fall of matte containing only 2 or 3 per cent copper (original charge) might answer the purpose of collectors as well as a 10 per cent matte, if the latter is produced only in small quantity. He also adds: "If the ores carry both gold and silver, and a fair quantity of matte is made, copper may be entirely avoided."

In connection with this reply, those interested should read Dr. Carpenter's interesting paper on 'Pyritic Smelting in the Black Hills.'[1] He had noticed that an increase in the grade of the matte beyond 10 per cent copper did not increase its virtue as a collector of the precious metals, and that even a 30 or 40 per cent matte was no better than the lower grade indicated. (The advantage of the richer matte might become obvious if the slag were basic or highly ferruginous, owing to the higher specific gravity of matte rich in copper.)

Fulton, in smelting the silicious gold ores of the Black Hills, South Dakota, at the Rapid City smelter, makes a slag, which I take the liberty of averaging as follows: Silica, 48 per cent; ferrous oxide, 18; lime, 28, and alumina, 5 per cent. This is a pretty silicious slag, the oxygen-ratio of acid to base being 2.33 to 1, provided alumina is reckoned on the acid side, as is the Mansfeld custom; indeed, it seems to me an even tougher proposition than Carpenter's average slags at the Deadwood & Delaware smelter, as the latter had about one-third of this lime replaced by magnesia, which gives a better slag than straight lime. So, as iron oxide is low, and lime is the only other base, it must have a high formation temperature; but, once thoroughly melted, should admit of an excellent separation of the matte. The smelting charge proper contains only traces of copper, and it was found that the slags frequently ran $1.50 to $2 per ton in gold. As in the Deadwood smelter, it was noticed that when the smelting conditions were such that a certain amount of metallic iron was formed during the fusion, the slags would be clean, but the formation of this metallic iron could not be satisfactorily

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[1] *Transactions* American Institute Mining Engineers, Vol. XXX, page 764.

controlled. Recourse was had to copper as a collector, and just sufficient (unprofitable) Montana copper ore is now added to bring the slag values down to 25 cents, or less, per ton. This is accomplished by adding 10 lb. copper for each ounce of gold present, which, I think, is about 0.5 per cent copper on the charge. Mr. Fulton finds that even this addition of copper does not collect the small amount of silver present so thoroughly as it does the gold. (See 'Sulphide-Smelting at the National Smelter of the Horseshoe Mining Co., Rapid City, S. D.' By Messrs. Fulton and Knutzen.)[1]

Koch at the Magistral mine, in Durango, Mexico, is smelting an ore that is tolerably free from metals or metalloids that might modify or obscure the behavior of pure iron-copper matte as a collector of the precious metals. His charge averages 0.5 per cent copper, and about 0.5 oz. gold and 0.25 oz. silver per ton. He makes a concentration of 15 into 1, with the production of a 7 to 8 per cent copper matte, and his slags are clean.

Lang holds that the quality, rather than the quantity, of the matte formed is the most important factor, and that a matte comparatively high in copper is essential, if there is a strongly ferruginous slag. He thinks that if an iron matte is to be a satisfactory collector, the accompanying slag must contain a fair proportion of earths. This, I think, in no way disagrees with the opinions of the other correspondents.

Lloyd states that, with a matte under 100 oz. silver and 30 oz. gold per ton, a charge having a copper tenor of 0.5 per cent, or even less, will make a close saving. He cites cases from his own experience where he has done clean work with even less copper. (The instances referred to by Mr. Lloyd would be even more valuable if the average composition of the slag were also indicated. A matte that may answer as an excellent collector where there is a silicious or earthy slag may be

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[1] *Transactions* American Institute Mining Engineers, 1904.

quite unsatisfactory where the slag is basic or highly ferruginous.)

Mathewson gives a concise and emphatic answer to this question. He says: "Copper is not necessary for the collection of the precious metals. They will collect readily in an iron matte."

Nutting, with whom zinc-blende is ever present, says that, when this mineral is absent, a fair collection can be made without any copper, but that the beneficial effects of even 0.5 per cent will be recognized. With a considerable amount of blende in the charge, he would desire 1.5 to 2 per cent copper, to induce a better separation of matte and slag.

Ingalls found that in smelting an ore containing 50 oz. silver and 1 oz. gold per ton, and 1 per cent copper, he made satisfactory recoveries. He believes, however, that if much blende were present, it would take more copper, so that the separation of matte and slag might be easier.

There seems to be a pretty complete unanimity of opinion that a very small percentage of copper in the charge will suffice as a collector for gold and silver, provided that the slag is favorable for the mechanical separation of the matte.

One per cent of the weight of the charge in copper, and less rather than more, would appear to satisfy the requirements of the gentlemen represented in this discussion, provided the slag is not too heavy in iron, or does not contain an excessive amount of zinc oxide.

Several of the contributors to the recent discussion state that an iron matte, free of copper, forms a satisfactory collector of the precious metals under suitable conditions. Others, evidently, have not found this to be the case; but positive evidence is better than negative, and in the light of such unimpeachable statements as we now have on the subject, we may, I think, accept as a fact that, under favorable conditions, a matte free from copper will collect the precious metals satisfactorily.

The question at once arises: What are these favorable conditions? As has been already stated, a rather silicious, liquid slag of low specific gravity is highly advantageous, for, without this, the light iron matte will find it difficult to settle properly.

There are, however, other factors that profoundly influence the collection of the precious metals in an iron matte; and, in order that this most interesting point may not be left incomplete, I shall take the liberty of discussing briefly some of the other conditions that bear upon it.

In the first place, this question, though most important to the pyrite smelter, does not belong any more to him than it does to the ordinary smelter of roasted ore. For many generations past, therefore, whenever the copper metallurgist has been smelting sulphide ores containing gold or silver, whether in blast furnaces or reverberatories, and whether raw or roasted, he has been accumulating facts that bear directly upon this question. It would seem, therefore, that by this time we ought to possess enough information on the subject to enable us to say positively that, when making a slag of such and such a composition, and with a charge of such richness in gold and silver, our matte must contain so and so much copper in order to be an efficient collector of the precious metals present. As a matter of fact, we can do nothing of the kind; and, in smelting two different ores, having the same values in precious metals and yielding a slag of identical composition, we may easily find that, in the one case, a matte entirely free from copper will prove an efficient collector of values, while in the other the matte must contain several per cent of copper in order to clean the slags effectually.

With the object of inviting discussion, I will suggest the following factors as having a bearing upon the collection of the gold and silver in an iron matte.

1. The physical make-up of the ore.
2. The influence of certain substances that often accompany the precious metals in very minute quantities.
3. The formation during the operation of smelting, of certain substances that may act as collectors.

1. *The physical make-up of the ore* is probably the least important of the three factors just enumerated, but must, I think, be recognized in certain ores. It is impossible to discuss the point properly except at length, but I have several times convinced myself that, in certain cases where free gold or tellurides exist in minute specks throughout massive quartz, and the amount of matte made is small and is produced from coarse iron sulphides that are not disseminated throughout the same ore, the slag may run high, owing to want of juxtaposition between the collector and the particles of gold.

2. *The influence of certain substances that often accompany the precious metals in very minute quantities.* The attention of many matte smelters was first directed to this point by Richard Pearce's experiments and deductions in his paper entitled 'The Association of Gold with Other Metals in the West.'[1] I know that this paper cleared up for me many obscure and apparently conflicting results encountered in the matte smelting of certain gold and silver ores, at times when my precarious supply of copper ores ran short. Mr. Pearce shows, by actual experiment, that: "Pure gold melted with pure iron pyrite is not attacked in any way by the fused sulphide of iron. A matte is obtained in which the gold exists in a pure state, but disseminated through the mass in very fine globules."

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[1] *Transactions* American Institute Mining Engineers, Vol. XVIII, page 447.

Mr. E. G. Spilsbury[1] described the smelting of a mixture of raw and roasted pure iron-pyrite concentrates, carrying about two ounces in gold per ton. The properly fluxed ore was smelted by Professor F. W. Clark, of the Massachusetts Institute of Technology, in a small reverberatory furnace. The well-melted slag contained from 1 to 1.6 oz. gold per ton, and the resulting iron matte assayed 4 to 8 oz. per ton, the gold values being very irregularly distributed in the matte, though the slag assays were comparatively uniform.

These experiments, in connection with results obtained in my own practice, lead me to believe that *pure* iron sulphide is a very poor collector of *pure* gold (or of gold containing very little silver), and this belief is held, I think, by most metallurgists whose attention has been called to the matter. It appears that, in such cases, the gold is simply held in mechanical suspension in the matte, and not in *solution,* as Pearce expresses it. The same observer found further, that the presence of minute proportions of bismuth (which occurs much more frequently in connection with gold ores than any one realizes, except, perhaps, the electrolytic refiner), and perhaps, also, tellurium, caused a complete solution of the gold in the iron matte; and it is quite probable that arsenide and antimonide compounds may exert a similar influence.

It seems to me probable, therefore, that, in cases where a good collection of the precious metals has been made by an iron matte free from copper, there have been present small quantities of bismuth, tellurium, arsenic, antimony or other substances that have aided this collection, and that we need a good deal of accurate and delicate chemical investigation on this subject before we shall be able to predict, in any given case,

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[1] *Transactions* American Institute Mining Engineers, Vol. XV, page 767.

whether we are going to be able to make clean slags without any copper in the matte.[1]

3. *The formation during the smelting process of certain substances that act as collectors.* That metallic iron, under certain conditions, will act as a collector of gold, though not of silver, has long been known to metallurgists. More than 50 years ago a blast furnace was erected for the purpose of smelting into metallic iron the rich gold-bearing black sands of the Siberian placer mines. It is said to have yielded clean slags, and a pig-iron rich in gold. It was eventually given up, owing to the expense of separating the gold from the pig-iron, and the difficulty of controlling the reduction of the iron oxides in the furnace.

The originator of the process hoped to be able to slag a large proportion of the oxidized iron as ferrous oxide, and to reduce just sufficient of it to obtain a moderate amount of metallic iron rich in gold. This he was unable to do with any certainty, as, if he reduced enough of the oxides to metallic iron to obtain clean slags, he found that, before he could stop it, nearly all of the oxidized iron in the charge would be reduced to the metallic state, thus producing a large amount of low-grade metal and robbing his slag of its needed flux.

This experience is useful to us as showing that metallic iron has, at least, some power in collecting gold; but, apart from this, it is of little interest to the matte smelter, as the conditions are completely changed by the presence of sulphides in the charge.

For the true pyrite smelter, it has an even more remote bearing; for not only may the presence of sulphides in the furnace obscure the reactions, but the metallic iron that he, under certain conditions, produces is a totally different substance from the metallic iron

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[1] Since the completion of this article, I have received a copy of a paper by Mr. Myrick N. Bolles, B. S., to appear in the *Transactions* A. I. M. E. for 1904, entitled, 'The Concentration of Gold and Silver in Iron Bottoms'; this contains much valuable original work on the matter discussed under this head.—E. D. P.

produced in smelting the Siberian black sand with coke. The cause of this difference is obvious. In the coke furnace, as run on these Siberian black sands, the product was simply cast-iron; the reduction was effected by the carbon, and the resulting iron contained carbon, as does ordinary pig-iron. In the pyritic furnace, the atmosphere averages distinctly oxidizing; else the sulphur would not be burned, nor the iron be changed into ferrous oxide, nor would it be 'pyrite' smelting, as I am employing this term.

Consequently, the metallic iron, which is occasionally produced in considerable quantities in the pyritic furnace, would appear to be produced under distinctly oxidizing conditions and not to be connected with the presence of carbon. I know positively that metallic iron can be made by blowing a powerful blast into molten sulphide ores, when making a very silicious slag, because I once inadvertently filled up my hearth with an iron sow, when trying to smelt raw sulphide fines *without carbonaceous fuel.* The fines ran down through my coarser silicious ore, the result being that the charge soon froze solid, but not until a sow of metallic iron, weighing several hundred pounds, had been formed.

Dr. Carpenter gives much valuable information on this matter in his paper on 'Pyritic Smelting in the Black Hills.'[1] Messrs. Fulton and Knutzen supplement this by a more recent paper.[2]

I quote from the latter: "The matte formed rarely contains more than 30 per cent sulphur, while the iron monosulphide contains 36.36 per cent, so that the matte is evidently a subsulphide. It also contains metallic iron, which can be readily abstracted by the magnet. We agree with Dr. Carpenter that it is this metallic iron in the matte which collects the gold, but, unfortunately, it is rarely present in the matte in sufficient quantity to

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[1] *Transactions* American Institute Mining Engineers, Vol. XXX.

[2] *Ibid.*, 'Sulphide Smelting at the National Smelter of the Horseshoe Mining Company,' 1904.

give clean slags. Paradoxical as it may seem, the quantity of metallic iron formed in the furnace is due to a large extent to the amount of oxidation which takes place in the furnace. . . . In our opinion, the production of sows is practically inseparable from sulphide-smelting when high concentration is done. The sows are due to the strong oxidizing effect of the furnace, as shown from the following data: A desulphurization of 80 per cent; the production of copper sulphate, found in layers in the accretions of the downtake; no evidence of carbon monoxide in the furnace gases; the volatilization of all the lead fed into the furnace; and the facts that, while no iron goes into the furnace as oxide, the slag contains from 18 to 20 per cent of iron oxide in the form of silicate. These data make it difficult to imagine that the reducing conditions in the furnace could exist sufficiently strong to produce metallic iron.

We believe that the sows are produced by oxidation in a similar way that metallic copper is produced during bessemerizing; taking as the first stage the melting of the pyrite, FeS_2, and the loss of the one atom of sulphur, thus forming the monosulphide, FeS; the second stage, the gradual oxidation of the sulphur in the monosulphide, producing a subsulphide; the third stage, the production of some ferrous oxide, part entering the slag and part reacting with the subsulphide present, producing sulphur dioxide gas and metallic iron, according to the following chemical equation:

$$FeS + 2FeO = 3Fe + SO_2.$$

Experience has shown that a larger quantity and higher pressure of blast result in an increased production of metallic-iron sow, and, from its analysis, it is seen that it contains practically no carbon."

It seems possible, therefore, that the production of a certain proportion of metallic iron in pyritic smelting may carry into a pure iron matte gold value that otherwise would not be saved.

5. THE PART PLAYED BY LIME.

None of the replies to the question covering the percentage of lime necessary to clean the slags, indicates that any particular specific virtue is attached to lime *per se*, as a cleanser of the slag from valuable metals. I think I am fairly interpreting the unanimous spirit of the contributors by stating that what they demand is a slag from which the matte will separate satisfactorily; that such a slag must be tolerably silicious, yet fusible; and, moreover, that it is advantageous to have it contain a fair proportion of earthy bases, so that it may not have too high a specific gravity. In the great majority of cases there is no substance that will fill the bill so completely, and produce the results just enumerated so cheaply and satisfactorily, as lime.

As a flux for excess silica, up to a certain limit, 1 lb. of lime will go as far as 2.58 lb. of ferrous oxide. This comes from the fact that not only is the atomic weight of lime considerably smaller (and, consequently, its oxygen contents larger) than that of ferrous oxide, but that, in the oxidizing atmosphere of the pyrite furnace (as clearly pointed out by Carpenter and Lang), the silica prefers to make only a bisilicate with lime, while it forms a *singulo*-silicate with ferrous oxide; and as the slag that we prefer to make in pyrite smelting will have an oxygen ratio of acid to base of somewhere about 1½ to 1, and is thus a mixture of the bisilicate of lime with the singulo-silicate of iron, it follows that, until we have reached our maximum desirable limit of lime in the slag, each pound of lime (forming a bisilicate) will go as far as 2.58 lb. of ferrous oxide (forming a singulo-silicate).

The actual percentage of lime necessary to effect a good mechanical separation of the matte varies so greatly under differing conditions that it is impossible to lay down any fixed rules in regard to it. A glance at the large table of questions and answers will show that most of the replies are based upon the local conditions under which the contributor is operating, and are most valuable when considered in connection with these conditions.

A few of the correspondents have given replies based on more general conditions. Austin says: "Slags containing as little as 5 per cent of lime have proved clean."

Carpenter avoids lime as far as possible in genuine pyrite smelting, not that he does not value the type of slag that it makes, but because it increases the *quantity* of slag, and takes the place of ferrous oxide in combining with silica. He says: "I think, all things considered, and, especially, if the ores carry alumina, lime had better be avoided, particularly in the first smelting."

Lang values greatly the qualities of a slag containing a suitable proportion of lime, but suggests that there are often occasions where it is commercially more advantageous to get along without it, and says that reasonably clean slags can be made where ferrous oxide is almost the sole base.

Lloyd expresses very much the same views as the preceding writer, and emphasizes the important point that, as a slag consisting mainly of ferrous silicate is heavy, and difficult for the matte to separate from, it becomes all the more necessary, in such cases, to provide peculiarly favorable conditions for the separation of slag and matte. Such, for instance, are large forehearths, maintained at a high temperature.

I think, therefore, that the pyrite smelter may assume that lime has no specific virtue in cleansing the slags, but that—apart from its great importance as a basic flux where too much silica is present—its main value is to make the slag lighter in weight, so that the matte globules can separate more quickly and more perfectly from it.

6. BEHAVIOR OF ZINC.

The answers to the question, "What percentage of zinc in the charge can be treated profitably?" are, on the whole, the least perfect and the least unanimous of any of the sets of replies. Of the ten contributors who appear in the large table, six do not answer the question at all, most of them not having had occasion to practise pyrite smelting on ores high in zinc. Our evidence, therefore, is limited to the replies of Messrs. Austin, Bretherton, Lloyd and Nutting, together with some interesting remarks on the same subject contained in the letters of Messrs. Godshall and Ingalls.

Even with this small amount of testimony, there is some conflict of opinion as to the behavior of zinc in the furnace. This disagreement, however, is apparent rather than real, and arises simply from the fact that pyrite smelting covers such a multiplicity of differing conditions, that almost any question may elicit answers totally at variance with each other, and yet, that are not in the least conflicting. In ordinary blast furnace smelting with coke, it may, I suppose, be accepted as a fact that when much zinc-blende is present in the charge, a portion of it is likely to enter the slag mechanically and be carried away still as a sulphide; this tends to make the slag thick and heavy, and to produce losses of values. In the powerfully oxidizing atmosphere of the pyrite furnace, however, it does not seem to me possible that the zinc sulphide can behave in this manner, except in those instances where there is merely a quick melting-down of the charge rather than a genuine pyrite smelting. Blende commences to burn at a dull red heat, and its oxidation, in the presence of an ample air supply, is so free and vigorous, that it is scarcely conceivable that any undecomposed zinc sulphide should be left to be carried into the slag. Consequently, I think that, in that variety of pyrite smelting where a high ratio of concentration is obtained by pushing the process of oxidation to nearly its extreme limit, we can eliminate this point entirely, and, for the moment, confine our attention to the behavior and influence of the zinc oxide that will result from the thorough oxidation of the blende.

This substance may act in three different ways:

1. It may be volatilized, and thus be removed from the smelting zone altogether; forming accretions on the furnace walls, condensing in the dust-chambers, or escaping into the atmosphere.

2. It may be dissolved in the slag as zinc oxide, without combining with silica.

3. It may enter the slag as a silicate, or, perhaps, as an aluminate or ferrate.

The first of these three conditions usually prevails where the percentage of blende in the charge is not excessive (perhaps 6 or 7 per cent), and where the ratio of concentration is high, owing to powerful and prolonged oxidation. This always means a highly silicious or aluminous slag, and a comparatively slow sinking of the charge. The zinc is exposed, before it reaches the smelting zone, to such a high and prolonged oxidizing temperature, that it is completely decomposed and volatilized.

The second and third alternatives may be considered together. These include the cases in which the zinc oxide, in part, enters the slag, but whether in simple solution, or as a zinc silicate, is not, so far as I am aware, definitely settled. At any rate, this is not the place to discuss such a point, and, while the distinction is one of a good deal of importance to the practical metallurgist, we must be content to hold the matter in abeyance for the present, and confine ourselves to studying the effects which zinc oxide produces when it enters the slag, whether it be there in solution, or whether in combination with the silica or other acid constituents.

Austin says that zinc, up to 10 per cent of the charge, is treated successfully, but calls attention to the fact that, in the presence of much alumina, it causes trouble.

Bretherton, who has unusual experience on this particular point, and is, at present, smelting zinky ores with a high ratio of concentration, a silicious slag, a minimum of coke, and a product that has to be free from zinc to escape penalization by the refiners, says: "I do not think it advisable to make a slag containing more than 12 per cent zinc; that is, 15 per cent zinc oxide, and the ferrous oxide must be reduced in proportion. So that for practical running, I would put the maximum amount of zinc oxide at 11 per cent, and the minimum amount of lime at 10 per cent, assuming that there is no magnesia nor alkaline bases in the slag."

Lloyd, referring to reverberatory, rather than pyrite, smelting, says: "My personal experience with zinky slags has been more through reverberatory practice than blast-furnace practice, and I have noted that zinky slag in a reverberatory, that is, a slag exceeding 10 per cent ZnO, while it never becomes as liquid as a slag containing an equivalent amount of silica, but with FeO as practically its only base, still smelts at quite as low a temperature, and, as far as copper is concerned, is quite as clean. I should say that ZnO has not so much the effect of lowering the fusing point, as of thickening the slag."

Nutting says: "This element, in the form of sulphide, is an unmitigated nuisance. Its bad effects are numerous, and will be noted in the tonnage, recovery, and product; difficulties start with as little as three per cent in the charge, and increase quite in proportion until twelve per cent is reached, when the bad effects become serious."

In the light of Mr. Nutting's reply, we may consider the unfortunate behavior of zinc blende in that class of cases where considerable heavy sulphides are present where the slag is not very silicious, and where the ratio of concentration is not high. Under such circumstances we do not have the prolonged and tremendous oxidation at a high temperature that I have referred to in a former paragraph. A certain proportion of the blende escapes oxidation, enters the matte and slag, and produces the disagreeable results so familiar to all of us. I do not at all wish to imply that it would be more advantageous for Mr. Nutting to form a more silicious slag, increase his oxidation and his ratio of concentration, and burn all his blende to oxide before it reaches the smelting zone. I have very little doubt, that he is pursuing the method best calculated for paying dividends, and refer merely to the chemical, and not the commercial, features of his difficult ores.

Both Godshall and Ingalls speak strongly on this very point. They claim that if ores contain so much zinc-blende that, in smelting them raw, any considerable portion of the zinc escapes oxidation, and thus enters the molten products as a sulphide, it may be better to transform the ZnS into ZnO by a preliminary roasting. I quite agree with them on this point, especially where the rate of concentration is to be low and the oxidation moderate; but in cases where there is high concentration, silicious slag, and powerful oxidation, I think that pyrite smelting can be done more cheaply and with less loss of values than roasting and smelting with coke. In a word, I believe that in this class of cases, we can roast the blende more cheaply in the blast furnace than we can outside of it, besides saving enough time and coke to show a considerable advantage for the pyritic method.

Thus far, we have no real conflict of opinion, as any differing expression in the views of correspondents has arisen from the fact that 'pyrite' smelting covers such a wide field, that it embraces conditions under which it can advantageously treat ores high in zinc-blende, as well as conditions under which it would make a poor showing on ores containing a similar proportion of that mineral.

There is one single point, however, on which Mr. Bretherton differs radically both from Mr. Godshall and Mr. Ingalls. Bretherton places the maximum amount of ZnO that the slag may contain at 15 per cent, and says that the FeO should be reduced proportionately, while there should be a minimum amount of 10 per cent of CaO or equivalent earthy bases. He does not mention any reduction in silica; so we may assume that he finds it practicable to run with his normal, rather silicious slag, even when a pretty high percentage of ZnO is present. It must also be explained that Bretherton's slags at Val Verde usually contain 8 to 12 per cent alumina, which he reckons on the acid side.

Godshall says: "High zinc slags invariably require a heavy percentage of iron, and render, therefore, the usually profitable, and always desirable, high silica slags an impossibility."

Ingalls says (quoting from the middle of a sentence): "And the slag must be high in iron . . . in order to carry off zinc."

Hofman, in referring[1] to the slagging of ZnO in the lead blast furnace, says: "Care must be taken that it is not reduced to metal; hence the smelting has to be done quickly, and at a low temperature. This requires a slag not high in silica, and with a preponderance of iron."

My own experience leads me, in certain cases, to concur in the views of the three last-named authors. But does not this divergence of opinion between Bretherton on the one side, and Godshall, Hofman, and Ingalls on the other, arise from the fact that the two opposing views are based on opposing conditions?

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[1] "Metallurgy of Lead," page 292.

Only a few months ago, I stood at the tunnel-head of Bretherton's Val Verde furnace, and saw a mixture heavy in zinc-blende and earths, and low in iron, shoveled into the charging-door. The slag from this charge was liquid, and, physically, of excellent quality to the eye, and the daily analyses showed that it contained approximately: Silica, 37 per cent; alumina, 10 per cent; ferrous oxide, 25 per cent; lime, 16 per cent; magnesia, 2 per cent; zinc oxide, 7 per cent; copper, 0.3 per cent, with silver 0.5 oz. and gold, 0.02 per ton. A good deal of the zinc in the charge had been volatilized, and the matte contained so little of that metal, that after a single concentration smelting of the matte, the enriched product was free from zinc. Figuring alumina as an acid, the above slag is almost exactly a bisilicate. It seems almost certain, therefore, that in this case, in the presence of a powerful oxidizing atmosphere, a high temperature, and a prolonged contact with free silica, such of the ZnO as escapes volatilization combines with the silica to form a silicate of zinc, and thus acts distinctly as a base, permitting a corresponding lessening of ferrous oxide in the charge, up to a certain limit.

The practical lesson that we may, perhaps, deduce from these observations is, that the proportion of blende that we can successfully handle in a charge, without producing accretions, foul slags, and other well known difficulties, probably bears a definite relation to the vigor of the oxidation in our furnace shaft, always assuming that there must be sufficient free silica to combine with the zinc oxide that escapes volatilization; and that this free silica must be in excess of the silica that it is necessary to have to form a bisilicate with the lime present, and a singulo-silicate with the ferrous oxide that will result from the iron-bearing sulphides or arsenides that we intend to decompose. This proposition is merely tentative, and for the purpose of inviting discussion.

7. DEGREE OF DESULPHURIZATION ATTAINABLE.

This question must, of course, be interpreted as meaning, "What is the degree of desulphurization that it is commercially advantageous to attain?" The average of the figures given by eight out of the ten tabulated answers is 76 per cent as the amount of sulphur driven off in pyrite smelting; in several instances this is modified by the statement that, with a silicious slag and slow smelting, a much higher degree of desulphurization is attainable.

Carpenter says that, at Deadwood, he has often oxidized all the sulphur in his charge, making no matte at all (his gold collecting in iron sows), while Lang reports burning off 96 per cent of the sulphur from a charge containing 11 per cent sulphur.

Beardsley touches the keynote of the question. His reply[1] indicates that almost any degree of desulphurization may be reached, and that what particular degree of desulphurization shall be attained in any given case is a commercial proposition, which only the metallurgist in charge can decide.

Lang points out how greatly the silica content of the slag influences the expulsion of sulphur, and states also, that even with the same percentage of silica in the charge, a mixture containing earthy bases, and especially lime, is favorable for the removal of the sulphur.

A few years ago, Lang made a run on the ores of the Blue Dick mine, near Prescott, Arizona, and obtained such remarkable results in the removal of sulphur and arsenic by an oxidizing smelting in the blast furnace, that it will be instructive to refer to it in this Review. I take the facts from his letter published in the *Mining and Scientific Press* of March 29, 1902.

The ore is a mixture of quartz and mispickel, containing a little pyrite, chalcopyrite, tetrahedrite, galena,

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[1] See page 81.

barite, and spathic iron. The values are in gold and silver. As it was received at the furnace, its approximate composition was: Silica, 45 per cent; iron, 17; arsenic, 17; sulphur, 17; and copper, 0.5 per cent. It was necessary to add about 50 per cent limestone to form the required slag. The circular trial-furnace was 36 in. in diameter at the tuyeres, and the cast-iron water jackets were only 30 in. high, the brick shaft extending to the charge door, which was 11 ft. above the tuyeres. The blast was cold, and the pressure only 9 oz. per sq. in. The resulting slag contained: Silica, 40 to 45 per cent; ferrous oxide, 24 to 27; and lime, 20 to 24 per cent. Fifty tons of charge was smelted per 24 hours, being over 7 tons per sq. ft. of hearth area; a most extraordinary record for a small furnace run with cold blast, light pressure, and an acid slag; and due, in great part, to the unusual proportion of volatile constituents in the stock. The rate of concentration was still more remarkable when one recollects that the ore contained 34 per cent sulphur and arsenic, being 27 tons of *ore* into 1 ton of matte; and this matte was free from arsenic, though not sufficient in quantity to entirely cleanse the slags from silver.

Mr. Lang himself was evidently surprised at this unique result. He says: "As one-half of the ore consists of combustible matters (the iron sulpho-arsenides and sulphides), it appears that the decomposition was very extensive. Nearly 90 per cent of the iron was oxidized and slagged off. Fifteen-sixteenths of the sulphur went up the chimney or into the slag; while all the arsenic was volatilized in some form or other. Vast quantities of deep yellow or red sulphide of arsenic, presumably orpiment, passed out of the smokestack, succeeded by thick masses of pearl gray fumes containing arsenious oxide, etc. A good deal of metallic arsenic also is sublimed, but this speedily becomes oxidized, and permeates the atmosphere as gray smoke. Not a single particle of speiss or any other indication of arsenic appeared at the bottom of the furnace. The matte presents no peculiarities except its brittleness, arising, I presume, from the absence of metallic iron, due to the highly oxidizing action of the blast. It carries about 10 per cent copper, which is not enough for a clean saving of the silver. Measures are being taken to procure a quantity of copper-bearing ores for admixture, so as to bring the copper contents of the matte up to 25 or 30 per cent, which will produce a cleaner separation of the silver."

Lang does not ascribe these results entirely to the oxidizing effects of the smelting, but believes that there happened to be a peculiarly favorable ratio between the proportions of sulphur and arsenic in the ore, which induced the extensive sublimation of these volatile substances, leaving the iron a prey to the oxygen of the blast. Such reactions as these furnish food for reflection and further experimentation.

Lloyd says: "On Butte ores, with cold blast, 75 to 80 per cent desulphurization is obtainable."

Mathewson says: "The degree of desulphurization attainable on pyrite is 80 per cent with the Butte ores; we readily get 75 per cent."

The smelting charge usually contains much of its sulphur in the shape of pyrite (FeS_2); and, as pyrite loses one of its two atoms of sulphur long before the period of fusion begins, it is evident that we shall remove 50 per cent of this sulphur even before we have to begin to consider what else we must do to effect our desired degree of concentration. Assuming that the charge has sunk to such a depth in the furnace-shaft that this feeble second atom of sulphur belonging to the pyrite has been sublimated, we have then left only the stable sulphides, such as FeS, Fe_2S_3, Cu_2S, PbS, ZnS, Ag_2S, NiS, etc. These stable sulphides are not further decomposed by heat alone, and will surely melt down into matte unless they can be brought into contact with oxygen, and it is this point that mainly distinguishes pyrite smelting. Unless so *little* coke, or so *much* blast, is used that there is always an excess of oxygen beyond the amount required to unite with the carbon of the fuel, there will be no oxidizing effect, and, consequently, no pyrite smelting. Even if there be an excess of air blown in, there will be but unsatisfactory oxidation of these stable sulphides, unless free silica be also furnished to combine with the ferrous oxide set free by the decomposition of the pyrite.

Consequently, in studying pyrite smelting, we invariably get back to the two foundation stones of the fabric —abundant oxygen, and free silica.

Since writing the foregoing, I have received a letter from Mr. Bretherton, of the Val Verde smelter, which contains the description of a little metallurgical incident so instructive, and bearing so exactly upon the present question, that I quote it herewith: "I want to mention a little experience that we had here just before the close of our last run. I had put on a charge figured at 47 per cent silica, supposing that the ore I was using, as unloaded, would assay 60 per cent silica; and expected to make a 20 per cent copper matte. I had instructed the feeders to carry the furnace very low. The furnace ran very slowly; the matte, instead of assaying 20 per cent copper, which would have made a concentration of nearly ten to one, assayed 33 per cent for two shifts. When I came to get the true analysis of the ore, I found that my silicious ore contained 74 per cent silica, which would have given me nearly 55 per cent silica in the slag, if it had not been that the slow running and high concentration forced into the slag the iron which would otherwise have gone into the matte; but, by running a very low charge, a regular scorifying action took place. As it was, my slag contained 48 per cent of acid; just enough below the bisilicate line to flow nicely." Truly, in the words of Dr. Carpenter, "The pyritic furnace chooses its own slag."

The heavy tonnage and powerful desulphurization obtained at the Boundary smelters in British Columbia, at Butte, at Mount Lyell in Tasmania, and at the Tennessee Copper Company's smelter near Ducktown, show that it is a powerful blast (whether hot or cold) that is needed, and that, in addition, the slag must be somewhat silicious if reasonable desulphurization is desired. The cases in which a heated blast seems almost *indispensable* are those in which the sulphides (heat-producers) are comparatively scarce, the earthy bases abundant, the slag silicious, and the ratio of concentration high. In such cases, the metallurgist cannot use sufficient coke to supply the heat required to complete the fusion, without interfering with his oxidizing atmosphere, and thus spoiling his concentration; and it is for the purpose of furnishing these few units of heat that are lacking that the heated blast is essential.

It seems to me, therefore, that, while we have no sufficient evidence to prove that a heated blast has any more *direct* effect than a cold blast in producing the oxidizing influence necessary to a high degree of concentration, it has an *indirect* effect of such importance that, with certain ores, it renders a degree of concentration possible that, with a cold blast, would be impossible. This matter has also been discussed under the question relating to the use of hot blast. In summing up, it may be said that, while almost any degree of desulphurization is obtainable in the pyrite furnace with suitable ores and ample heated blast, the average result thus far in practice is the removal of about 75 per cent of the sulphur, and that local conditions must, in each case, determine how much beyond this it is profitable to go.

8, 9, and 10. RELATIVE CAPACITY AND ECONOMY.

This phase of the subject has already been partially considered in article No. 1, which treats of the types of ores suited to the process.

The replies to the question involving the economy of pyrite smelting are all highly favorable. This is quite natural, as these replies, without exception, come from men who are pyrite smelters, and who would not be practicing this method if they did not find it advantageous.

This closing article, therefore, is primarily addressed to those members of the profession who are not practically familiar with pyrite smelting.

There is no metallurgist in the world so situated that he would not gladly save fuel in smelting. The great majority of smelters would thankfully avoid a preliminary roasting of their ores, with its expensive plant, and its attendant cost, delay and loss. The ability to produce a slag containing 5 to 12 per cent more silica than is practicable by the ordinary method would be a great boon to nine smelters out of ten.

When, however, a method of gaining these several important advantages simultaneously is offered to a metallurgist, and he is assured that he can enjoy them all without making any important alterations in his works, and by merely shutting down his roasting plant and feeding less coke into his blast furnace, he feels, very naturally, a strong suspicion that, to obtain such manifold and striking benefits, without attendant drawbacks, is too much to hope for. Sudden and radical alterations of either principles or practice are disliked by the experienced smelter.

I know, from my private correspondence, that many able men are startled by the results obtained, for instance, by Beardsley and Sticht in smelting the pyritic ores of Mt. Lyell with 0.5 per cent of coke and a cold blast, and making concentration of seven into one; or by Bretherton's running on a silicious charge of raw zinc, lead, antimony, and arsenic-bearing ores, and concentrating fifteen into one with less than 5 per cent of bad coke, and an almost complete elimination of the impurities just mentioned.

It must be borne in mind that such results as these are the fruit of years of experience, experiment and expense, and cannot be immediately duplicated in a new enterprise, and especially not by metallurgists who lack a long apprenticeship before the pyrite furnace.

There is, however, a wide and fruitful territory lying between the two extremes of blast-furnace practice—that is to say, between the common roasting and reduction smelting of copper ores on the one hand, and the most advanced type of pyrite smelting on the other; and it is quite practicable for the conservative metallurgist to feel his way cautiously along the newer line of practice without materially altering his plant, and without losing touch with the old method which he thoroughly understands.

No metallurgist would feel that he was taking any serious risks in making the three following slight modifications in his blast furnace practice:

1. Using some rather poorly roasted ore.

2. Lowering the ore column in the furnace.

3. Increasing the silica contents of the charge by a few per cent.

Under proper conditions, he would then find that he was producing a matte of about the same grade as by his ordinary method, and that, consequently, he was burning off more of the sulphur in the blast furnace and oxidizing more of the iron than usual; and that his furnace was smelting rather more slowly than before, in consequence of the more silicious slag.

This is the beginning of pyrite smelting, and I obtained my first clear ideas of it many years ago from Bartlett's work at his small smelter in Portland, Maine.

Very little financial advantage is obtained by this method of running a furnace. There is a saving in roasting, as poorly roasted ore can be used in the blast furnace; there is, also, usually a decided advantage in being able to employ a more silicious charge. There is, generally, no saving in coke, owing to the more refractory slag and the low ore column, which offset the heat derived from the decomposition of the sulphides in the charge.

As the furnace men become familiar with the new method of working, further changes can be made. The ore column is gradually restored to its full height, and the necessary strong oxidizing effect is maintained by increasing the blast. The coke is gradually diminished and the proportion of raw sulphide ores increased, the ratio of concentration being regulated mainly by the silica contents of the charge.

It cannot be too often repeated that there will be little oxidizing effect and little concentration until the silica of the charge is provided as follows:

a. Sufficient silica to form a bisilicate with the lime and magnesia.

b. Sufficient silica to form a singulo-silicate with any oxidized iron or manganese that the charge may contain.

c. Sufficient silica to combine as a singulo-silicate with the ferrous oxide that will be produced by the oxidation of sufficient of the pyrite present to leave only enough sulphides to produce a matte of the desired grade.

The total silica, therefore, that must be present in the charge will be a+b+c, a+b representing the silica that is already neutralized by the oxidized bases present in the original charge, while c represents the excess of free silica that is unprovided for, and that lies waiting unfused, and delays the smelting operation until sufficient pyrite is decomposed by the blast to furnish the ferrous oxide that is required to satisfy this silica to the degree of singulo-silicate.

It is this delaying and holding back of the operation by the excess of free silica that makes pyrite smelting, other conditions being equal, a decidedly slower process than the simple fusion of an already oxidized charge. If there were no free silica present, the sulphides would simply melt down with great ease and rapidity into a low-grade matte, as so frequently occurred in the early days of raw smelting of heavy sulphide ores with too basic a slag. It follows, consequently, that this excess of free silica is an essential portion of the pyrite charge, and the delay that it caused by holding up the process until sufficient pyrite had been oxidized to furnish it with FeO has been obviated by heightening the ore column and greatly increasing the pressure and volume of the blast. The decomposition of the pyrite that, with the light blast, progressed slowly and reduced the capacity of the furnace much below the normal duty of a furnace of similar hearth area running on roasted ore and coke, now, with a heavy blast, proceeds with rapidity.

The economy of a smelting process stands, to some considerable extent, in relation to the capacity of the furnace employing that process. Owing to Beardsley's valuable communications to the *Journal*,[1] we have some quite exact information as to the capacity of furnaces at the Mt. Lyell smelter, which employs genuine pyrite smelting, and makes a concentration on heavy pyrite ore of about seven to one on the first fusion.

A furnace 210 in. by 42 in. with a cold-blast pressure of 20 to 30 oz. per square inch, averaged 271.43 tons of charge per 24 hours, which is 4.42 tons per square foot

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[1] See pp. 81 and 219 of this work.

of hearth area. The same furnace, with a blast pressure of 40 to 45 oz., averaged 530.76 tons of charge per 24 hours, which is 8.65 tons per square foot of hearth area. Thus the capacity of the furnace was nearly doubled by an increase in blast pressure of some 20 oz. per square inch, and its duty per square foot of hearth area was brought up close to the highest average record of any of the large blast furnaces running on roasted ore.

The interesting results obtained by W. H. Freeland on the pyrrhotite ores of Ducktown, Tennessee, demonstrate with equal conclusiveness that, in using this method on heavy sulphide ores, a cold blast gives entirely satisfactory results, providing there is enough of it, and that the capacity of the furnace depends largely upon the volume and pressure of this blast. The pyrite furnace demands a much larger blowing plant than does a furnace of equal capacity running on roasted ore.

There is one more decided disadvantage with which the raw smelting method must be debited. It is a very much more difficult process to conduct than the ordinary smelting of roasted copper ores with coke. The chemical and physical phenomena that accompany the production of heat from the oxidation of carbon are simpler and more manageable than those that are involved in the production of heat from the oxidation of metallic sulphides, and the great amount of solid residues from the combustion of the latter substances also has to be cared for. In plain language, it is easier to burn coke in a blast furnace than it is to burn pyrite.

Consequently, in pyrite smelting the entire operation is very much more delicate and precarious than when using carbonaceous fuel. A shutdown of an hour may affect the behavior of the furnace for the succeeding one or two shifts; great skill and watchfulness are necessary on the part of both metallurgist and furnacemen, and campaigns are seldom very long.

SUMMARY OF RESULTS.

A rough balance sheet may serve to bring out the principal pros and cons more clearly, it being understood that it applies solely to heavy pyritic ores.

Pyrite Process—Advantages.

1. No roasting required. (Saving in plant, in costs, in losses of metal, in time.)
2. Great saving in coke.
3. More silica can be used in charge.
4. More complete elimination of Pb, As and Sb.
5. Heavy spar less objectionable than in ordinary smelting.

Pyrite Process—Drawbacks.

1. Heavier blast.
2. Exceptional care and skill.
3. Shorter campaigns.
4. Possible reconcentration of matte.
5. Zinc more injurious than with a roasted charge.
6. Sulphur fumes are wasted.

Local conditions must decide on which side the balance of profit lies. In the majority of cases the pyrite process will be found the more economical on the heavy sulphide ores that we are now considering. The cost of fuel is the greatest item of expense in ordinary blast furnace smelting, and a method that promises a saving of one-half to nine-tenths of this costly material is entitled to the most serious consideration.

If the smelter is so situated that the sulphur dioxide derived from the roasting of the ores can be used for the manufacture of acid, it becomes a question whether the profit derived from making acid is greater than the gain that will arise from diminishing the coke. It may, perhaps, be found feasible to employ the fumes from the pyrite smelter direct for the manufacture of sulphuric acid. Beardsley finds that the gases from the Mt. Lyell blast furnaces contain little or no free oxygen.

The concentration-smelting of a low-grade matte up to a point suitable for the converters is not so serious a matter as is sometimes believed by metallurgists who have not had occasion to practice it under proper conditions. The simple fusion of matte in the blast furnace (even with the smallest possible addition of coke) is a waste of time, unless the proper chemical reagents are supplied to the charge. It comes out in very much the same condition as it went in. The average enrichment of a 50 per cent matte for one year's work of the former re-melting cupola at the Parrot converter plant at Butte was only about 3 per cent copper, and a considerable proportion of this gain was due to the elimination of the slag that was adherent to the matte when it came to the cupola.

The reagents that must be supplied in order to effect a concentration of the matte are, of course, oxygen and free silica, and, with their aid, a low-grade copper matte may be enriched to 50 per cent, or more, with certainty and economy, and with but a small cost per ton of original ore. As this subject of matte concentration seems to be regarded, in the profession, with a certain amount of suspicion, I quote from two authors who have had large experience with this class of work.

Beardsley says:[1] "Raw-mattes, smelted pyritically, can be concentrated from 2½ to 3½ into 1 without much difficulty; and 10 per cent matte can be made into 35 per cent matte and converted as such; and 20 per cent matte can be raised to 55 per cent matte in a single operation."

Freeland, who has obtained such economical results at the Ducktown Sulphur, Copper and Iron Company's works by substituting pyrite smelting for the ordinary roasting and fusion with coke, says: "The reconcentra-

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[1] *The Engineering and Mining Journal*, June 30, 1904, p. 1035.

tion of the first matte (20 per cent matte concentrated to about 50 per cent copper) presents no difficulties at Ducktown. There is no limit, within the range of matte, to this second operation. A 6 per cent matte may be brought up to 50 per cent quite as successfully as may any higher grade initial matte; nor is this statement based upon odd or occasional samples, but upon more than one carload of matte so made. The degree of concentration, whether in the first (raw ore smelting) or second (raw matte smelting) operation is, in the main, proportionate to the speed at which the furnace is driven, and is controlled by the proportion of quartz on charge, or by the manipulation of the blast, or both. But on these seemingly simple measures hinges not only the grade of the matte, but the life of the campaign."

The presence of even 5 per cent of zinc-blende in the charge adds materially to the difficulties encountered in the raw smelting of heavy sulphide ores. Nutting speaks strongly regarding its injurious effects, and also experiences much trouble from having to use ores in which the sulphides are mostly in a finely divided condition.

When these two drawbacks are not present in an aggravated form, it seems probable that heavy pyrite ores can usually be smelted raw with cold blast more economically, and with less loss of metal, than by the ordinary method, and that even the necessity of a concentration-smelting of the first matte, to raise it to the proper grade for the converters, will not outweigh the advantages of the pyrite method.

As raw smelting demands a more silicious slag than ordinary smelting, it follows that when the charge is a basic one, and no silicious ores are available (as at Ducktown), more barren quartz must be added than in ordinary smelting. We know also that the pyrite furnace requires a powerful blast, and if, in addition, the price of coke is low, wages are light, and climate and conditions are suitable for heap-roasting, I can well imagine that the ordinary roasting and smelting might be the more advantageous of the two. If, however, any of the above advantages are lacking, it does not seem probable that it can compete with pyrite smelting; while if coke and labor are tolerably expensive, and the climate too wet for advantageous heap-roasting, there can be no question as to greater economy of the newer method. If, in addition, there is an available supply of profitable silicious ore, the raw method shows up to still greater advantage.

Since the Mt. Lyell Company has been able to replace barren quartz by a supply of silicious ore of good grade in copper, and can thus eliminate worthless flux, and also produce at the first smelting a matte suitable for immediate converting, it has greatly decreased its metallurgical expenses. During the past half-year it has cost the company only $1.87 per short ton to smelt its raw ores into matte suitable for converting.

The cost of producing metallic copper from the heavy pyrrhotite ores of the Ducktown Sulphur, Copper & Iron Company, Ltd., is 3.018 c. per pound less than by the former method of heap-roasting and smelting with coke. This statement is given on the authority of the smelter superintendent, Mr. W. H. Freeland, and if we assume (without authority) that the ore yields 2.5 per cent copper, the saving effected by the use of the pyritic method is $1.51 per ton of ore. The amount of coke used in this smelting is, I think, about 3 per cent of the weight of the ore, and the capacity of the furnace, as determined after smelting some 70,000 tons of ore by the new method, is 83.5 per cent of its former duty when smelting roasted ore.

The second variety of pyrite smelting, which has been discussed at some length in the preceding articles, is characterized by a comparatively small percentage of sulphides in the charge, and a correspondingly increased proportion of earthy ingredients. In order to obtain the powerful oxidation required for a high degree of concentration, the coke must be reduced almost to the danger limit, and the tendency to chill at the tuyere zone is, in part, counteracted by warming the blast. The rate of concentration is very high, and the matte production small; the slag is silicious to a degree that would seem impossible in ordinary smelting, the elimination of lead, zinc, arsenic and antimony extensive, and the result of the slightest irregularities likely to be serious.

If the furnace becomes too cold, the addition of more coke is impracticable, as it would immediately interfere with the established degree of oxidation, and result in two evils: 1. An increased production of low-grade matte. 2. The consequent robbing of the slag of the iron necessary for its formation.

I happened to be at the Val Verde smelter just after the night foreman had feared the furnace was chilling, and had added a very minute amount of coke to each charge for a short time. In a few hours the inevitable result followed. The matte increased in quantity, and dropped from something like 37 per cent copper down to 30 per cent, and the slag became dangerously acid. Bretherton heated up the furnace and restored matters to their normal condition by using two or three good-sized charges of coke, and substituting for the regular ore charge a properly fluxed mixture of furnace cleanings, foul slag, etc., which already contained its copper contents in the form of properly concentrated matte, and possessed sufficient bases to flux itself, so that the powerfully reducing action of the large coke charge could have no harmful effect on it, and merely melted it and warmed up the furnace at the same time. Thus this thin layer of neutral, and already sufficiently oxidized, material served as a buffer to receive without damage the reducing action that was unavoidable, if an excess of heat was to be quickly conveyed to the lower portion of the cold furnace.

As I have pointed out in a previous article, there is a large field in the United States and Mexico (and, doubtless, in many other countries) for this *partial* pyrite smelting. The abolition of a roasting plant, and the saving of from one-half to four-fifths of the coke, make it peculiarly suited to remote districts where freights and fuel are expensive.

The decrease in depth of the lead contents in many of the limestone deposits of the Rocky Mountains and Mexico, and the increasing production of dry silicious silver and gold ores carrying only moderate proportions of sulphide minerals, point inexorably to the substitution of matte for lead as a collector of the precious metals; and with the comparatively low tenor in copper, and the general complexity of the ores in the regions referred to, there appears to be no method yet invented so suitable as raw smelting with a high degree of concentration; or, as it has been termed in this review—'pyrite' smelting.

Through close competition and long experience, the lead smelter has learned that, if he is to avoid ruinous loss of values, he must keep his slag within certain prescribed and narrow limits. It is not enough for him to hold the *silica* contents of his slag somewhere between 30 and 35 per cent. He must also establish and maintain a suitable ratio between the *iron* and *lime* contents of the slag, and this is a very expensive thing to do, where the requisite ores do not exist in the proper proportions.

It is here that the smelter of roasted copper ore has an enormous advantage over the lead smelter; in a general way it may be said that if he keeps the silica in his slag somewhere between 30 and 40 per cent, and provides ample settling capacity, he need worry very little about the other constituents of the slag, and may vary them greatly every day without serious trouble. This makes the matting operation a much simpler, more universal, and more economical process than lead smelting, and fits it particularly for the handling of miscellaneous custom ores of diverse composition.

The variety of pyrite smelting that we are now considering partakes of both of these systems. It demands a care in the ore-mixture, and an exact determination of the composition of the slag that is to be produced, that is, at first, quite irksome to the ordinary copper smelter. On the other hand, it is, with one single exception, more elastic as to the character of the slag that it may produce than any other smelting operation with which I am familiar. The exception referred to is, of course, the fact that the pyrite furnace cannot advantageously produce a basic slag. Unless there is sufficient free silica present to delay the smelting until the desired amount of pyrite is oxidized to ensure the proper degree of concentration (and, incidentally, to combine with the ferrous oxide thus formed), it will not be pyrite smelting; it will simply be a melting down of the charge into a low-grade matte, as has been so many times insisted upon in this review.

However, in the variety of pyrite smelting that I am now considering, this inability to produce a basic slag will very seldom prove an injury. This kind of pyrite smelting deals mainly with dry silicious ores, and every pound of extra silica that the slag will stand means just so much more profit for the smelter. Consequently, the charge is made up to contain the maximum amount of silica that it can carry, and still melt at a temperature that can be produced economically in the furnace. The metallurgist constantly hovers near the danger limit, and, on the whole, finds his most convenient guide not in the per cent of silica that his slag contains, but in the oxygen ratio between the basic and acid constituents of the slag.

Dr. Carpenter in his daring and successful inauguration of the Deadwood & Delaware Smelter, in South Dakota, relied mainly upon this ratio, and found it a safe guide. He points out[1] that if the slag is kept bebetween $4RO, 3SiO_2$ and RO, SiO_2—that is to say, between a sesquisilicate and a bisilicate—it will generally be satisfactory, alumina being figured as a base. Bretherton, with a much higher per cent of alumina in his ores, finds that he must figure this substance on the acid side, giving it, however, something like two-thirds the replacing value of silica.

This is, however, only one of the two points which determine the proper constitution of the slag. The second point to consider is whether—even though the proper ratio between base and acid exists—the bases are of such a nature that the slag will melt at a suitable temperature. These matters belong to the study of slags, and cannot be taken up in this review, except to point out that the presence of at least 10 to 15 per cent of ferrous or manganous oxide is indispensable.

Where proper knowledge and experience are available, this partial pyrite smelting seems to offer the most economical outlet for the vast supply of low-grade silicious sulphide ores and pyritic concentrates, carrying the precious metals, and but little copper or lead, the production of which is at present hampered by treatment charges which are higher than a large proportion of these ores can bear.

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[1] *Transactions* American Institute Mining Engineers, Vol. XXX, page 774.

CONTRIBUTION BY HIRAM W. HIXON.

The Editor:

Sir—My experience with copper ores does not agree with that gained in smelting copper-nickel ores, and I find it necessary to get out a revised creed and smash a few of my former dogmas.

The difference is radical, and seems to depend principally on the behavior of copper matte as compared with copper-nickel matte. The fusing point of copper matte is much lower than that of copper-nickel matte, which is another way of stating that the freezing point of copper-nickel matte is higher than copper matte in proportion to the nickel contents. The result is that as the nickel increases more fuel is needed on the charge, and this fuel prevents the oxidation of sulphides and makes raw smelting of nickel-copper pyrrhotite impossible. Another feature is that the copper matte can absorb a greater number of heat units in the smelting and give them up to the slag without freezing than can nickel matte. This results in preventing accretions at the tuyeres and the elevation of the smelting zone, which takes place rapidly in the case of copper-nickel ores.

The behavior of the settler on a furnace smelting copper-nickel ores is equally affected by this lack of heat-carrying capacity, and the tendency of the walls to thicken under conditions that with copper matte would cause a runaway, are quite remarkable when observed.

It has been my experience in Ontario that a charge of copper-nickel ores could not be smelted with less than 10 per cent of coke, and here we can run on 5 per cent, smelt 75 per cent of coarse crude ore and make 35 per cent matte at one operation.

Previous to my experience here I had maintained that the fairy tales which were published about raw smelting were without foundation, but I see now that I have been to a great extent the victim of my experience. I believe that, given the proper conditions, with a furnace large enough, and blowing engines to give a positive delivery of blast regardless of pressure, and of sufficient quantity, a furnace may be driven at the rate of 500 to 1,000 tons per day on such ores as Mount Lyell, Tennessee Copper Company, or Rio Tinto with less than 1 per cent of coke, and effect a concentration of 5 or more tons of ore into one ton of matte. Further, that this matte may be concentrated, by re-smelting under similar conditions, to a converting grade.

Few men have the opportunity to experiment with conditions of this magnitude, but I believe that it will be demonstrated that it is possible and more profitable to handle ores of this character in a two-stage raw-smelting plant, with cold blast, than by roasting or with hot blast.

It is only a year since I made a journey of 2,000 miles to see a furnace that was said to be running on a charge of raw ore with less than 3 per cent coke. When I got there it was smelting roasted ore with 10 per cent coke, and I returned condemning the whole story as a myth. I have since found that given the proper conditions it is quite possible, and I hasten to get on the band-wagon and say "me too."

HIRAM W. HIXON.

Tezuitlan, State of Puebla, Mexico, May 27, 1904.

CONTRIBUTION BY WALTER E. KOCH.

The Editor:

Sir—Following up my letter of April 29, I am able now to give further results, covering the test between hot and cold blast. In this last 36-days' run, the cold-blast furnace was apathetic and delicate, while the hot-blast furnace was robust and vigorous. My old 48-in. furnace was handicapped by leaky tuyeres, but still kept ahead of its competitor, which, in spite of being petted and fed with plenty of matte, calcite ore and pyrite, was continually kicking. I find that cold blast means less ore put through, more coke, more flux, more matte and a higher pressure of blast.

The run covered 36 days in April and May.

	A	B
48-in. diam. twin furnaces	Hot blast	Cold blast
Time of run	36 days	36 days
Ore charged per 24 hours	40 tons	33.5 tons
Coke charged per cent of charge	7.25 to 7.5	8.75 to 9
Coke charged per cent of ore	9	11.33
Percentage of ore in charge	82	77

Before starting this run, I moved the blower to within 10 ft. of the cold-blast furnace, and it improved matters greatly. I also used a hot charge with plenty of matte, calcite ore and hot pyrite, giving an easy charge and fluid slag, and under these conditions the cold blast did better. Coke was very bad part of the time—small, dirty and 20 per cent ash. We have to stop and clean out heavy lime deposits due to bad water every four or five weeks, so 36 days is rather a good run. Our normal furnace charge for hot blast is 85 per cent ore, 4.5 per cent matte, 4.5 per cent slag, and 6 per cent coke. The ore contains one-half of one per cent copper.

Slag: SiO_2, 45 per cent; FeO, 37 per cent; CaO, 8 per cent; Al_2O_3, 8 per cent. Temperature of blast below 200° C., never over this. Concentration during this run, 17 to 1.

WALTER E. KOCH.

Santa Maria del Oro, June 3, 1904.

CONTRIBUTION BY G. F. BEARDSLEY.

The Editor:

Sir—I had intended to follow up my first contribution before now. The following notes are suggested by the reading of the articles you have recently published:

I note first that Mr. Koch remarks that "zinc sulphide is a very good fuel." Theoretically it is a good heat-producer, and when burned under proper conditions it will yield a temperature of 1992° C. In blast-furnace practice, however, its heat-producing properties are much obscured by some other properties that it also possesses. Mr. Nutting says it is "an unmitigated nuisance." From an experience a few months ago with pyritic ore containing 6 per cent of zinc, I am prepared to join the latter gentleman's emphatic condemnation of it as against the sustaining authority of Messrs. Austin, Bretherton and Lloyd.

Mr. Mathewson states "fuel can be eliminated entirely provided there is sufficient pyrite in the ore charge." Working a pyritic ore of 90 per cent FeS_2 purity under the most favorable conditions of ore, furnace and blast, and with steady and intelligent furnace crews, seven to nine days were the longest campaigns obtainable without serious trouble. I think 14 days was the longest campaign, and this included an attempt to burn the furnace out with an increase of coke after the trouble had commenced. The experiment of running without fuel was kept up for some time with two furnaces, and the running was altered as suggested by the varying furnace conditions. It was fully demonstrated that running entirely without fuel under the conditions obtainable was neither practical nor economical. The result of the test, however, was the determination of the exceedingly small amount of coke that was really necessary and the way in which it should be fed when used as a corrective for certain symptoms indicative of trouble going on under the surface of the charge. As stated in my former article, on some charge combinations, 0.4 per cent coke was all that was required in a first smelting.

Again quoting from the same authority: "Where the matte is too low grade, by the raw-smelting process, roasting has to be resorted to," and "matte charged into the furnace raw loses hardly any sulphur in the smelting process, as it merely smelts and runs through the charge." Raw mattes, smelted pyritically, can be concentrated from 2½ to 3½ into 1 without much difficulty. Ten per cent matte can be made into 35 per cent matte and converted as such, and 20 per cent matte can be raised to 55 per cent matte each in a single operation. I note that Mr. Fulton smelts his matte several times, but with little concentration so far as the copper is concerned; but his conditions seem to call for it.

Mr. Bretherton mentions, as a result of hot blast, "brighter tuyeres." This was not noticed in the work under my supervision, but rather the contrary. From three to four hours after blowing in, all the tuyeres would become dark with hot blast as well as cold, and no light could be seen in them again during the campaign, unless some hot slag happened to run down an air channel into a tuyere. The blast entering the furnace and meeting the down-coming slag robs it of a part of its heat and produces a chill over the tuyeres of greater or less thickness, permeated by irregular air channels. Before iron, sulphur and the oxygen of the air can unite they must be heated to their combining temperature. In the case of the air it can get only the additional heat between the temperature it has when sent into the furnace and the combining temperature from the chill over the tuyeres kept hot by the down-coming slag and matte running over it. This chill is an interior hot-blast stove in effect, and the endeavor to keep the tuyeres bright in many cases is costing more coke than is actually necessary.

The slag in pyrite smelting begins to run down through the charge at a heat but little above the formation temperature, and there can be but little superheating, so to speak, as the heat liberated by the rapid oxidation of the sulphur and iron is carried away both by the matte and the slag as fast as generated. In the entire absence of coke, the blast, to gain its heat, abstracts it from the slag in immediate contact with the chill, thereby adding to it and forcing the smelting zone higher and higher, until finally the charge smelts on the top and the furnace goes out. Hot blast will retard the growth of this tuyere chill, but will not prevent it. The quantity of coke required, therefore, seems to be an amount necessary to counteract the chilling effect of the incoming blast upon the descending low-temperature slag and to establish a balance which shall keep the smelting zone in its proper place.

Mr. Bretherton further remarks, "hot blast reduces cost of smelting nearly one-half on heavy sulphide ore." This I take to mean an ore consisting principally of pyrite. This effect of hot blast was not observed either, but note that 30c. per ton of ore was saved by doing away with it altogether. I believe that hot blast shows to the best advantage on those ores, or rather furnace charges, which are short in heat-producing iron sulphide. Each worker that speaks so favorably of hot blast, I am sure, is smelting a charge which contains more than 20 per cent silica and less than 30 per cent iron in the form of pyrite. Variations from this limit toward more silica or less pyrite call for more heat, and to furnish the deficient heat a relatively small increase in coke may be augmented by heating the blast.

The observations of Dr. Carpenter on the utility of hot blast are in parallel with my own; but I do not hold with him, however, as to the entire absence of lime in the slag. I have usually aimed at a quarter slag, supplying lime to make up the sum of the earthy bases to that ratio. A one-fifth slag will work well and clean, but reducing the sum of the bases, other than iron, below that ratio raises the iron too high and increases the specific gravity of the slag. Buttons of matte commence to show in the bottoms of the slag pots. In fast driving, on pure pyritic ore, excess of lime will also prevent a good separation of matte by scattering it in minute, hollow beads through the slag.

The smelting practice at the United States works at Salt Lake, up to the converting, as described by Mr. Read, is in many respects similar to the Mount Lyell practice.

Mr. Godshall and Mr. Weinberg ask for more information with respect to the elimination of hot blast at Mt. Lyell. I doubt if very exact information could be obtained for the reason that the variations in the practice were not along a single line at any one time. At the start the blast was heated to between 500° and 600° F., with the pressure varying from 12 to 16 oz. It was soon observed that the furnaces drove faster under increased blast, and when the No. 2 plant was built provision was made for an increased quantity of blast per furnace. Starting No. 2 plant with 18 to 20 oz. of blast, the pressure was gradually increased to 30 oz. The blower room in the No. 1 plant, being most disadvantageously situated on account of the growth of the plant from two furnaces to six, was abandoned, and in putting up the new blower-room the same size blowers were put in as at No. 2, although the furnaces of the No. 1 plant were but 40 in. by 168 in., as against the 42 in. by 210 in. of the No. 2. Greater efficiency in the furnace work reduced the number of furnaces run in the No. 1 plant to three and four, so that within certain limits there was unlimited blast for them. The cutting down of the fuel in the hot-blast stove then commenced, and as the furnaces worked colder more blast was put on until the stoves were taken off entirely and the blast raised to 40 oz.

In addition to the simultaneous variation of blast heat and blast pressure, ores purchased from outside mines came in, and the use of pure silica was more or less replaced by a copper-bearing aluminous schist, until it finally became necessary to run as much of the latter material as possible. A comparison between the cost of smelting at the time of low-blast pressure, hot blast, smaller furnaces and reduced tonnage, as against the time of high pressure, cold blast, larger furnaces and increased tonnage can be made, which naturally is much in favor of the later practice. A direct comparison between hot and cold blast, on a plane of similarity in all other conditions, can not be made for the above reasons. The cost of the hot blast under the earlier conditions averaged about 30c. and under the later condition from 20c. to 24c. per ton of ore.

In regard to the loosely held atom of sulphur in FeS_2: considerable effort was made from time to time to arrive at the proportion of sulphur driven off before actual fusion of the pyrite began. Although nothing very definite was arrived at, the following was noted: It was usual upon shutting down a furnace, as soon as the blast was off, to turn a hose in at the feed doors, cooling off the top of the charge and allowing the men to get in that much sooner for the purpose of cleaning out. At times the shut-down has been immediate; that is, without blowing down, due to failure of a jacket, cooling coil, etc., and consequent flooding of the lower part of the furnace. With the furnace drenched with water, top and bottom, an action, such as roasting, etc., is cut short and stopped. Careful examination of the charge under such circumstances has shown no recognizable ore below a depth of 2.5 ft. from the surface of the charge, the charge column being from 10 to 11 ft. above the tuyeres. The ore, when found, had the composition of Fe, 55, and S, 35.3, which closely approximates the formula Fe_8S_9, the ordinary constituents of the ore making up the balance. The heat of the gases coming through the charge on a normal furnace was several times determined and found to average about 680° F. It would seem, therefore, that from the surface of the charge to three feet below the surface of the charge the temperature must have risen from 680° F. to the melting point of pyrrhotite. At the point of disappearance of the pyrrhotite, the slag, which was the next easiest constituent of the charge to melt, was apparently untouched, so far as heat was concerned, for the edges of the pieces were as sharp as when put into the furnace.

The formation of the pyrrhotite and its sudden disappearance to a great extent proves the absence of any considerable quantity of free oxygen in the upper part of the charge, for Valentine[1] in his experiments found that FeS_2 might be subjected to almost any degree of heat up to 2,600° F., in the absence of air, and retain at least 30 per cent of the sulphur out of the 53 per cent present. Another curious thing noticed was that pieces of slag from low down in the furnace, with the edges melted off, upon breaking after cooling, would often show more or less pyrrhotite in very thin laminæ in the checks or splits in the center of the piece.

G. F. BEARDSLEY.

San Francisco, June 5, 1904.

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[1] *Transactions* American Institute Mining Engineers, Vol. XVIII, page 78.

CONTRIBUTION BY S. DILLON-MILLS.

The Editor:

Sir—Being interested to some small extent in copper mining, I have read with great pleasure some of the discussion on pyrite smelting, and recalling some of my experience of bygone years in blast-furnace work, it has occurred to me that where fuel oil can be obtained at a reasonable cost it would furnish not only a convenient means of experimenting on the amount of extraneous fuel required with different varieties of ore, but also a safeguard against chilling, as in Mr. Mathewson's work; the supply of oil can be adjusted so as to furnish either a reducing or an oxidizing flame, and any irregularities in the working of the furnace can be met as soon as perceived, instead of having to await the working down of the extra fuel with the stock from the furnace top.

I recollect on one occasion having five out of the seven tuyeres of a charcoal blast-furnace closed permanently by a succession of slips when working on a very dusty, highly aluminous, brown hematite ore. Everything looked fair for a regular freeze-up, as we had been having trouble before and were in no shape to meet this additional mishap. I had some oil blow-pipes fixed up at once, and in 36 hours had my tuyeres all bright again, and the iron notch in good shape, it having frozen up solid. This was done with about three-fourths of a barrel of oil, and the furnace saved from freezing solid. If the price of oil permitted doing so, the additional fuel required might be altogether supplied in this way, as the oil blow-pipe can be easily adjusted to form a permanent attachment to the tuyere, without interfering with the line of sight from the 'peek-hole,' to any inconvenient degree. Of course in the case above mentioned the tuyeres had to be removed, but in ordinary cases of chilling I have not found it necessary.

S. DILLON-MILLS.

Toronto, Canada, July 6, 1904.

CONTRIBUTION BY L. S. AUSTIN.

The Editor:

Sir—Dr. Peters, in his review of pyrite smelting in your issue of July 21, leaves the question of the behavior or rôle of zinc and alumina in the slag undecided. In so doing he avoids a controversy which has prevailed among metallurgists in the past, and especially as regards the action of alumina. Many, especially those engaged in iron smelting, have held that alumina is to be regarded as an acid element acting much in the same way as silica. But for those who are engaged in pyrite smelting, where lower temperatures and irony slags are used, the conditions are very different from those of the iron blast furnace. The conditions of the silver-lead treatment of miscellaneous ores are, in many respects, similar to those prevailing in pyrite smelting, with the difference that any departure from a type slag in silver-lead smelting is accompanied by unsatisfactory working and metallurgical losses. Such slags have, therefore, been closely studied. Indeed, much of the success of those engaged in pyrite smelting has been due to the fact that men engaged in it have also operated silver-lead blast furnaces. Indeed, it may be personally asserted that those who have not followed the latter practice are at a disadvantage compared with those who have, because they have not been compelled to study slag composition so closely.

The behavior of zinc and alumina in a slag is such as to obscure the mutual action of its acidic and basic elements, and we are finding to-day a portion of our metallurgists adopting a very different way of looking at these relations, which has resulted in their using a working hypothesis, capable at least of satisfying the varying conditions of their practice. This consists in looking upon both zinc and alumina, not as bases, but as elements singly dissolved in the molten magma; that is, they are non-effective bases, which, as their relative proportion increases, singly stiffen the slag much as sand stiffens mortar. Under this view the variables to be estimated and allowed for are, for the acid constituent, silica, and for the bases, the alkaline earths, iron and manganese. Under this hypothesis, zinc, whether as sulphide or as oxide, stiffens the slag, and where the quantity of slag is increased by the use of a larger proportion of silicious ores, the slag runs smoother. This, of course, is the same as diminishing the percentage of zinc in the slag. Alumina is to be regarded as acting in the same way, and should have like treatment.

To illustrate the manner of estimating the fluxes needed in making a slag of a given type, we will take the case of two slags, known as three-quarter slags; that is, slags in which the ratio of lime to ferrous oxide is as three to four. We have:

	1.	2.
SiO_2	35.4	30.8
FeO (MnO)	31.8	27.7
CaO (MgO, BaO)	24.7	21.5
Al_2O_3, ZnO, ZnS, K_2O, Na_2O	8.1	20.0
	100.0	100.0

In the first instance (1) the alumina and zinc are small relative to the active bases, while in the second (2) the percentage is large. The ratios of silica to the active bases, it will be observed, remain the same. In the second case (2) the zinc and alumina become too much to be capable of being dissolved in so small an amount of accompanying silicates, and hence the reason why I have, in a former article on pyrite smelting, limited the zinc to 10 per cent of the slag for easy running. However, while the proportions given are ideal for obtaining lead-free slags, when it comes to the question of fusibility only, the ferrous iron may be advantageously increased and the lime decreased from the proportions just given. What I wish particularly to bring out is that, in our working hypothesis, neither zinc nor alumina is to be reckoned as an active base, nor is the latter as acidic in its action. When an aluminous slag has a so-called acid appearance, it is due, not to the alumina acting as an acid, but because it is a solute which has made the molten mass more viscous.

The preceding observations apply to type slags which do not exceed 36 per cent silica. When we come to the acid slags, other conditions prevail. Since any given element in a slag seeks to combine with other elements in proportion to its needs, it follows that the large proportion of silica present is eager to regard alumina as a base, and it no doubt does so. How, otherwise, can we account for the fusibility of the viscous high-silica slags, sometimes reaching 65 per cent silica? Such slags have a corrosive effect upon basic or even on neutral linings, and where lead is present are cleaned from it with great difficulty.

L. S. Austin.

Houghton, Mich., Aug. 8, 1904.

CONTRIBUTION BY HERBERT LANG.

The Editor:

Sir—Properly conceived answers to the ten questions on pyrite smelting which have been propounded, would seem to cover the purely practical features of the art very satisfactorily, but their bearing is too severely practical to admit of as broad a treatment as the subject in general demands, or, perhaps I should say, as the requirements of the metallurgical student make necessary. May I suggest, then, in view of the probable perpetuation of the replies in book form, that an extension of the topic along more theoretical and general lines would be welcome to the beginners in smelting, to whom, I suppose, all such publications are ultimately addressed? At the risk of trenching upon your valuable space I will take the liberty of propounding two more questions, in my opinion of a nature to bring out the general relations of the pyritic to the ordinary matting process, and will provide these with the very briefest of answers, more in the hope of evolving the opinions of others than in the expectation of adding a lasting and important part to the completed treatise.

The questions that I would ask and answer are these:

What are the differences in principle between the pyritic and the ordinary matting process?

What are the differences in practice?

Question 1.—Before attempting to consider the differences in theory between the two processes, it will be useful to briefly review the state of opinion regarding the principal chemical reactions which occur in the ordinary form of matting. In this connection it is the custom to group the phenomena in three classes; the first including those reactions that furnish heat, the second those to which slag formation is due, and the third those concerned in the production of matte. Of the first class, the only one which concerns the present inquiry is the reaction between the oxygen of the blast and the carbonaceous fuels. The imported products of this reaction are carbon monoxide and carbon dioxide, both of which, being gaseous, tend to escape into the atmosphere. Incident to this reaction is the reduction of certain iron and manganese compounds from the condition of peroxide to that of protoxide, and of lime from that of carbonate to the caustic form, fitted to combine with the silica present and to carry out the reactions of the second class. The latter consist in the combination of the thus prepared and other bases with the free silica, or the re-formation of previously existing silicates—all these products becoming dissolved into slag. Attendant phenomena, as the reduction to the metallic state of small quantities of iron, and the production of heat through the silicatization of oxides, may be disregarded in this connection. The third group of reactions is more obscure, but their net result is the production of matte by the agency of sulphur, which attacks with varying degrees of energy the metals exposed to its influence, allying itself preferentially with the copper, the silver and the nickel, and secondarily with iron, lead, etc., and forming fusible and comparatively stable compounds with all of them. This bald statement of accepted fact must serve as a basis of comparison between the two processes.

In pyrite smelting the fuel, instead of coke, is substantially the sulphide of iron. It may be matte which, for the second time, is passing through the smelting furnace; it may be pyrrhotite; or it may be pyrite which, in descending to that part of the furnace where its fuel properties become available, has parted with a portion of its sulphur. No other sulphide which possesses the requisite properties is found in sufficient quantity to make it available in the process. The reactions produced between the molten sulphide and the air blast are comparable in their heat-making powers with those between carbon and the air; but here the comparison ceases. In pyrite smelting the three classes of reactions previously cited, namely, the heat-making, slag-making and matte-making, are concurrent and coincident to a remarkable degree, which furnishes the first point of theoretical difference between the processes. Thus, the heat-producing reaction between the metallic base of the sulphides and the oxygen of the blast has, as a secondary and simultaneous effect, the production of oxides for the formation of slags, and it is this detail of the reaction which constitutes the indispensable characteristic of the operation and differentiates it from all other smelting processes. In a previous writing, I took occasion to point this out as the chief characteristic of the method, and to deduce from it the definition of the process as follows:

Pyrite smelting is that department of blast-furnace smelting in which metallic oxides are formed in the furnace.

This effect, while not absolutely peculiar to the pyritic method, yet furnishes its chief characteristic.[1] By this one reaction (or rather by the group of reactions which we are in the habit of typifying by this one) the heat necessary for the work is generated, and the principal slag-making bases necessary for absorbing the silica present are produced. Notice the peculiar reciprocal relations of the matte-making, heat-making and slag-making forces. With a given charge, the heat set free and the amount of oxides formed are in inverse

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[1] The unimportant formation of ZnO from ZnS in the lead furnace has been instanced as affecting the theoretical accuracy of this definition.

ratio to the amount of matte made. The greater the quantity of sulphides decomposed, the greater the quantity of heat generated, the greater the amount of iron driven into the slag, and the less the amount of iron remaining to swell the matte-fall. This feature of the process is very remarkable indeed, and is worthy of much consideration by the student. He should not fail to bear in mind that it is the iron which really does the work, and that without it we could have only the faintest approximation to pyrite smelting. It is idle to speak of zinc as being in any sense a useful substitute for iron in pyritic work. Iron is the key of the process and always will remain so.

The simplest practicable charge for the pyritic blast furnace consists of iron sulphide mixed with quartz. The ready fusibility of the former, coupled with the infusibility of the latter, bring about some significant conditions which are receiving much attention of late. The sulphide melts at a very moderate temperature, say, of about 1,000° C., at which the quartz is not in the least affected. When thrown together into the furnace, the sulphide melts before it has got down far, and the liquid tends to descend into the hearth, while the quartz sinks by itself in a mass downward to the tuyeres. The fluid sulphide, getting hotter as it descends through the interstices of the quartz mass, meets the upward current of air from the tuyeres, while both, as well as the quartz, are at their hottest, and in condition to be most quickly acted upon. The reaction that ensues results in cutting away the quartz mass, which is as rapidly supplied by fresh fragments from above. Slag is thus produced, while the residue of the melted sulphide forms matte, and both gather in the hearth. No doubt the effective heating of the blast, so often alluded to by contemporary writers, is done by contact with the descending fluids rather than by the hardened masses of partially cooled slag which attach themselves about the tuyeres, forming noses. It is very evident that if a mass of slag stops on the road down its usefulness as a heating medium must soon be lost, for the heat contained in it would be as nothing compared with that necessary to elevate noticeably the temperature of the blast. It is the sinking mass which heats the blast. We may recognize virtually two working zones, the one being that where the sulphides are melted, the other that where the infusible materials await the action of the fused sulphides. This statement, which recognizes the perpetual existence of what I have called the quartz mass, better characterizes the condition of things than that wherein the delayed mass is spoken of as accretionary and accidental—a view got by inspecting the furnace's interior through the tuyeres.

When more complex charges are fed, containing limestone, natural silicates, slag, and perhaps other materials, more complicated reactions will ensue, and somewhat different results will be got. The quartzose mass will still exist in form, but its composition will have changed. It will still be infusible by itself, since it is not conceivable that the pyritic process could go on without the aid of the iron oxide in the formation of slag. We may believe that the quartzose materials are cut away by the descending matte stream, something in the manner that ice is cut away by a stream of hot water. It being the freshly formed oxide that unites with the silicious matters, it is evident that the quartz must remain until enough oxide is formed to destroy it. If there is more matte than necessary for uniting with the silica, it brings up in the hearth, carrying in combination with it the gold, silver and copper of the charge.

If we may judge by what they have written, it is the general impression among metallurgists that the controlling reaction of the pyritic furnace should be represented thus:

$$\text{(A)} \qquad FeS + 3O = FeO + SO_2.$$

And that this is succeeded by another reaction, in which the ferrous oxide is scorified and added to the slag as silicate. Uncombined oxygen must, according to this view, exist in the lower part of the furnace, else there would be no decomposition of the sulphide. Also the decomposition, with concurrent production of heat, would be in proportion to the amount of air present. Then the quantity of air present would inevitably govern the smelting, without regard to the incombustible constituents of the charge. According to this view, it would be necessary to have at all times a certain amount of free oxygen in the furnace, else the work would cease, and neither heat, slag nor matte would be produced. Some considerations affecting these results make it seem improbable that this can be precisely the way in which the effects are brought about. Let us suppose a furnace working on the pyritic plan, wholly without carbonaceous fuel. Next let us suppose that a portion of coke be added to the charge but which remains otherwise as before. The coke having a greater affinity for the oxygen than any other substance present, the effect will be to diminish the quantity of the oxygen available for decomposing the sulphides, and, as a consequence, there will be less oxide of iron set free to slag the silica. The matte, being swelled by the addition of the unaltered sulphides, will fall more copiously. The effect of the increased acidity of the slag will be to cause the furnace to run slower, and perhaps to close up entirely from the formation of incrustations about the smelting zone. But such is not the experience. We may add 5, 10 or even 15 per cent of coke to such a charge without inducing any such effects, or even without any injurious effects at all. When the coke exists in so large a proportion, it is difficult to see how the atmosphere in the shaft can remain in a condition to bring about the real pyritic effects if they depend so entirely upon the presence of free oxygen, and especially if we remember that much satisfactory pyritic work has been done with as high a coke charge as 12 and even 15 per cent. These high coke percentages, which to the everyday view would seem to forbid pyritic work entirely, really allow uncommonly extensive decomposition of the sulphides, evidenced by a high-grade matte product, while the work goes on swiftly and regularly. Much evidence might be cited to prove that the strongly oxidizing powers inhabiting the interior of the pyritic furnace are not so closely contingent upon the assumed presence of free oxygen as has been assumed. In such cases we have not only the presumptive evidence of the coke charged, but that of the evolved gases, which often have been observed to contain free elemental sulphur, sulphide of arsenic, etc., which could not exist in the presence of free oxygen in the heated furnace for a moment. What lessons do we draw from these suggestive facts?

In the first place, may we not question the accuracy of the notion that free oxygen is the governing factor in the case? May it not be that silica rather than oxygen is the controlling factor? My view of what takes place in the carbonless charge I have embodied in the following equation, which seems to me to conform more closely than the other to what is known to occur:

(B) $2FeS + SiO_2 + 2O = (FeO)_2 SiO_2 + S_2$.

This equation indicates the simultaneous decomposition of the sulphide, the slagging of the produced oxide of iron, and the expulsion of the sulphur. The latter will burn if there be free oxygen to combine with it, but if not it will issue from the stack in the elemental form. Usually it passes off as a mixture of sulphur with sulphur dioxide. When it exists in the stock in large proportions it is impossible to burn all to dioxide. In the converter process the case is the same, part of the sulphur escaping as dioxide, part as elemental. In both processes the iron is found to be far more avid for oxygen than the sulphur, whence the latter is kept from combination when there is not enough oxygen for both the elements. From a heat-producing point of view sulphur is not of great importance in either process, although the impression is very common that pyrite smelting owes its existence to the heat-giving effects of the sulphur. I presume that if we were to examine closely into the rationale of such a process as heap-roasting, for instance, we should find that to the oxidation of the iron and not to that of the sulphur were due its principal effects, and that the latter played very little part in the work.

The familiar assumption that the pyritic effects are only possible in the presence of a large proportion of free oxygen, seems to be shaken by the foregoing consideration, and it even may be rationally asked if any free oxygen at all is necessary to the process. Evidently oxygen must be had from somewhere, and if we imagine conditions wherein all that contained in the air would be taken up and wherein none in an available state is brought in by components of the charge, it would still be an unsettled question whether the process could go on or not. That it can go on in the presence of 15 per cent of coke is the most remarkable peculiarity of the process yet made out. The probable effects in eliminating the oxygen at a short distance from the tuyeres naturally lead to the important question if the oxygen necessary may not be derived in such cases from the carbon dioxide previously formed by the action of the blast upon the fuel. This is not an unlikely supposition by any means, as analogous reactions are not uncommon. Further research is necessary, however, to settle the subject conclusively. If we assume that the dioxide has such an action, it would put the process on a very different plane from that it now occupies, but would still leave us in the dark as to the extent to which the addition of coke could be carried. It is quite evident that ultimately a point must be reached at which a further addition would be fatal to the process, as taking up all the available oxygen with the formation only of monoxide, which can hardly be expected to have any action on the sulphides, although it is known to have effect upon certain metals of the iron group. The chemical formula which represents the interaction of iron sulphide upon silica and carbon dioxide is:

(C) $2FeS + SiO_2 + 2CO_2 = (FeO)_2, SiO_2 + 2CO + S_2$.

My views as to the functions of silica in the pyritic furnace were first published in *The Engineering and Mining Journal* as early as 1897. I positively assert the validity of reaction (B) as the real controlling and characteristic reaction of the pyritic furnace, although not by any means the only one that occurs or may occur therein.

Equation (A) I am in the habit of calling the iron-sulphur-oxygen reaction, while (B) in my terminology is the iron-silica reaction. In both I assume that the singulo-silicate of iron is formed, which I believe to take place invariably in this form of smelting.

The differences between the slag—and especially the matte-forming activities of the two processes—are exemplified by the behavior of the inferior elements, arsenic, antimony, bismuth and lead, whose expulsion from the pyritic charge affords an unequaled means of purifying the furnace products. A discussion of these effects, or even a further reference to the general question of slag- and matte-formation, would occupy too much space for the present, and I will pass on to Question 2—*Differences in practice between pyritic and ordinary matting.*

Neither furnaces nor accessory apparatus have been much modified as yet by considerations of theory.

In the early days of pyrite smelting, when light blast and slow driving were the rule, furnaces with straight sides had their advocates, myself among them. Their virtue, probably a real one, has been obscured by the faster and harder rate of driving now in vogue. The universal tendency has been to reproduce the familiar forms of the lead- and copper-smelting furnaces and adjuncts, without regard to possible improvements other than increased size and blowing power. Blast pressures have been increased quite up to the limit of the ordinary rotary blowers, and the tendency being toward still higher pressures, it seems as if piston blowers are likely to become fashionable in this pursuit, especially as it is declared by some practitioners that a high-pressure cold blast can be made to serve the same purpose as a warm or hot one of lower pressure. The choice evidently is to be determined by which is the cheaper, the cold, hard blast or the warm, or hot, light blast. If I interpret the progress and the tendencies of the age aright, the time is not far away when pyrite smelters will be borrowing the furnaces and apparatus of the pig-iron makers. In fact, I have for some time believed that we could not do better than put up works in precisely the style of the great iron blast-furnace plants of the East to do pyritic work in. The furnaces doubtless would have to be cut down somewhat, but aside from this I can see no alteration that good sense could dictate. The blowing engines, the firebrick hot-blast stoves, the charging apparatus, the slag- and metal-conveying machinery—all these are so much more effective than the comparatively trivial and labor-wasting tools of the copper smelter that there is no comparison.

HERBERT LANG.

Oakland, Cal., August 8, 1904.

CONTRIBUTION BY WALTER E. KOCH.

The Editor:

Sir—The following additional notes may be of service: We find slags below 5 per cent in lime chill quickly, and that slags above 10 per cent in lime do not separate out the values so well as when kept below 10 per cent lime. A 7 to 8 per cent lime slag is ideal. On cleaning flues we found the cold blast made more than double the quantity of flue-dust made by the hot blast, and, worse still, the cold-blast flue-dust from the same ores contained double the value of the hot-blast flue-dust. Altogether, as compared with hot blast, the cold-blast runs were miserable and expensive failures. In July we had another freeze-up on cold blast. The July run showed a trifle better in coke consumption (10.5 instead of 11 per cent), but the ore charged was only 30 tons per 24 hours and the matte outfall was unsatisfactory. However, with July our cold-blast runs came to an end—for which we are truly thankful. This month we hope to run with hot blast, gas-engines and modern appliances and leave behind us the Paleozoic period of cold blast and steam-engines. With hot blast, gas-engines and electric transmission, the twentieth century smelter has a brilliant and economical future.

WALTER E. KOCH.

Santa Maria del Oro, Sept. 2, 1904.

CONTRIBUTION BY G. F. BEARDSLEY.

The Editor:

Sir—Before the collected articles on pyrite smelting appear in book form I would like to add a few words on the effect of alumina in the pyrite furnace.

It has been noted that the furnace, in this process, seems to select its own slag. This is quite true, and an apparent reason is at first offered by the fact that the iron oxidized by the furnace is a factor in the formation of the slag over which the metallurgist has but little control. The quantity of lime may be varied in the slag within certain limits, so also any other constituent added, but not so the iron. Small causes will vary the rate of oxidation in the furnace, and consequently the amount of iron oxide supplied to the slag, but this variation does not affect it materially. The relation of silica to iron remains about the same.

The true reason of the pyrite furnace apparently selecting its own slag will be found in the fact that the slag is emitted from the furnace at the lowest formation temperature possible with the slag-forming material supplied. Matte and slag that are free to move from a source of heat will commence to flow at a temperature not far above that of their formation. If the flow is hindered by their having to make their way or part of their way through a mass of incandescent material, or by having to move by a tortuous passage to an exit, they will continue to absorb heat only until the heat imparted to them is balanced by the heat removed by the rapidity of the flow.

The oxygen ratios in the slags formed by this method of working vary from the singulo-silicate, in which the ratio of the oxygen of the acid to the oxygen of the base is as 1 to 1, to those slags in which the ratio is as 8 to 5. The majority of slags fall within the 4 to 3 ratio. The following block out of Prof. Richards' slag table, as published by Hofman,[1] covers the whole range of pyrite slags, as far as I am aware, my observations extending over some 1,800 slag analyses made in three different works.

Formulæ of Silicates.	O in bases to O in acid.		Percentages.			
$2RO\ SiO_2$	2 : 2	FeO		57.58	53.24	48.90
		CaO		12	16	20
		SiO_2		30.42	30.76	31.07
$5RO\ 3SiO_2$	5 : 3×2	FeO		53.58	49.24	44.93
		CaO		12	16	20
		SiO_2		34.42	34.76	35.07
$3RO\ 2SiO_2$	3 : 2×2	FeO	55.60	51.20	46.70	42.25
		CaO	8	12	16	20
		SiO_2	36.40	36.80	37.30	37.75
$4RO\ 3SiO_2$	4 : 3×2	FeO	52.66	48.22	43.78	39.34
		CaO	8	12	16	20
		SiO_2	39.34	39.78	40.22	40.66
$5RO\ 4SiO_2$	4 : 4×2	FeO		46.62	42.17	37.72
		CaO		22	16	20
		SiO_2		41.37	41.83	42.28

If these are traced through his tables of formation temperatures they will all be found to be among the lowest temperatures determined.

With a pure pyrite or pyrrhotite ore, to which, if not already present, silica is added, there will result, after an oxidizing smelting, a simple silicate of iron of low formation temperature and low melting point. Lime added to this in proper proportion invariably lowers the formation temperature, and some of the lime may be replaced by alumina without materially affecting the slag in this respect. The general tendency of alumina, however, is to raise the melting point of slags and to increase their viscosity, the latter being a very undesirable quality, as it interferes with the proper settlement of the matte.

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[1] 'Temperature at which Certain Ferrous Silicates and Calcic Silicates are Formed in Fusion,' etc. *Transactions* American Institute Mining Engineers, Vol. XXIX, page 682.

It is seldom that uncombined alumina forms a part of the smelting charge, for the reason that pyrite ores are usually associated with schist, porphyry, etc., and often the ore itself contains aluminous minerals, as is the case with the Tennessee pyrrhotite. If there is added to the smelting charge a natural silicate of alumina of the bi- or tri- or higher silicate order, there will be no breaking up of this combination, as the heat of the low temperature slags is not sufficient to effect it. There will be dissolved just so much of this aluminous silicate as the slag can carry at the given temperature, and no more. If there is more of this silicate than the slag can dissolve, it is left behind in the lower part of the furnace, and the fusion zone is forced higher and higher until the charge finally smelts on top and the furnace suddenly freezes up.

In furnaces running on a reduction smelting, with 9 to 15 per cent of coke on the charge, slags from the fusion zone above the tuyeres continue to absorb heat or become superheated by running through the incandescent fuel below, and, as their heat rises, they become more and more capable of breaking up or absorbing combinations having a higher melting-point.

It seems to me that the slags first formed in any blast-furnace work must be of the simplest, low-temperature formation, silicate order that can be made from the constituents present at that point, and as the superheating goes on, silicates of a higher formation temperature are merely dissolved in it with a probable decomposition after solution.

The working limit of alumina in pyrite slag made direct from ore is about 7 per cent, and although this may be forced to 8 per cent or 9 per cent for a time, trouble is pretty sure to result. A furnace concentrating low-grade matte, from the first operation to a 40 per cent or 50 per cent converter matte, runs very much hotter than an ore furnace, and under these conditions it should be possible to add to the slag a full 10 per cent of alumina through the fluxes, if required.

To illustrate the effect of an excess of an aluminous material in pyrite work, the following is cited: At a plant which had been heap-roasting their output for a number of years an ordinary orthoclase quartz-porphyry was used as a flux for the roasted material, and the furnace ran very well. The analyses of the green ore (unroasted pyrite) and the porphyry were as follows:

	Green Ore.	Porphyry.
Silica	5	77.30
Iron	35	2.75
Alumina	Trace.	15.20
Lime	Nil.	0.63
Zinc	6	....
Sulphur	43	....
Copper	6	....

With the exception of possible trouble with the zinc, the ore seemed eminently suitable for pyrite treatment, and this was resolved upon. As quartz was a somewhat expensive flux and the porphyry exceedingly cheap, an endeavor was made to retain the latter, though it was soon found to be conducive of bad work. Later an experimental run was made, with the following results: The furnace was blown in as usual at 9 a. m., and run on a quartz flux until 12:50 p. m., to make all furnace conditions as perfect as possible; at that hour no other alteration was made in the charge except to change quartz for porphyry. At 4 p. m., three hours and ten minutes after feeding the first porphyry charge, the furnace began to show signs of distress, and at 6:30 p. m. slag and matte ceased to run. The furnace was allowed to cool down and the end jacket removed. A section was cut down on the chilled material, and the following noted: The unfused porphyry was found to occupy a space extending from some six inches below the tuyeres to the same distance above the bosh line, a thickness of about 2 feet in the middle of the furnace and 2 feet 6 inches to 3 feet against the jackets. Above this the unfused portion of the charge extended to the feed-floor.

A portion of the unfused porphyry was broken out and carefully separated into white unfused porphyry and interstitial slag, and analyzed, with the following result:

	SiO_2	FeO	CaO	Al_2O_3	ZnO
Porphyry flux	77.30	2.75	0.63	15.20	....
Unfused porphyry	81.80	4.81	0.91	11.45	....
Interstitial slag	57.70	23.01	4.09	15.33	....
Free-running furnace slag	33.30	48.75	5.10	7.20	4.41

The viscid high temperature interstitial slag remained near where it was formed, and, blocking the air passages through the porphyry flux, cut off the blast from the upper part of the charge. Later experiments, in conjunction with quartz, proved its entire unsuitability in any proportion. The aluminous material used at Mount Lyell is a pyrophyllite schist, the aluminous mineral being a bisilicate and very finely diffused through the rock. The aluminous minerals in the quartz porphyry with orthoclase and oligoclase were trisilicates, and were distributed in large grains and crystals.

G. F. BEARDSLEY.

New York, Oct. 10, 1904.

CONTRIBUTION BY LEWIS T. WRIGHT.

The Editor:

Sir—Having given considerable attention to the smelting of raw-sulphide ore, with hot blast and without coke, at Keswick, I am in a position to answer many of the questions you submitted to me a year ago.

Pressure of business has not given me time to attend to this matter until now, and, if not too late for your purpose, I will give you a summary of my experience. The time at my disposal even now does not permit of going into the subject with much theoretical detail, and I must be content, therefore, with a brief recapitulation of facts.

The ore is cupriferous iron pyrite, rich in sulphur and free from gangue. The first practice was to roast the coarse ore in large heaps. This practice was carried out on such a scale that it was found economical to load the roasted ore into railway cars by means of steam-shovels. The fine ore was roasted sweet in mechanical furnaces with water-jacketed arms and stirring columns. This type of furnace, now very common, was first developed at Keswick. In spite of an early failure in pyrite smelting, Mr. Fielding, managing director of the Mountain Copper Company, remained convinced that the practice was economically possible, and, therefore, encouragement and impulse were given by the board of directors to our efforts to increase the amount of raw sulphide smelted and to reduce the use of coke. About three years ago the heap-roasting of the coarse ore was finally abandoned, and it was smelted direct in its raw condition; later, the fine ore was no longer roasted, but also was smelted raw.

Your first question asks: "What types of ore are suited to the process?" My experience only relates to those containing iron and sulphur, and these, when smelted with a suitable flux, certainly can be smelted raw without the use of coke and with a satisfactory degree of concentration.

Your second question, "Is hot blast advisable?" I can answer in the affirmative. Several years ago, whilst studying the question of hot blast, I had occasion to converse on the subject with an eminent metallurgist. His opinion was distinctly unfavorable to the use of hot blast, on the ground that it was not logical to employ a roundabout process, to burn fuel outside the furnace in heating the blast, and then to conduct the heated air (with inevitable loss of efficiency) to the furnace, when the same fuel might be used direct in the furnace and to more effect. The argument impressed me for the moment, but fortunately, on consideration, it occurred to me that it was not yet proved that we obtained the full efficiency of the carbonaceous fuel, charged into the furnace at the top, and that, probably, some portion of this fuel might be consumed towards the top of the furnace, and before it reached the smelting zone, and thereby partly lose its effect.

One of the principal objects I had in view in the use of hot blast was, in the case of raw-sulphide smelting, to increase the concentration, or, in other words, to increase the oxidation so as to obtain a higher grade of matte in the first operation. Carbon is a reducing element and is counteractive in respect to concentration and oxidation. If, therefore, we could eliminate the carbon from the furnace, we should have more of the oxygen of the blast left for the oxidation of the sulphide and the grade of the resultant matte would be higher. I determined to try hot blast, and my early experiences demonstrated: first, that a certain quantity of carbon, consumed outside the furnace in heating blast, replaced a larger quantity charged at the top of the furnace as coke; second, that there was a marked increase in the concentration; and, third, that there was a great improvement in the running of the furnace. These experiences fully justify the answering of your second question in the affirmative.

In reply to your third question, I can say that carbonaceous fuel in the furnace can be, and is, entirely eliminated in the smelting of raw-sulphide ores when the conditions permit of the oxidation of the iron sulphide, and the slagging of the oxidized iron at a certain rate per unit of time, and at a certain proportion per unit of charge.

Your fifth question I would answer by saying that lime is as useful in lowering the copper content of the slags in the case of smelting of raw, as in the case of the smelting of roasted ores.

Your sixth question, as to the percentage of zinc that can be profitably treated in the charge, is difficult to answer. Zinc is as troublesome in the case of raw-sulphide smelting as it is in the case of the smelting of oxidized ores. As the percentage of zinc rises, more fuel must be used, whether the ores be roasted or raw. Pyrite containing 5 per cent zinc can be smelted with hot blast without the use of coke.

The answer to your seventh question, as to the desulphurization attainable, depends very much upon the nature of the flux, because the rapidity of the oxidation varies with the chemical character of the flux; 80 per cent of the sulphur should be volatilized as SO_2.

Your eighth question, as to the possibility of the capacity of a furnace, reminds me that the furnace campaigns smelting raw-sulphide ore entirely without the use of fuel are short. The furnace in its running period will run as nicely as with roasted ore and coke, but when the furnace gets into bad condition, and begins to slow down, no attempt is made to save it in any way by the use of coke; and if it does not recover of itself, it must be tapped out and another furnace started in its place. The limitations of the process must be the economical ones, and, in that way, I would answer your ninth and tenth questions.

It has been demonstrated at Keswick that the use of coke can be entirely eliminated in the smelting of raw sulphide, even when the charge contains a large quantity of slag, converter slag, flue-dust, and furnace-cleaning, if the flux is sufficiently acid.

It may appear singular that raw-sulphide smelting has not been more practiced and that it is only recently, so to speak, that iron pyrite is being smelted with its own fuel. It is true that an appeal to theory indicates that the oxidation of iron pyrite should yield enough heat to raise the temperature of its smelting products to a state of fusion; but it was not found easy to put theory into practice. It is really easy if suitable conditions are present, and if one is prepared to dig out a furnace at a cost of $50, rather than to save it at a cost of several hundred dollars' worth of coke. The fact is that in practice the heat afforded by the combustion of the pyrite does not leave very much margin for the pyritic-smelting process. In the first place, one must recollect that there is a difference between carbon and sulphur in respect to combustion. It is possible to effect the oxidation of carbon in a furnace without the use of any excess of air. This is not the case with sulphur and sulphide, which cease burning before the surrounding atmosphere is exhausted of its oxygen, and whilst it already contains a certain, and not small, percentage of this element. The sulphur dioxide forms what is called an extinctive atmosphere, and when it has reached a certain proportion, or tension, say about 12 per cent, the reaction, $S + O_2 = SO_2$, ceases entirely.

From a reference in Roberts-Austen's 'Introduction to the Study of Metallurgy' I infer that Plattner was aware of the effect of this limiting atmosphere in the roasting of sulphide, and that a certain excess of air must be employed. I have had the opportunity of observing this limit with various furnaces, such as blast furnaces, roasting furnaces, converters, etc., and have found it much below that which would be the case if no excess of oxygen were required, as it is. Here is one reason why it has not been found easy to reach the smelting temperatures by the simple oxidation of pyrite. Another reason that comes into prominence is the speed of the reaction. In practice we cannot escape from the element of time, an element respecting which the chemical reactions, as graphically set forth in our text-books, are silent.

There is a certain working temperature necessary to produce the minimum flow of molten material from our furnaces, and to produce and maintain this working temperature a certain minimum amount of heat must be produced, per unit weight of charge per unit of time. If the heat production in unit of time should fall below this amount, the process of smelting would cease. The substances we employ in the charge must, therefore, produce heat at a certain rate. In the case of iron blast-furnace practice, it has been shown by Lowthian Bell how the heat required, per unit of charge, is distributed to meet losses of heat (by radiation, expansion of blast, tuyere water, fusion of slag, and in escaping gases), and to carry on the chemical reactions of the furnace, such as the reduction of the ferric oxide. In the case of cold blast, he found that, per unit of charge, a total of 1.73 Cal. was required. Dividing this into two categories, I find that 0.79 Cal. was required by the chemical processes, and 0.94 Cal. (and in case of very high furnaces and hot blast as low as 0.67 Cal.) for the physical heat processes. In the case of iron smelting, higher temperatures are required than with copper. For the purpose of illustration, let us suppose that the chemical reactions in the blast-furnace are neutral, neither yielding nor consuming heat. In that case about 0.7 Cal. per unit weight of charge should provide for all heat requirements, and keep the charge melted and flowing. Since one unit weight of carbon gives 8.08 Cal., 0.083 weight unit of carbon will give 0.67 Cal.; 10 per cent of coke (containing 83 per cent carbon) burnt to CO_2 will furnish the heat for the work to be done, supposing that the chemical reactions in the furnace are neutral, and that we require for copper furnaces the same degree of heat as is required by iron blast-furnaces. FeS gives on combustion 1.29 Cal. per weight unit. A charge of FeS would give the required heat of 0.67 Cal., if 0.52 weight unit of the FeS were completely oxidized per unit of charge. The point in practice is to get the iron sulphide to oxidize at a sufficiently rapid rate. If the conditions are not favorable to the production of this rate, naturally the operation will fail.

I have not enough information to be able to name with the same degree of certainty, as in the case of iron blast-furnace practice, the amount of heat required per unit of charge to carry on the copper-smelting operation. The matter is further complicated in the case of iron pyrite because we do not know just how much heat is produced by the oxidation of iron pyrite, and, secondly, we do not know in what manner the sulphur is burned. One-third of the sulphur of iron pyrite is volatilized below its melting point. This sulphur escapes at the top of the furnace, making a hot top, and does not produce its full value in the smelting operation. We all know, however, that we can smelt a charge, consisting chiefly of roasted ore, with less than 10 per cent coke with cold blast. Ten per cent of coke containing 83 per cent carbon furnishes 0.67 Cal. per unit weight of charge. We ought not to require so much, judging from iron blast-furnace practice. FeS gives per unit weight 1.29 Cal., therefore, if we could burn 0.52 weight unit of FeS per unit weight of charge, and could consume this proportion of the charge at a sufficient rate, and slag the iron, the process should go on.

I have no experience with pyrrhotite, but on referring to the interesting account given by Mr. Freeland, of Ducktown,[1] in his case, I find that 0.292 Fe went to slag, and 0.165 S went to SO_2 per unit of charge. I will assume the total of these two components, that is, .457, to be calorifically equal to FeS. Thus 0.457 by 1.29 Cal. = 0.589 Cal. generated per unit of charge. That does not seem to be quite enough, and I am, therefore, not surprised to find that 0.023 weight unit of carbon was also consumed. This quantity is calorifically worth 0.023 by 8.08 or 0.186 Cal., or in all a heat production of 0.775 Cal. per unit weight of charge, which, from comparison with iron smelting practice, should be enough. In our case at Keswick we can smelt raw sulphide without coke when about 0.21 of iron goes to slag and 0.24 of sulphur goes to SO_2 per unit of charge, and we use 0.10 Cal. in heating the blast.

I do not know the calorific value of pyrite. One-third of the sulphur volatilizes below the smelting temperature and its heating effect is mostly lost in the gases. If we give to the FeS of the FeS_2 its value of 1.29 Cal., and to the second atom of sulphur one-third of its calorific value as an element, we have in all for the 0.45 unit of pyrite:

0.43 FeS @ 1.29 Cal.	0.55 Cal.
0.12 S @ 2.2 Cal. x ⅓	0.09 "
Hot blast	0.10 "
Total,	0.74 Cal.

produced per unit weight of charge, not allowing anything for the heat of combination of FeO and SiO_2.

I hope I shall not be criticised for being too approximate in my estimates of the heat requirements of the blast-furnace, but we have no means yet of arriving at a very exact figure. I did get, however, a vast amount of

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[1] See page 108.

help and encouragement when trying to find out how much might be reasonably expected to do the work. A charge, chiefly of oxidized ores, does smelt with an amount of carbon equivalent to 0.7 Cal. per unit weight of charge and less. No allowance has been made in the above estimates, which are confessedly approximate (and only employed for the purpose of focusing the subject on a point of view I found of great help), for the heat of combination between FeO and SiO_2. There is the strongest reason for believing that this source of heat is important; so important that unless the silica used as flux be sufficiently acid, and free and not combined, the pyritic process will not, in our experience, go on unaided by coke, and then the concentration is unsatisfactory. In cases where the only flux available is one mainly consisting of combined silica, a compromise process employing a certain proportion of the ore in a roasted form must be adopted. I found great aid in my efforts by keeping in mind that there was a certain minimum heat production necessary per unit of charge to keep the charge melting and running, and that it was somewhere about 0.70 Cal. per unit weight, and also that there was a certain minimum weight to be smelted in a given time. The rate of chemical change had to be equal, therefore, to so many calories per second.

It is impossible for me to leave this subject without expressing my indebtedness to the work of Mr. Robt. Sticht, of Mt. Lyell, who is the practical pioneer of pyrite smelting, so far as I am concerned, because his work has been always an example and incentive to me. In carrying out this work at Keswick, I was ably assisted by Mr. A. S. Haskell and Mr. J. A. Balch, respectively superintendent and assistant superintendent of the smelter there.

LEWIS T. WRIGHT.

CONTRIBUTION BY CHARLES S. PALMER.

The Editor:

Sir—The information on pyrite smelting, accumulated from the columns of this Journal, is calculated to impress thoughtful practice everywhere. It represents a body of timely technical contribution almost unparalleled in the history of metallurgy. Indeed, no new process, involving so much that is valuable and untried, has ever reached approximate success so directly and quickly. Its principles are older than the range of any living metallurgist; its field is broader than the tuyere area of the blast furnace. All smelter men are watching with interest the mistakes, no less than the success of this method, for there is a pathology as well as a normal physiology of the pyrite blast; and the careful dissection of the frozen charge has shown much of the probable cause of strangulation. The collective comment of the papers comprise a composite monograph which is occasion for congratulation. But we should not infer from this that all the incidental questions are answered; so far from this is the real state of the case that one may say that not only are some of the problems not solved, but some are not even stated. The convenient analysis of that which has been accomplished would recognize two groups of topics; namely, first, the points which seem to have been established by general observation and experience; and, second, the points which are still undetermined. In the first group I would enumerate: The use of hot blast; the heat obtained from the burning of FeS; the need of much blast (oxygen), and the value of free silica. Mr. Lang has done a service in reiterating the three phases of pyrite smelting, namely, the heat-making, the slag-making and the matte-making; and it should be remembered that the blast furnace represents the seat of war of many conditions which find their equilibrium in the temperature and fluidity of the products escaping at the throat and at the spout; none of these conditions can be changed without affecting all the rest. One writer has commented on the wonderful way in which the blast furnace selects its own slag. It is true, everything in nature is wonderful; but beyond this it would be seven times more wonderful if the furnace did not under any conditions select its own slag and matte in accordance with the energy (heat) and the flux furnished. This point of interlocked equilibrium of all parts and functions must be constantly borne in mind—the furnace is a physiological unit.

It is a fortunate matter that we should have learned the value of hot blast. The testimony seems conclusive that it is of advantage in saving coke and in forcing silica into the slag, especially on ore lean in pyrite. How the blast shall be heated is a matter of local economy.

The heat is produced from the burning of the sulphur and also of the iron of the pyrite; for, while sulphur alone furnishes but little heat, yet it may be just enough to raise the charge above the danger limit. The comparatively small quantity of free sulphur in flue-dust would indicate that most of it is consumed in the furnace to sulphur dioxide.

The need of free silica in absorbing the oxidized iron is clearly emphasized. The lesson that this should be auriferous quartz, or the like, has been well shown in lining for converter practice.

Among the second class—the unanswered questions—may be enumerated: The copper minimum in the matte; the heat of the reactions:

$$2FeS_2 + 5O_2 = 2FeO + 4SO_2,$$
$$2FeO + SiO_2 = Fe_2SiO_4,$$
$$CaO + SiO_2 = CaSiO_3;$$

and the condition of zinc oxide and aluminum oxide in the slag, etc. The question as to the amount of copper required to hold the precious metals in the matte, and the correlated question as to the influence of infinitesimal quantities of tellurium, bismuth, arsenic, antimony, and other similar value-thieves, as quoted by Peters from Pearce, are curiously answered by most of the catechists. One-half per cent is a very small piece of soap for so large a hand. There is more yet to come from this problem. As to the thermo-chemistry of the equation, as indicated, there is no excuse in this age of science, and endowment kings that we do not have the exact knowledge of all the more important metallurgical thermal constants. That the heat-toning of the reaction, represented by each equation herewith given, is considerable cannot be doubted.

The question as to the condition of zinc oxide in the slag has several aspects. Lloyd says that the zinc slag may melt at as low a temperature as the corresponding iron slag, though it will be less fluid. Peters mentions a slag that was fluid, though high in zinc and earths and low in iron; figuring the alumina as an acid, the slag was a bi-silicate. Others speak of zinc compounds as floating in the slag much as snow floats in ice-water. This might easily be a case of mass suspension. In fluid homogenous zinc slags the zinc may be a silicate, an alumino-silicate, an aluminate, or a mutual solution of these.

As to the rule of alumina, I have discussed this from one aspect in another issue of this Journal (March 2, 1905, page 420). But there is another side rich in possibility. It seems a common custom (note the experience of Bretherton) to figure alumina as two-thirds of the silica; but this needs definition. Alumina may react as the met-aluminate, or as the normal aluminate; and it may be compared with metasilicic acid or normal silicic acid. In these cases it would function respectively as 2 to 3, 2 to 5, 1 to 1, and 4 to 5. But in view of the great tendency to the formation of alumino-silicates on the one hand, and on the other the absence of calcium aluminates or met-aluminates as such (though ferrous met-aluminates are known in ismorphous iron mixture), there is another view that is broader and more probable, and to which I will revert later.

Magnetite seems to be a standing nuisance. It deserves a separate study; meanwhile we should remember that it is not a simple oxide, but a 'salt oxide'—a salt compound of a lower and a higher oxide of iron.

But the richness of this bill of fare must not make one a metallurgical *gourmand*. If one is challenged to produce some distinct product of pyrite-smelting theory and practice, he need only refer to the actual concentration of matte. This is practical in the regular copper furnace with high fuel by those who know how to do it.

In discussion, fact is the basis; but it will be well to awake from the lethargy that comes of too close scrutiny of detail. Let us look at the question fairly, as we ask: What is the broad teaching of this phase of blast-furnace practice? There is a growing tendency among matte-smelters to reject the theory of any single and constant slag content. Such may occur as exceptional. But, on the other hand, we have in well-attested practice the gradual swinging from high iron and low lime to high lime and low iron. Where is the limit here? It looks as though we are dealing with a mixed solution and intersolution of several accommodatingly soluble media. The fusibility of slags is only partly worked out; but beyond that, the question of the mutual solubility of several slags calls for measurement. This does not ignore the probable calcium meta-silicate (bi-silicate) nor the ferrous ortho-silicate (uni-silicate); but it does put the question on a broader basis, and it also propounds certain comparatively simple laboratory experiments on mutual solubility which it is not to the credit of metallurgy to neglect longer. Thus, if a molten flux contains two fusible, non-ionizable and mutually soluble salts (which we will call A and B), there are four units to consider, namely: A and B themselves, the solution of A in B and the solution of B in A. But if there is ionization, then new possibilities and, presumably, new limitations would appear, which must be investigated on the basis of the well-known phase-rule of Gibbs.

The essays on 'Pyrite Smelting,' when published together, will find a place on the desk of every working metallurgist; they comprise a storehouse of specialized fact and practice not excelled in this generation.

CHARLES S. PALMER.

New York, March 14, 1905.

CONTRIBUTION BY HIRAM W. HIXON.

The Editor:

Sir—Pyrite smelting has been the subject of much discussion in *The Engineering and Mining Journal*, and under certain peculiar conditions, which are purely local, it is entitled to be called a process, as distinguished from ordinary blast-furnace smelting.

All blast-furnace smelting, even lead smelting with the use of 14% of coke, will eliminate some sulphur from the charge, which passes off with the flue gases, and some iron must be liberated and go into the slag as a result. Not more than 70% of the sulphur on a lead charge can be accounted for by the matte produced under conditions when a heavy reducing action is indicated by the production of a small amount of speiss along with the lead-copper matte. This indicates that, even in the presence of so much as 14% of coke, sulphides do burn, and, once combined with oxygen, they are separated by the reducing action of the fuel, or the gases of partial combustion.

The proper conditions for the greatest amount of *reduction*, or the reverse, the greatest amount of *oxidation*, in a blast-furnace are so much affected by the method of feeding and the arrangement of the particles in the furnace that, without taking these conditions into consideration along with the volume of blast per minute, there is no certainty what kind of a slag will result. For example, suppose a furnace is running on a charge producing 30% copper matte, and making a slag containing 30% SiO_2 and 0.3% Cu. The percentage of fuel may be anything from, say, 3 to 12, depending upon the amount of pyrite on the charge, and also the amount of copper and the degree of roast, if any. Now, suppose we speed up the blowers or put on more blast in any suitable way; we would notice the following results: (1) The tonnage smelted will be increased, (2) the matte will contain a higher percentage of copper, and (3) the slag will contain more iron, less silica and more copper, not in exact proportion as the blast is increased, but governed by it. All of these conditions are changed by simply increasing the blast volume per minute.

Pressure does not mean anything except resistance, and it may be caused by charge burden or slag in the tuyeres, or it may mean that the tuyeres are too large or too small. After these conditions are adjusted to an equilibrium, suppose we change the method of feeding from an even distribution of fine and coarse particles to placing the coarse pieces near the wall and the fine in the center of the shaft. The result will be a further oxidation of sulphides, resulting in a further increase of iron in the slag and a corresponding decrease of silica. The percentage of copper will increase in both the slag and matte, and the amount of matte produced will decrease. All of these changes of composition of the furnace products will be affected, *not* by any change of charge or fuel, but by increasing the volume of blast and altering the arrangement of the particles composing the charge. The furnace will not continue to run for long without crusts forming on the walls, and this may increase at one end or on one side and diminish or entirely disappear on the other, for the passage of the furnace gases is so restricted that it results in blowholes; and these affect the oxidation to such an extent that it frequently happens that two furnaces running on the same charge will produce slags varying 3 or 4% SiO_2 or Fe, and mattes varying as much as 10% in copper content.

In this manner we may vary the action of the furnace, and if we continue to increase the blast we finally arrive at a point where the copper losses in the slag, together with the precious metals or others if there be any, would be too great to allow a further concentration on a commercial basis. The slag losses increase much faster than the grade of the matte, and while we may have increased the copper in the matte from 30 to 40% the slag will have increased from 0.3 to 0.6% or more. All this goes to show that the pyrite is not the only thing that is oxidized; copper and nickel and all the associated metals in the charge get their share of oxidation, and that is opposed to the collecting action of the matte. Slags formed under such conditions resemble slags made in the copper converter; they are foul and contain a considerable percentage of peroxidized iron, which renders them magnetic. They will not decompose with acids without fusion with an alkali carbonate.

The remarkable effect of reversing the distribution of the pieces composing the charge is shown in lead smelting, where it has been demonstrated that it is absolutely necessary to feed the fine to the walls and the coarse to the center in order to get the reduction necessary to prevent excessive oxidation and loss. About the year 1883 the flat-top lead furnace was introduced in nearly all the Colorado smelters, and at first they were fed through a narrow thimble; but in every case this had to be abandoned because it had the effect of placing the entire charge in the center until it sank below the end of the thimble, releasing the coarse pieces. which rolled to the sides and ends of the shaft, while the fine all settled down in the center. The blast naturally passed up through the coarse pieces and caused over-fire behind the thimble, the reduction stopped, and almost immediately the bullion production decreased to an alarming extent. These were 'pyrite smelting' conditions applied to lead smelting, and the reverse of what should have been.

The feeding was then done by shoveling the coarse to the center and the fine to the walls, forcing the carbon monoxide formed at the tuyeres to penetrate the charge to the center and act as a reducing agent all the way up through the charge. Later, when automatic feeding from a car was introduced, it was found that the charge had to be dumped so that it would be highest against the walls, and the large pieces could roll to the center, preserving the conditions as described. If at any time a number of charges were dumped in the center the conditions would become reversed, over-fire would start, and bullion production would stop or decrease.

The bell-and-hopper feed of the iron blast-furnace is admirably suited for a round furnace, as it forces the stock to the outside and leaves the center lower, so that the pieces arrange themselves in the order required for reduction; but this method of feed is not suited for rectangular furnaces for lead smelting, and is not required for copper smelting, as was demonstrated at Aguas Calientes, Mexico, in 1896.

In pyrite smelting, dumping the charge-cars from the side results in the coarse pieces going to the opposite side and the fine landing in the center, especially if the surface of the charge is 5 or 6 ft. below the level of the feed-door. The result is that there is a ring of fire and escaping gases around the edges of the charge, and the center is perfectly dead. I saw this strikingly illustrated about two years ago at the works of the Tennessee Copper Company, where a man fell into a furnace smelting 600 tons per day. He picked himself up and walked around on the dead center and tried to climb out, until the blast was shut off and a ladder lowered to him, when he climbed out within reach of men at the feed-door. The only injuries he sustained were to his hands and face and parts of his body where his clothing had burned. It was thought that his lungs would be injured but this was not the case. Afterward it was found, by putting in pieces of paper, that there was a down draft of air in the center induced by the blast around the sides. As the carbon monoxide did not penetrate to the center of the charge it could not exert any reducing action, and therefore the conditions for oxidation and concentration of matte were present and active, resulting in making a 40% matte out of the 3% ore.

Under all such conditions of high concentration by oxidation, the resulting slags are much higher than they would be if the conditions of blast, fuel and feeding were reversed to produce the conditions necessary for reduction. The commercial aspect of the case is, however, governed by local conditions, and depends entirely upon which method gives the greatest net return from the ore. Some men argue that it pays better to waste metals and save coke than the reverse. However this may be, I note where one company turned a surplus into a deficit by following the plan. It would appear that a dollar's worth of coke saved is not worth as much as two dollars' worth of metals wasted.

The two-stage operation of first making a low-grade matte 12 to 20%, re-smelting and concentrating this to 35 or 40%, and re-smelting the slag from the matte concentration when it is high enough to justify it, seems to be the best plan devised. Trying to make a converting grade in one operation results in too high slag losses and short furnace campaigns. At La Aurora, Mexico, a middle course was taken. A 6% ore containing 11% Zn was part roasted in stalls and piles and smelted with equal part of green 'fine' and 6.5% coke. The resulting matte, about 35%, was converted and the converter-slag added to the charge. The blast-slag contained 10% Zn and 0.6% Cu. The high copper content of slag was due to high blast and zinc, otherwise the zinc gave no trouble. An ore containing zinc will make a higher concentration than if the zinc were replaced by iron.

The degree of concentration is all dependent on the blast, the fuel, feeding, and the amount of iron in the ore. An ore containing a small percentage of iron will make a higher concentration—that is, more tons of ore into one ton of matte—than an ore containing a higher percentage of iron. Hot blast may be a benefit, but it is a luxury that at many plants they have found they can get along without.

It is a difficult point to define, but I should say that pyrite smelting does not begin until the slags show the characteristic peroxidation of the iron as evidenced by chilled samples refusing to decompose without fusion. If a chilled sample will not give white silica without fusion, you may be quite sure that it has been produced under conditions of oxidation characteristic of 'pyrite smelting.'

The capacity of any plant smelting raw sulphides with, say, 3% coke and high blast is less than half its capacity of pile-roasted ore with 10% coke when producing the same grade of matte (35% to 40%).

HIRAM W. HIXON.

Victoria Mines, Ont., April 4, 1905.

CONTRIBUTION BY S. E. BRETHERTON.

The Editor:

Sir—Having just read the favorable comments on pyrite smelting and the use of hot blast, by your Mr. Charles S. Palmer, in *The Engineering and Mining Journal* of March 30, I thought it an opportune time to add a little more, through your valued columns, to this important subject. In regard to the amount of copper necessary to collect the gold and silver, this depends on the amount of zinc with which the metallurgist has to contend: I know from experience that, with considerable zinc, 5% Cu in the charge gives much cleaner slag than only 0.5% Cu, even when making a much richer matte in Au and Ag, and smelting the same character of ore.

Alumina, for convenience, can be classed as silica, and I prefer to limit its use to 12% in the slag. To be exact, when figuring a bi-silicate slag for copper smelting, the formula would be: $2Al_2O_3+3FeO=48.8\%\ Al_2O_3$, that is, 2O in the acid to 1O in the base, classing the Al_2O_3 as an acid, which corresponds to the formula: $FeO+SiO_2$ slag $= 45.6\%\ SiO_2$; or $CaO+SiO_2$ slag $= 51.9\%\ SiO_2$; as the proportion of Al_2O_3 in the slag increases, the amount of SiO_2 should be reduced. To allow the SiO_2 and Al_2O_3 together to get above the bi-silicate limit will eventually cause trouble.

A silicious slag for copper smelting seems to be the most suitable for concentration and clean work. Of course, with plenty of Cu and good matte-fall, the danger of loss is lessened; first, by having plenty of Cu and matte to gather value; and, second, by the fact that there is a smaller proportion of slag to act as a means to carry off value.

An acid slag, especially when low in Fe, requires more heat than a basic slag, and for this reason; it is necessary to use 4 or 5% coke in the charge in hot-blast smelting, to avoid accidents, especially when there is danger of trouble with belting, pulleys, etc., as in the case of a new, small plant attempting to do all its work with one engine and machinery generally poorly arranged.

The heat applied to the furnace from the outside, by means of hot blast, is more effective for economy, rapid smelting, and other advantages; but the furnace cannot be banked for any great length of time without preparing for it by first adding several charges with extra coke. These extra charges should consist mostly of slag or oxidized ore, on account of the extra coke (reducing agent) added.

The metallurgist of a custom copper-smelter often finds himself 'between the devil and the deep sea'; he has no iron except as a sulphide, and that in limited amount, which compels him to make a silicious slag, high in lime. To avoid trouble with this character of slag (and, perhaps, accidents with poor machinery) he should use 5 or 6% coke on the charge—nearly one-half the normal amount used with cold blast. On the other hand, the use of too much coke prevents the necessary concentration to force the Fe (which he must have) into the slag, and the elimination of impurities, such as Zn, Pb, As, etc., which should not be allowed to enter the matte.

The metallurgist can be saved much trouble and worry by having the capital necessary to enable him to wait for proper ore mixtures, and to repair or set up proper machinery so as to avoid accidents; indeed, important parts should be duplicated.

S. E. Bretherton.

Val Verde, Ariz., April 10, 1905.

CONTRIBUTION BY LEWIS T. WRIGHT.

The Editor:

Sir—Seven years ago the ideal of the metallurgist working with sulphide copper ore was to roast it in mechanical roasting furnaces, after such preliminary crushing as was necessary, to press the fine calcines into briquettes and smelt these with coke. To-day we are smelting these sulphide ores, coarse or fine as they come from the mine, with no other combustible than the iron and sulphur they may contain.

The chemistry of pyrite smelting is the chemistry of iron and sulphur. It is the oxidation of this iron and sulphur by the air pumped in the base of the furnace that provides heat required to maintain the temperature of the furnace at that degree which is necessary for the digestive process of pyrite smelting.

The human body is a blast-furnace and its temperature must be maintained at a living grade by the oxidizing action of the air pumped in and out by the lungs; because, if the production of heat fall below the amount required to compensate for loss of heat by radiation and other sources of loss, its temperature will fall below the initial temperature of the chemical reactions; production of heat will cease and the body will 'freeze up.'

An interesting study in pyrite smelting (and by pyrite smelting I mean the smelting of crude sulphide without carbon) is the amount of air required. This branch of the subject has been too much neglected. The metallurgist in charge of the pyrite blast-furnace must stand literally with one eye on the blower, and the other on the furnace with its charges, its slag and matte.

The blower turns should be as closely watched and recorded as any other of the incidents of the operation. The air is the breath of life of the furnace.

The heat losses are considerable, and a certain temperature must be maintained to allow the slag and matte not only to flow out of the furnace, but in many cases to be held a little while in a molten state. There will be, for any furnace, a certain minimum rate of heat production required to maintain this minimum degree of temperature, and this in pyrite smelting demands a certain rate of oxidation and a certain supply of air.

The rate of heat production is expressed by two factors, the weight of charge smelted, and the proportion of the oxidizable materials in the charge (the iron and sulphur) oxidized. The former is the rate of running, or the speed of the furnace, and is expressed in tons per hour or day, or number of charges per hour or tons smelted per square foot of furnace area per 24 hours. The latter factor is the 'concentration,' the 'sulphur elimination,' or the 'oxidation.' The product of the two factors is the rate of heat production, and this can also be conveniently expressed in terms of weight or volume of air or oxygen used in a given time.

In the human economy the air which is inspired with the oxygen content of 20.8% is expired with 16% of oxygen. The efficiency of air is thus in the human blast-furnace not very high, being only about 25%. I have been at some pains to determine its efficiency as an oxidizing agent in our pyrite-smelting practice, and have found it to be as high as 70 per cent.

It is now time to discuss the amount of oxygen or air required by pyrite in its oxidation. The pyrite in this process is the fuel. When FeS_2 is heated in a neutral atmosphere, one-third of its sulphur volatilizes, thus $3(FeS_2)=Fe_3S_4+S_2$. It is said that, under certain circumstances of limited oxidation, one-half of the sulphur is recovered as such. I have not yet had an opportunity of confirming this statement, the belief in which must be the origin of the metallurgist's 'volatile atom.' In pyrite work we do see signs of free sulphur in the furnace gases.

The amount of air required for the complete oxidation of pyrite in smelting is $FeS_2+5O=FeO+2SO_2$, or 120 parts by weight of pyrite require 80 parts by weight of oxygen, or 1 lb. FeS_2 requires 2-3 lb. oxygen.

A pyrite furnace, with area at tuyere level of 43.75 sq. ft. and running well, smelts its charge at the rate of 11 tons of total burden per sq. ft. of furnace area per 24 hours, or 20.05 tons of charge per hour, or 668.3 lb. per min. The amount of air used is most conveniently dealt with on the basis of the minute. I take as an example a case from actual practice. The charge consists of pyrite containing some chalcopyrite, the requisite fluxes, the converter slag resulting from the second operation, and flue dust in proportion as normally produced. The pyrite operation is ideal and there is no coke on the charge. The FeS_2 charged (calculating all the iron in the ore as FeS_2) is at the rate of 343 lb. per minute, and is close to 50% of the total burden. The composition, the texture of the charge, and the air supply are properly adjusted and the oxidation is high in consequence. The degree of oxidation of the FeS_2 is 90%. Thus, $343 \times 0.9 = 308.7$ lb. of FeS_2 being oxidized every minute. The furnace gases contain 12% of SO_2 immediately on top of the charge. Each pound of FeS_2 requires, for complete combustion, 2-3 lb. oxygen.

A pound of oxygen is contained in 55 cu. ft. of air at 72°F. The efficiency of the air is 70%; therefore, 78.6 cu. ft. of air are required to furnish 1 lb. of oxygen. On this basis, 52.4 cu. ft. of air will be required for 1 lb. of FeS_2 oxidized; 308.7×52.4, or 16.176 cu. ft. of air per min., are therefore required by this furnace. The amount of air actually measured by the displacement of the blowers was, in the instance I cite, found to be 16,985 cu. ft. per min. In this case, 0.21 of iron and 0.24 of sulphur were oxidized per unit of charge. This expresses the degree of oxidation. The rate of oxidation was 308.7 lb. FeS_2 per minute.

It is stated that 1 kilogram of FeS2 when oxidized to Fe_2O_3 gives 2.253 calories. Adjusting this value to oxidation to FeO, which is the case in making slag, we have as the heat of combustion of pure pyrite 1.976 calories.

In the above-cited case the pyrite of the charge was yielding heat at the rate of 0.89 calorie per unit weight of charge. To produce this amount of heat from coke containing 93% of carbon, 11¾% would have been required. The furnace at this period was running very fast, because it was very hot; but the heat, with the exception of the small amount afforded by the heated blast, was being produced from the pyrite.

A pyrite furnace is doing good work when it is using per min., for each sq. ft. of area, as many cu. ft. of air as there are days in the year, and oxidizing as many pounds weight of FeS_2 per minute as there are days in the week; and when the furnace gases on the top of the charge contain as many per cents of sulphur dioxide as there are months in the year. This, though it may sound cabalistic, is the true *memoria technica* of the pyrite smelter.

Lewis T. Wright.

Keswick, Cal., May 1, 1905.

CONTRIBUTION BY P. L. MARSTON.

The Editor:

Sir—I have read with much interest various communications on sulphide smelting, and I wish to give my experiences in confirmation of some of the views already expressed.

In re-heated blast I have found that it always gives a decided advantage over cold-blast practice, even where the rise in temperature is a relatively small one. In one instance 60° made a very noticeable difference both in the speed and control of the furnace. At the plant of the Ouray Smelting Company the blast was heated by the waste gases from the furnace. A number of horizontal convolutions of the blast-pipe were placed in the path of the gases from the down-take and an average temperature of 130° F. was maintained. Just before charging the furnace the temperature would reach 150° to 160°, while shortly afterward the temperature would drop to 100°. The pre-heated blast is of importance in promoting speed in the furnace; but, from my observations, the cold blast gives a higher rate of concentration. Too great speed may obviously cause low concentration, from the fact that it requires a certain length of time to oxidize the sulphides; and, with a heavier blast to promote speed, the result is a hotter top and a low ore-column, since the sulphides will melt and pass to the crucible before they have had time sufficient to be acted on by the blast. Moreover, the regulation and control of the furnace is rendered easier when a heated blast is used. Irregularities of feeding or of charged material do not affect its capacity or its equilibrium so easily as with cold-blast practice, and slag of a higher silica content can be made and handled with greater ease.

The minimum of fuel depends on several factors; among these, cost, quality and speed of furnace come first. At the plant referred to we used 8% of the ore on the first run, speed being a more important requisite than high concentration. For concentration of the matte 2.5 to 3% of coke was used, except for the first 10 or 15 charges, when no fuel at all was used, as the heat from the first run kept things good and hot for some time. By using but 2.5% fuel and a large amount of silicious ore it gave us a relatively slow-driving charge, and also allowed of a concentration of 3 into 1; whereas with 5% coke the concentration would be about 3 into 2, and sometimes even less, the matte merely melting and running through. This first matte averaged 8% copper.

Much also depends on the care used by the crew below as to how the furnace will work on a concentration run. If the jacket-water is discharged at the boiling point, all the time, the furnace will act much better, do better all round work and be far easier to handle.

Blast pressure depends altogether on conditions prevailing at the time. Here, with rather fine charge and an ore-column of 8 ft., we used a pressure of 24 oz. This gives a hot top and causes much flue-dust to be carried over. Of the latter about 4½ tons were made each 24 hours, and the gold and silver contents were highest near the furnace, and lowest at the stack, 100 ft. from the down-take. None of the fume was ever collected, as no means were at hand to do this.

The loss of sulphur from the charge averaged 75%. The furnace at this plant is 36 by 106 in., with a continuous-flow, blast-trapping spout. With a fairly fine charge and blast pressure as given, it would smelt 140 tons of charge per day, and could have handled 20 tons more if adequate settling arrangements had been furnished.

Slags could not be settled clean if the forehearth was crowded too fast.

The ordinary blast-trapping spout for continuous slag flow, such as was used here, is, generally speaking, a troublesome device and requires too much care and attention. A plain tap-jacket, of good design and construction, will do better work and give better satisfaction in the long run.

The ore charge as given to the furnace would average 0.8% Cu, and this would give clean slags when the speed of the furnace was kept down to reasonable limits. A slag of high-silica content would clean better than one of high-iron content, under the same conditions of charge, blast pressure, speed, etc. With 0.5% Cu on the charge, clean slags can be made, unless much zinc be present. With a high-zinc slag to be made, the copper on the charge should be much higher; 3% at least, and 5% is even better. The concentration made here on the first run was 7 or 8 to 1; and, as I have stated, the concentration of the first matte, 3 to 1. Any ores are suitable for pyritic or semi-pyritic work, except those the lead content of which is above 5%. As low as 15% of FeS_2 or its equivalent can be used to good advantage in this process; but it is better to use 25% if the full effect is to be derived from the burning of the iron and sulphur. Where a slag high in iron is made, a moderate amount of zinc has but little, if any, effect on the fluidity of the slag or its driving qualities, unless much alumina is present. Considerable of the zinc is volatilized, practically all of the lead, too, and this phase causes loss of both gold and silver; the loss is greater in the case of silver, however. The slag made at the Ouray plant averaged as follows: FeO, 25.1; CaO, 15.2; BaO, 3.1; ZnO, 5.0; SiO_2, 45.1; MgO,Al_2O_3 and alkalies to 100%. The average values in the slags were: Au, 0.13 oz.; Ag, 1.6 oz.; Cu always less than 0.1%. These losses, with proper facilities for settling, would be reduced by half.

Dr. Carpenter has spoken of the apparently automatic action of the furnace, under certain conditions. A good example of this selecting process came to my notice here. Our iron sulphide was of poor quality and variable; one noticeably poor lot was put on without my being able to sample it. At 2 A. M. the furnace stopped making matte, and none was made for eight hours, although it had been making 800 lb. per hour, and did so again as soon as it regained its normal condition. An analysis of the slag, made during this interruption, gave: FeO, 19.4; MnO, 0.9; CaO, 12.1; Zno, 5.8; SiO_2, 56.2; Al_2O_3, 1.4. All the iron in the charge was used for the slag, leaving none for the matte. The assay of this slag showed retained values as follows: Au, 0.008 oz.; Ag. 1.8 oz.; Cu, trace.

P. L. MARSTON.

Ouray, Colo., May 24, 1905.

Contribution from J. Parke Channing.

The Editor:

I realize that it is impossible ever to be able to say the last word on any technical subject, and this must be particularly so in regard to metallurgical operations. At the works of the Tennessee Copper Company we have made many experiments, and have much improved our practice in pyrite smelting, but we expect to do better still. Not until our enlarged plant is completed do we hope to get the full benefits of pyrite smelting, and, therefore, all that I can do for the profession is to report progress.

The month of May gave us the best results which we have attained since our change, but we still can see many points in which improvement is sure to come; in other words, we look forward to a somewhat ideal condition in which, instead of every kind of accident or setback every month, we will have only one kind each month. In May we smelted about 19,000 tons of ore in our three furnaces, together with 1,000 tons of custom matte. Including converter-slag and blast-furnace products and first matte re-smelted, we handled a little over 30,000 tons of material per month, not including coke. The total coke burned for this 30,000 tons of material was about 1,500 tons, which is considerably higher than we should consume, partially on account of using some fine material from coke stock, and also on account of the extra coke used in smelting off crusts.

Permit me to re-state briefly the principal features of the present Tennessee plant. It consist of three blast-furnaces, each 56 by 180 in. at tuyeres, the total height from tuyere to feed-floor being 18 ft. Of this, 14 feet are available for depth of charge, the top 4 feet being simply added so as to get proper distribution of the charge. Each furnace has 26 four-inch tuyeres, and the blast is furnished from piston-blowing engines. This blast-furnace is exactly the same one that was formerly used for smelting roasted ore, and we have not found the necessity of any marked change in pyrite smelting. Perhaps, here may be a good place to mention that the only change which it was found necessary to make in the furnace was the elimination of the old Boston and Montana type of cast-iron, brick-lined crucible, and its replacement by water jackets and cast-iron soleplate; so that the present furnace is practically water-jacketed all the way down.

Two furnaces are used for treating ore that comes from the mines; the third furnace concentrates the matte from these two furnaces.

The Tennessee Copper Company has three mines, of which, at present, only two are being wrought—the Burra Burra and the London. The Polk County mine will not be put in commission again until the beginning of next year, when the enlarged plant will be in operation.

The ore is a massive sulphide, consisting of nearly equal parts of pyrite and pyrrhotite, together with a certain proportion of chalcopyrite and silicious minerals. Below are approximate analyses of the Burra Burra and London ores:

	Burra Burra.	London.
Copper	2.2%	3.0%
Sulphur	30.0 "	21.0 "
Iron	37.5 "	31.0 "
Zinc	2.0 "	0.8 "
Lime	6.2 "	6.1 "
Magnesia	1.9 "	2.5 "
Alumina	3.9 "	4.4 "
Silica	10.3 "	26.3 "

The ore, as it comes from the mine, is crushed coarsely, so that the maximum size does not exceed six inches. It is then sent to the smelter bins, where it is fed into the charge-cars. The ore from each mine is kept separate on account of the difference in silica content, the ordinary charges being as follows:

	Burra Burra.	London.
Ore	4,000 lb.	5,000 lb.
Quartz	1,000 "	400 "
Coke	120 "	150 "

These proportions are not always adhered to, the amount of quartz and the amount of coke being varied to suit conditions of the furnace and the grade of the matte. In loading the charge-cars, the sulphide ore is placed underneath the quartz; a train generally consists of four cars of ore charge and one car containing either 480 or 600 lb. coke. The coke is not dumped into the furnace, but is thrown on the opposite side of the track upon the feed-floor. The charge-cars dump into the furnace, two on each side. Practice at our plant (like that at Mt. Lyell) shows that it is advisable to get the quartz near the center of the furnace, and this is best accomplished by putting it on top of the ore charge. On the contrary, it is found advisable to keep the coke as near the edges and corners of the furnace as possible, and it is fed by hand. As far as we are able to judge, in our charge the coke performs no distinct metallurgical function, but seems a necessary evil which must be endured in order to prevent crusts adhering to the jackets. This crusting of the furnace is the *béte noir* of pyrite smelting, and it is only by careful manipulation that it can be avoided. The furnace must be carefully watched, and, on the first sign of approaching crust, either it is removed with long bars, or else large chunks of sulphide ore are thrown against this crust to prevent its increasing in size.

If the coke is dumped into the furnace and not fed against the jackets, it will also tend toward increasing the crust. We have found repeatedly that a one-night's run (where the furnace crew was short and the coke could not be properly shoveled in) would suffice to crust the furnace so badly that it would take three or four days of careful manipulation to bring it around into good shape. We had a great deal of trouble in this respect during the last six months, not only on account of shortage of labor, but also by reason of the fact that we lost the use of the 'loop' track around the furnaces during the enlargement of the smelter building.

In the old days of roasted-ore smelting, the average depth of charge above the tuyeres was probably not more than 8 or 9 feet. In pyrite smelting, if the furnace is running well, the depth of charge will vary from 12 to 14 ft., and under these conditions the furnace runs better and gives a better elimination of sulphur than with a less depth of charge.

Under normal conditions, a furnace will treat 375 tons of sulphide ore per day, and, at that rate, should be in action about 27 days per month; that is, each furnace may be expected to be down for one week every two months, thus giving the average ore per furnace per month of about 10,000 tons. I believe in time we shall be able to exceed this, as we have frequently put through over 400 tons in 24 hours, not including quartz flux.

When running at this rate of 375 tons (per day of 24 hours) our engines will make about 68 to 75 rev. per min., or an average of about 17,000 cu. ft. of free air per minute. These figures are accurate and allow for losses due to slip or leakage. These figures will be particularly interesting, as we can determine the slip and leakage with our piston blower—a correction rather difficult to make with a rotary blower. This volume of air is not much different from the amount used when we were smelting roasted ore, only under former conditions the furnace would put through a larger tonnage. We have repeatedly tried using a larger volume of blast on the furnace, but not with great success, the tendency being to crust the furnace. What we may do in the future is hard to tell.

The furnace is arranged as usual, with a continuous overflow, the resulting matte and slag going into a 16-ft. circular 'settler,' which we now line with chrome brick. The slag overflows into cars and is carried to the dump. The first matte is intermittently tapped into other slag-cars, moved to a slightly inclined yard in front of the furnace building and there poured into flue-dust beds. After it has solidified and cooled, it is broken by hand (as near 6-in. pieces as possible), loaded into railroad-cars and brought up to the smelter-bins.

The second operation consists in smelting this matte, together with quartz, converter-slag and blast-furnace cleaning, so as to produce a matte of a grade sufficiently high to convert. The same kind of furnace is used for this second operation, but the furnace is changed occasionally, as it is found that a matte-charge will clean out a crusted ore-furnace, and conversely an ore-charge will clear out a crusted matte-furnace.

The matte-concentrating furnace does not run as rapidly as the green-ore furnace, the tonnage per week varying from 800 to 1,300 ton of matte, depending a great deal upon the grade of the matte and the condition of the furnace. In the early days of our pyrite smelting we found no great difficulty in running an old-type furnace on a green-ore charge; but, whenever we attempted a concentration charge, in less than 24 hours the crucible became so hot that it leaked in every direction, and it became impossible to keep the matte in it. One by one the crucibles were changed by the substitution of 'baby jackets,' the first furnace adapted being put on concentration charge (such as I will show herewith), running from December 9 to 15, 1904. I also give the average of the matte produced on the first and second operation, respectively, during the month of May, that on the first operation being about 12% and that on the second about 43%. This grade of matte on the second operation is somewhat complicated by the addition to this charge of 45% custom matte, which is simply re-melted in this furnace, and then converted. It tends to mask the real operation, though we assume that it has no particular effect upon the concentration.

MATTE ANALYSES.

	First.	Second.
Copper	11.9%	42.8%
Sulphur	25.6 "	24.6 "
Iron	54.0 "	29.2 "
Zinc	1.7 "	1.0 "

It will be remembered that the slag which we produced in ordinary roasted-ore smelting ran about 33% to 35% silica. It is not yet found possible to do this in pyrite smelting. The fact seems to be, as has been before stated, that, with a given ore and a given furnace, the combination makes its own type of slag, from which it is impossible to make any radical changes. Therefore we have, so far, arrived at the fact that the slag at our first operation will run about 40% silica. It makes no difference whether we increase, or diminish, the amount of quartz on the charge; the slag will remain at about 40% silica, and the only change will be an increase, or decrease, in the grade of the matte. In fact, the method of running the furnace is to vary the quartz charge so as to keep a constant grade of matte. This grade varies from 9% to 16%, and, under present conditions, we consider a 12% matte the most economical. While making this grade of matte, the slag will run from 0.18% to 0.27% copper. Herewith I give analyses of our slag on both the first and second operations:

SLAG ANALYSES.

	First.	Second.
Copper	0.2%	0.6%
Sulphur	0.7 "	1.3 "
Iron oxide	42.5 "	52.6 "
Zinc oxide	1.5 "	1.1 "
Lime	7.5 "	2.6 "
Magnesia	2.0 "	0.7 "
Alumina	5.5 "	4.2 "
Silica	40.0 "	36.5 "

It will be observed, from these analyses, that, on the first operation, we make about a 'quarter' slag, the factor of which is 0.29. We notice, however, that the silica is much higher than the ordinary 'quarter' slag such as a lead furnace will make; but, as I have said before, it is absolutely impossible for us to keep this silica down. I have no doubt that it is because of this 'quarter' slag that our copper runs so low on the first operation. A straight charge of Burra Burra ore will give a better ratio than a London charge, or even a mixture of the two. We did make an experimental run for about two days, adding some barren limestone to the charge so as to make the formula an exact 'quarter' slag, but the campaign was so short that we were not able to see any improvement in the slag-losses.

In the first concentration, carried out in December, 1904, we took low-grade matte from our first green-ore smelting, which probably did not run higher than 9% copper, and in one operation brought it up to 50%. Under these conditions we made practically an iron-silica slag, with a specific gravity of about 3.8. Under these conditions the slag ran high in copper. The results of runs for two different weeks in December, 1904, are given herewith, simply to show what can be expected with concentration of this kind. In this operation we found no difficulty in making matte of any grade desired, simply by varying the amount of quartz; and, as we had a large accumulation of low-grade matte, we thought it better policy to concentrate highly so as to get a maximum output from the converters. We soon found that these concentration losses could be reduced by adding a certain amount of sulphide ore to the charge. This not only decreased the rate of concentration, but also improved the composition of the slag by adding to it a certain proportion of other bases. This improvement in composition can be seen by looking at the complete analyses of slag, as herewith.

For comparison, I give the results of three different week-runs on concentration, the first two being straight, and the last one showing about the type which we now follow:

TYPICAL CONCENTRATION RUNS.

	Dec. 9 to 15. 1904. Tons.	Dec. 16 to 22. 1904. Tons.	Jan. 2 to 8. 1905. Tons.
Sulphide ore	....	....	279
Converter slag	130	210	162
Blast-furnace slag	....	....	320
Quartz flux	562	527	307
Custom matte, 45%	....	....	252
First matte, 10% to 12%	1,296	1,285	808
Total	1,988	2,022	2,118
	Tons.	Tons.	Tons.
Coke used	89	71	151
Coke percentage	4.5	3.5	7.1
Slag Analyses:			
Copper	1.01%	1.03%	0.65%
Iron oxide	58.6%	58.4%	54.2%
Silica	33.9%	34.2%	38.8%
Matte Produced:			
Copper	48.2%	52.5%	44.2%

We still have on hand an accumulation of custom matte, which we are trying to clean up; and, as soon as this is out of the way, it is our intention to add barren limestone to this concentration charge so as to make a 'quarter' slag and determine, experimentally, whether we can concentrate a 13% matte to a 40% matte with comparatively low slag-losses. As far as I am aware, this particular kind of work has not been carried out anywhere else in the United States, and it will be interesting to see what results are achieved. In certain of the Salt Lake valley smelters a concentration of low-grade matte is made in which the silicious material added is usually an ore containing more or less iron, and also considerable fine; and, as a rule, to this charge some limestone is added. Under these conditions it is difficult to get more than a concentration of 3 into 2, or 2 into 1, at the utmost. I am not prepared to say whether in our case there would be any commercial gain in adding limestone, as unquestionably more quartz would have to be used; and, as both quartz and lime cost money and it takes money to smelt them, it is a question of balancing the copper saving as against the added cost of the flux. As I have before stated, mechanical difficulties (due to the inexperience of our men and also to the interference incidental to the enlargement of our plant) have, to a large degree, made our operation irregular.

I give herewith the results on a fairly average week's run in which two furnaces were smelting straight sulphide ore and the third furnace was treating matte (from the first operation and also custom matte). The amount of matte concentrated was not equal to that produced in the first operation. The figures are only given to show about how the furnace tonnage ran. The plant, as at present constituted, is not properly balanced, and it is our intention, when more furnaces are available, to treat the converter-slag with the green ore, so as to reduce its copper content as much as possible. Under present conditions it is treated in the concentration furnace, and so, when finally discharged, it carries at least 0.6% copper.

WEEKLY TONNAGE.

	No. 1. Tons.	No. 2. Tons.	No. 3. Tons.	Total Tons.
Furnace.				
Sulphide ore	2,667	2,540	253	5,460
Converter slag	....	....	721	721
Quartz flux	559	456	220	1,235
Custom matte, 45%	....	....	231	231
First matte, 12%	....	20	557	577
Total charge	3,226	3,016	1,982	8,224
	Tons.	Tons.	Tons.	Tons.
Coke used	92	99	174	365
Coke percentage %	2.9	3.3	8.8	4.4
Tons of charge per day	461	431	283	

It is almost unnecessary to state, that in all of our work, we have used nothing but cold blast; and, thus far, the use of hot blast in treating a heavy sulphide ore like ours has not, as far as I can see, been of any advantage. At Mt. Lyell it was formerly supposed that it was necessary; but, after four years' practice, the use of it has been finally abandoned. At the plant of our neighbors, the Ducktown Sulphur, Copper & Iron Company, Mr. Freeland put in a

U-pipe hot-blast stove, but could find no improvement either metallurgically or commercially therefrom, either on roasted or unroasted ore. He, therefore, for the present has abandoned any further attempt to utilize hot blast.

In *The Engineering and Mining Journal*, June 8, 1905, in two references to hot blast, both of them seem to agree that cold blast is an aid to concentration. The fact is, my colleagues and I believe the time is not distant when we, perhaps, will be able to make from our ore, in one operation, a matte that can be directly converted. At the Ducktown Sulphur, Copper & Iron Company's plant the furnaces have occasionally produced high-grade matte for 24 hours at a time; yet, so far, they have not been able to keep up this concentration and at the same time maintain the capacity of the furnaces.

The accompanying table, showing the composition of our matte, may prove interesting, particularly as it shows that the amount of sulphur present is considerably less than that which would be expected (from an accepted formula for matte, $Cu_2S.FeS$). For a time we thought that we had an excess of iron in our matte, but we finally found that the actual iron very nearly agreed with the theoretical, the only difference being that a certain amount of the sulphur was replaced by oxygen, presumably part of the iron being FeS and the other part Fe_3O_4.

We hope that the time will come when the barring of furnaces will be reduced to a minimum. At present, if a furnace gets in such bad shape as to be pretty well crusted on top, we allow the furnace to run down, adding several charges of slag and coke. The blowing engine is then reduced in speed, so that it delivers very little air to the furnace, and long 1½ in. steel bars, with chisel-shaped ends, made of 2 by 4-in. steel, are driven down behind the crusts. A chain is slipped over the bar and attached to one of the electric locomotives, which slowly pulls it and the crusts over into the furnace. This is a vast improvement on the former method of attempt-

SP. G. OF SLAGS AND MATTES.

COMPARISON OF MATTE HELD IN SUSPENSION IN SLAG WITH SP. G., ASSUMING THAT THE COPPER IN THE SLAG IS IN THE FORM OF PARTICLES OF MATTE.

SAMPLE	Cu.	SiO_2	Fe	Fe O	Al_2O_3	Ca O	Mg O	S	Zn	Zn O	Mn	Mn O	Sp G	Gms. of Matte in 1 Gm. of Slag
Fur. No. 3, Dec., 1904....Slag	.86	33.16	......	57.42	1.92	1.42	.63	2.15	....	1.96	....	.46	3.836	.0187
Fur. No. 3, Dec., 1904..Matte	45.78	......	25.93	......	....	......	....	24.51	2.09	....	.14	....	5.003	......
Fur. No. 1, Dec., 1904....Slag	.27	36.35	......	44.54	4.10	7.90	1.99	1.67	....	2.66	....	.64	3.568	.0232
Fur. No. 1, Dec., 1904..Matte	11.62	......	53.80	......	....	......	....	25.12	2.77	....	.32	....	4.802	......
Fur. No. 1, Sept., 1904...Slag	.21	37.18	......	42.73	4.96	8.71	2.11	1.32	....	2.30	....	.41	3.514	.0205
Fur. No. 1, Sept., 1904..Matte	10.24	......	53.90	......	....	......	....	25.41	1.66	....	.43	....	4.820	......
Fur. No. 1, Feb., 1905....Slag	.37	39.06	......	45.67	4.14	6.04	1.62	1.52	....	1.44	....	.40	3.383	.0158
Fur. No. 1, Feb., 1905..Matte	23.43	......	43.70	......	....	......	....	26.52	1.48	....	.15	....	4.900	......
Salt Lake Valley Slag.........	.24	42.35	......	30.66	3.02	20.38	1.23	.67	....	.93	....	.24	3.215	.0120
Salt Lake Valley Matte.......	20.00	......	......	......	....	......	....	......	....	....	....	....	......	......

ing to attack these crusts with small bars and by hand alone.

The addition to the blast-furnace plant will consist of four furnaces, 56 by 270 in. at the tuyeres, but of the same height and general shape as the old furnaces, excepting that the lower jackets will be carried down to the sole-plate. These furnaces will have rounded corners, so as to facilitate barring, and the number of tuyeres will be increased to 26 on the rear side of the furnace and 24 on the front side, each tuyere being of about 3-in. diameter. We are of the opinion that a large number of tuyeres, more closely spaced, will give us better results. In other words, each tuyere is a nucleus of activity; so that, if carried to its logical extreme, a narrow slot would be the ideal opening for the air blast. This, of course, is an impossible construction. The tops of the furnaces will be so modified that in case it becomes necessary to do any heavy barring, extra doors above the feed-doors will be thrown open and bars held vertically will be driven by men on a platform above the feed-floor. Each furnace will be supplied with air by a horizontal-piston blowing engine, capable of delivering 30,000 cu. ft. free air per minute. Only three of these new engines will be provided, as it is estimated that in the final plant, which will consist of seven furnaces and six blowing engines, one furnace will always be out of blast.

It is still a little too early to determine exactly what the commercial saving will be on pyrite smetling. We have certainly eliminated the roast-yard expense, which amounts to 40c. per ton. I believe that the blast-furnace expense per ton of green ore will be about the same; and I am of the opinion that we will save at least four pounds more of copper than we did in the old method. In addition to this, much to our gratification, we have found that the gases from the furnace are so rich in SO_2 that we can make acid from them by the chamber process. This will give us a by-product, which will be of great value in the

South for the purpose of making fertilizers, and not only will it operate toward reducing the volume of gases given to the atmosphere, but will prove a source of considerable revenue. We are, however, providing the new plant with a stack 325 ft. high (internal diameter 20 ft.), so as to give us perfect draught, which is desirable in pyrite smelting, as without proper draught, and with gases leaking from the doors, the feed-floor is by no means a comfortable place.

I regret that I am not in a position to say that we have absolutely settled everything to our satisfaction. Perhaps it is well that such is the case, for all industries either advance or recede; they never stand still.

J. PARKE CHANNING.

New York, June 12, 1905.

RATIO OF HEARTH AREA TO FURNACE CAPACITY.[1]

BY W. RANDOLPH VAN LIEW.

It would be a matter of great value to metallurgical practice if, instead of each manager, or superintendent, having his own particular notion as to the width to be given a blast furnace, it could be determined on a scientific basis what should be the width best adapted to secure the most economical costs, and the largest tonnage smelted, per square foot of hearth area at the tuyeres.

I say tonnage smelted per square foot of hearth area at the tuyeres, as it is the only true standard by which to gauge, or compare, the workings of different types of furnaces. A furnace 15 ft. long, smelting 500 tons of material a day, could just as well be made 150 ft. long, and to smelt 5,000 tons per day. It is simply a question of spouts and settlers at regular intervals. Two furnaces may be run side by side, one smelting 700 tons, the other 400 tons, yet the latter may be doing half as much more work, as it all depends on the number of square feet of area available for smelting at the tuyere zone.

Until recently, very little has been published concerning the tonnage treated by furnaces of different sizes situated in different localities. This has made it difficult for those either contemplating the erection of plants or the modification of existing furnaces, to form an idea of what the different types of furnaces can do in the way of capacity.

This spirit of secrecy, in the end, always operates against those who practise it. It seems to have charac-

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[1] *The Engineering and Mining Journal,* March 21, 1903, page 442.

terized the metallurgy of copper more than that of any other metal. As an instance, I may mention the fire refining of copper, the art of which, until the past few years, was jealously guarded, and practically handed down from father to son. Again, in the province of the electrolytic refining of copper, until quite lately, the doors were closed to all, and this spirit even survives to-day in some Eastern refineries, in spite of their general operations, currents and strength of electrolyte being, within narrow limits, known to most of the profession.

As an illustration of the extremes to which this spirit lends itself, it is but necessary to point to our English cousins, in the metallurgy of iron, where this same narrow spirit is exhibited, each company wrapping itself within its own walls, so that their representatives, coming to this country, look in open-mouthed astonishment at the way in which the doors of our large steel plants are thrown open to them, and at the freedom with which our methods are explained to them; this very interchange of ideas, and improved practice among ourselves, no doubt being one of the principal factors in placing the United States in that foremost position which she holds in the steel industry to-day.

Fortunately, those engaged in the metallurgy of copper are, at last, following in the footsteps of our friends, the iron metallurgists. Among those who have taken up the cudgel in this respect are Mr. William A. Heywood, of the Tennessee Copper Company, and Mr. Paul Johnson, of the Greenwood, British Columbia, Company, who, in recent issues of *The Engineering and Mining Journal*, have given interesting data, that afford a valuable study in widths of furnaces as related to their output. Why cannot each superintending metallurgist who may read these articles contribute his quota also to the fund of general experience, that, from the whole, we may all of us benefit, and, having considered the difference in different ores, decide, within approximate limits, on that width which would seem to give the best results?

As I have said, the length of a furnace is immaterial; as to width, the narrowest furnace, of the usual rectangular type, that I know is 35 in. wide at the tuyeres; the widest is 56 in., or 65 per cent wider. It was my privilege, while in charge of one of the large blast-furnace plants in Montana, to observe the workings of furnaces of different widths on the same ores, the one style 44 by 100 in., the other 56 by 180 in., and the result from every standpoint was in favor of the narrower furnace.

The 56-in. furnace was much higher from the tuyeres to the charging floor, but the burden on the tuyeres was kept as nearly as possible at the same height.

The 56-in. furnace was fed by charging cars, the 44-in. by charging wheelbarrows, dumped directly into the furnace. Slags were very nearly the same, 2 to 3 per cent more iron not uncommonly being given to the slags from the 56-in. furnace.

The slag contents in copper were also the same, and very low. The 44 by 100 in. furnace would average, month after month, 215 tons per 24 hours, not including coke.

The 56 by 180 in. furnaces, at two different Montana plants smelting Butte ores, smelted 400 to 500 tons per day, with an approximate average of 420 tons.

The 44 by 100 in. furnace had 30.5 sq. ft. of hearth area at the tuyeres, which gives 7.06 tons smelted per square foot of hearth.

The 56 by 180-in. furnace had 70 sq. ft. of area, which gives 6.0 tons smelted per square foot of hearth.

The 44-in. furnace required 19 oz. blast, the 56-in. furnace required 27 oz. blast and 10 per cent more total fuel to do the smelting, the slags in both cases running between 38 and 42 per cent silica, 18 and 21 per cent iron, 18 and 24 per cent lime, and 0.2 to 0.4 per cent copper.

The tonnage of a furnace is affected also by the amount of iron in the slag; up to a certain point the more iron the faster the furnace will smelt.

Mr. Heywood, of the Tennessee Copper Company, in his interesting and valuable article, contained in the *Journal*,[1] states that on Tennessee ores, which, if I am correctly informed, carry enough iron to offer the desired quantity for good, easy running slags, the average work of a 56 by 180 in. blast furnace was 495 tons per day, or 7.07 tons smelted per square foot of hearth. For one month 561 tons were smelted per day, or 8.01 tons per square foot of hearth; for seven days the average output was 615 tons per day, or 8.8 tons per square foot of hearth, while the maximum tonnage for any one day was 657 tons, or 9.4 tons smelted per square foot of hearth area, the blast pressure varying from 20 to 40 oz., while composition of the slags was not given.

Turning from the wide 56 and 44 in. furnaces, smelting Tennessee and Montana ores, we will see, by referring to the equally valuable article by Mr. Paul Johnson, of the Greenwood Smelter in British Columbia, in the *Journal*,[2] that we can form an idea of the work of a still narrower type of blast furnace.

The furnaces at Greenwood are only 42 in. wide. The size of the furnace at the tuyeres is 42 by 150 in., or 43.7 sq. ft. of hearth area. The daily tonnage of ore is given as 380.5 tons, or 8.7 tons per square foot of hearth area. The highest average tonnage treated during one month was 428.6 tons per day, or 9.8 tons per square foot of hearth, and the highest for a single day was 460 tons of ore, or 10.5 tons per square foot of hearth. The average slags are stated to contain 39.8 per cent silica, 23.6 per cent iron, 19.6 per cent lime and

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[1] ***The Engineering and Mining Journal*, July 26, 1902, page 119.**

[2] ***Idem*, Aug. 23, 1902, page 251.**

0.321 per cent copper, when producing a 50.1 per cent matte and using a blast pressure of 14.15 oz.

We come now to a still narrower type, a furnace 35 in. wide at the tuyeres. It was my privilege to operate such a furnace on Old Dominion ores, duplicating all the conditions that would be required of it under matte smelting and converter practice. The trial lasted five days, when the sulphides in the bins were used up and the furnace returned to black copper smelting.

The results are of interest as bearing on the subject of widths of furnaces. The furnace in question was 35 by 122 in. at the tuyeres, that is, it had 29.68 sq. ft. of hearth area. Not including coke, and under its duplicated converter conditions, the tonnage was as follows:

	Total tons.	Tons per sq. ft. of hearth area.
First day	255.0	8.60
Second day	282.0	9.50
Third day	283.8	9.56
Fourth day	322.0	10.84
Fifth day	334.0	11.25

On the fourth and fifth days the furnace ran on 7 per cent coke, and was running faster, when changed back to black copper smelting, than two days before, when placed on 7 per cent fuel.

The slags made varied between 37 and 41 per cent silica, 25 and 29 per cent iron, 11 and 15 per cent lime, 9 and 14 per cent alumina.

The slags ran from 0.20 to 0.30 per cent copper for 42 to 48 per cent matte; from 0.30 to 0.40 per cent copper for 48 to 53 per cent matte. The blast was kept at 21 oz.

While I appreciate that a campaign of five days does not possess the same value as one of many months, still the fact remains patent that even so short a trial will possess its value, as indicating the possibilities of the narrow type of furnace.

Of the four types of furnaces—the 56-in. on Tennessee ores, the 56 and 44-in. on Butte ores, the 42-in. on British Columbia ores, and the 35-in. on Old Dominion ores—the advantage seems overwhelmingly in favor of the two narrower types of furnaces, both from the standpoint of tonnage smelted per square foot of hearth area at the tuyeres, and of economy of power for blast pressure. In the matter of feeding, I have found but little difference between hand feeding and charging direct from barrows—if the workmen have been trained to place their charges where needed; but comparing either of these to charging by cars, my own experience duplicates that of Mr. Paul Johnson. While automatic charging from cars permits larger bin capacity, the location of bins farther from the furnace, and a certain amount of labor saved, yet it can be used only by sacrificing other conditions, the maintenance of which to the metallurgist—and to the company—are more to be desired. It is poor metallurgical practice to use any manner of feeding that will increase the fuel demanded, and also choke the output of your furnace. These two things charging by cars unquestionably does, while considering the extra labor involved in maintaining the other conditions about the furnaces, there is not so much labor saved, on the whole, as the uninitiated believe. Figuring simply the labor cost per ton of ore, the advantage will be in favor of the mechanical feeding by cars, but placing against this saving the resultant interior condition of furnace, the extra coke, and the decreased tonnage per square foot of hearth area, the smelting costs will be higher with the car charging.

To add to this data, it would be of much value and interest to the profession if the managers or superintendents of smelting plants, possessing the 40 and 50-in. width of blast furnace, would give us the possibilities of those types.

CONTRIBUTION BY J. PARKE CHANNING.

The Editor:

Sir—I have read Mr. Van Liew's communication on this subject, in your issue of to-day, and quite agree with him that it is desirable to disseminate as much as possible all information concerning metallurgical practice for the common good of all concerned.

The object of smelting any ore is to secure as great an extraction as possible at a minimum cost. In other words, the cost of extraction is the operating expense, plus the value of the slag loss.

Certainly, Mr. Van Liew's figures seem to indicate that a narrow furnace gives a greater tonnage per square foot of hearth area; but are we striving for this particular record? Is it not rather the broad question of cheap and efficient smelting?

I do not agree with Mr. Van Liew that a furnace 150 ft. long would do ten times as much work as one 15 ft. long. If I am correctly informed, at the Boleo property in Lower California, some 30-ft. furnaces were erected, were found unsatisfactory, and finally divided into two. I fear that a furnace of this kind might be much like the Siamese Twins, with one having a congestive chill and the other a burning fever.

Theoretically, providing the hearth area were the same, a circular furnace would be the cheapest to construct, and in Arizona they still consider that they get better results from an elliptical cross-section. If furnaces were made narrow and long, it would mean increasing the size of buildings, particularly if the furnaces were put end to end, which seems to be the best method of arranging when mechanical charging is used. If this point is taken into consideration, I think that the total cost of the plant would be found higher than in the case of furnaces approaching a square in their cross-section.

Our experience in Tennessee with a 56-in. by 180-in. furnace has been very satisfactory, and with a constantly increasing tonnage, so that we, on our part, are of the opinion that there should be a certain ratio between length and width, and that if we made our furnaces, say, 240 in. long, it would be well to increase the width to 66 in. This, however, is purely theoretical.

At the Great Falls plant of the Boston & Montana Company, Mr. Frank Klepetko, in 1896, made some extensive experiments with three furnaces, each 180 in. long, and widths, respectively, 42, 56 and 72 in. The concensus of opinion, after several months of experimenting, was that the 56-in. furnace gave the best results, and it was for this reason that, in designing the Tennessee Copper Company's smelting plant, I followed the result of Mr. Klepetko's experiments. The relative tonnage of the three experimental furnaces at Great Falls, in the same order as above given, was 330, 440 and 400.

Turning now to the question of coke consumption, we find in Tennessee that we use somewhat less coke than our neighbors, the Ducktown Sulphur, Copper & Iron Company, which runs small Herreshoff furnaces. Certainly at Great Falls there was no noticeable difference in the coke consumption of the three furnaces.

It certainly would take more men on the feed floor and the tapping floor to run five 300-ton furnaces than it would three 500-ton furnaces; even assuming that the cost of charging was in both cases the same.

As regards power consumption for blast, I am quite ready to admit that the large furnace consumes more horse-power, but I am of the opinion that this increased cost is more than offset by other advantages.

To say that mechanical charging is not defensible is tantamount to saying that we should not replace hand labor by machinery. I am well aware that metallurgists consider that a hand-charged furnace will run faster than one in which the material is irregularly dumped. In large furnaces, however, it is not as apparent, and the top of the charge can be kept level almost as well as in a small hand-charged furnace.

In the month of January, 1903, the Tennessee two-blast furnaces smelted 28,699 tons of charge in 57½ furnace days, not including blast furnace products re-smelted, and the expense was as follows:

Coke, 0.1233 tons, at $4.93	$0.6082
Quartz, 0.0958 tons, at $0.90	0.0862
Supplies, including coal for power	0.1266
Labor and superintendence	0.2111
Total	$1.0321

In the above figures, the price of coke is abnormally high, on account of the activity of the Southern iron market, though we expect that, with the completion of new ovens, the price will return to a normal one of about $3.50.

Labor averages about $1.55, so that we may assume about 0.14 of a shift per ton of ore. This includes, in addition to the furnace labor, the proper proportion of surface and power-house labor, the balance going to converter expense.

The charging expense for the month of January, which included motormen, car-loaders and dumpers, repairs to motors, cars, tracks and scales, and power for operating the electrical locomotives, was as follows:

Labor	$0.0340
Supplies	0.0119
Total	$0.0459

I doubt very much if hand labor could in any way compete with these figures, or whether the increased efficiency of the furnace would more than counterbalance the saving in handling the material.

Our slags in Tennessee run about as follows: SiO_2, 31.54; Cu, 0.49; FeO, 52.25; Mn, 0.79; Zn, 2.02; Al_2O_3, 3.50; CaO, 5.93; MgO, 1.86; S, 1.25.

I cannot forget the opening sentence of the late Prof. Egleston, in his first lecture to me as a student, in which he said that "metallurgy is the science of extracting money from ores." I fear that many of us are carried away with the idea that high tonnage per square foot of hearth area, or low slag, is what we are striving for, and often forget the commercial side.

J. Parke Channing.

New York, March 21, 1903.

CONTRIBUTION BY WILLIAM A. HEYWOOD.

The Editor:

Sir—A number of interesting articles on tonnage in copper blast-furnace smelting have appeared recently in *The Engineering and Mining Journal,* and doubtless all metallurgists are interested in the publication of the results obtained at different smelters. I have received a number of letters in reply to my communication published July 26, 1902, and in response to requests for further and more recent information, give the following record of the work done at the Tennessee Copper Company's smelter during the month of February, 1903. Two furnaces were in blast for 28 days, and smelted as follows:

	Tons.
Total charge smelted, not including coke..........	30,095
Total ore smelted..................................	26,757
Total coke used....................................	3,269
Charge smelted per day for entire month..........	1,074
Charge smelted per day per furnace..............	537

The average slag analysis for the month was: Insoluble, 31.22 per cent; FeO, 51.39; Cu, 0.44. The remainder is mostly alumina. Our ores contain little lime, and we are compelled to use quartz to flux the large excess of iron. On account of the high specific gravity of these irony slags, they carry away more copper than they would if part of the iron were replaced with lime, which would make not only a more fusible but a lighter slag. The average number of men and boys employed in the blast-furnace department, per day of 24 hours, is 54, making an average of about 20 tons per day smelted for each man employed. The cost of smelting was given in detail in *The Engineering and Mining Journal* of February 14, 1903, being $1.05 per ton of ore for the entire year 1902. Last month, notwithstanding coke has increased in cost, the smelting cost was under that figure.

Following is a summary of the figures given in the recent articles published on blast-furnace smelting tonnages:

Name of Smelter.	Authority.	Size of furnace.	Area at Tuyeres in sq. ft.	Tons per day.	Tons per sq. ft. of area.
Tenn. Copper Co. (Feb. '03)		56 by 180	70	536	7.65
"Two Montana Plants"	W. R. VanLiew	56 by 180	70	420	6
Granby, B. C. (Dec. '01)	E. Jacobs	44 by 160	49.5	354	7.1
B. C. Copper Co. (Jan. '02)	P. Johnson	42 by 150	43.7	428	9.8
Old Dominion (5 days)	W. R. Van Liew	35 by 122	29.68	295	9.9

Mr. W. R. Van Liew, in an interesting article on 'Ratio of Hearth Area to Furnace Capacity,' states that "tonnage smelted per square foot of hearth area at the tuyeres is the only true standard by which to gauge, or compare, the workings of different types of furnace." If we adopt this standard the above table shows the results to be in favor of the smaller furnaces, not because they smelt more tons at a given cost, but because they have a smaller divisor. The standard I would use in comparing different types of furnaces would be their relative economy. As conditions vary at every smelter, the size of furnace that is most economical must be determined by actual experiment. Tonnage is only one of the elements to be considered, but so far the experience in copper, as in iron, smelting appears to indicate that the large furnaces are more economical.

I cannot agree with the two writers who have recently mentioned the advantages of hand-feeding over mechanical feeding. A proper system of mechanical feed has proved so much more economical than hand-feeding that it is, in my opinion, merely a question of time when it will be used on all large furnaces. Regarding the claim that mechanical feeding produces stoppages due to irregularities in the running of the furnace, I would state that the last campaign of our No. 1 furnace lasted 6 months, 18 days, during which time the furnace did not lose a single shift, and smelted 98,886 tons of material, not including coke, all of which was fed mechanically.

William A. Heywood.

Copperhill, Tenn., March 24, 1903.

CONTRIBUTION BY GEORGE W. METCALFE.

The Editor:

Sir—Following out the suggestion recently made in these columns as to the publication of data on this subject, I give below an account of some experiments recently made here which may be of interest to the profession, though not yet furnishing sufficient data to warrant any general conclusion.

The blast-furnace plant here was designed and built in accordance with the results of the Great Falls experiments of Mr. Klepetko, mentioned by Mr. Channing in his recent letter; the furnaces were 56 by 180 in. at the tuyeres, 72 by 180 in. at the top of the jackets and 18 ft. from tuyeres to charging floor. The method of charging used is hand dumping of large coke barrows and mechanical dumping of large tram-cars containing approximately 5,000 lb. each. Materials used are Butte ores, coarse concentrates and briquettes of flue dust and slime, fluxed by converter slag and limestone.

On first starting up in February, 1902, considerable trouble was experienced from heavy crusts forming on the jackets, both ends and sides. Much of the end crusting was done away with by cutting off the end tuyeres—originally there were 12 tuyeres in front, 14 at the back, and 3 at each end—but the side crusts seemed to be due to improper distribution of the charge caused by the partial separation of its coarse and fine components by sliding from the car over a sloping charging plate and falling three or four feet into the furnace. The tendency was for the larger and heavier fragments to fall in the center, the finer materials remaining at the sides. The higher this drop the more pronounced the sorting action; so, as the natural expedient of keeping the furnaces full was found to make even worse crusts, the experiment was tried, in building two new furnaces, of making them respectively 3 ft. and 6 ft. lower than the original five furnaces. At the same time. as it was the intention to run with less depth of charge and lower blast pressure, the jackets were drawn in at the bottom so as to make these furnaces 48 in. instead of 56 in. wide.

Blast used on the original five furnaces was 28 to 30 oz., on the No. 6 (15 ft. deep) 26 oz., and on the No. 7 (12 ft. deep) 24 oz. Other blast pressures were used at times, but these were finally settled on as yielding the best results.

During a six weeks' run No. 7 furnace (12 ft. deep) averaged 352 tons of charge per day on 10.2 per cent of coke, while four furnaces of the original type in that time averaged 397 tons of charge per day on 9.8 per cent coke. During this period a constant attempt was made to run No. 7 on the same charge and coke percentage as the other furnaces, but it invariably became crusted badly, and had to be put on a more fusible and ferruginous charge with a higher coke per cent until the crusts were burned out. However, during all this period, as was expected, it made about 4 per cent better matte than the others, which averaged 40.4 per cent copper. The average slag of all the furnaces assayed: Cu, 0.19 per cent; SiO_2, 43.3; FeO, 26.0; CaO, 21.8, and Al_2O_3, 8 per cent.

During another six weeks' run No. 6 furnace (15 ft. deep) averaged 383 tons per day on 9.7 per cent coke, while the four furnaces of the original type averaged 407 tons per day on 9.6 per cent coke, all the furnaces being practically all the time on the same charge. No. 6 made matte averaging 42.6 per cent copper, the average of the others being 39.7 per cent copper. The slags of all the furnaces averaged: Cu, 0.18 per cent; SiO_2, 44.0; FeO, 26.0; CaO, 21.8; Al_2O_3, 6.8.

During four of the six weeks of this latter experiment No. 3 furnace also had the jackets drawn in to make the size 48 by 180 in. at the tuyeres, though remaining as before 18 ft. in depth. During that time it averaged 397 tons per day on 9.3 per cent coke, while the three unaltered 18-ft. furnaces averaged 409 tons on 9.6 per cent coke. The No. 3 matte averaged 0.5 per cent less than the 56 by 180 in. furnaces. Fed high and blown the same as the others it would nearly keep up in tonnage, but it made the lower grade of matte; with less blast, even down to 20 oz., and fed low it still made 0.5 per cent poorer matte, and, of course, fell still further behind in tonnage.

The above figures as to tonnage are compiled from our charge foreman's reports. The actual amounts smelted, owing to our method of loading the charges, always ran about 10 per cent more after being checked up by the railroad weights; quite the contrary result from that obtained by most plants on checking against railroad weights.

None of these changes seemed to have any appreciable effect in lessening the crusts in the upper part of the furnaces, though the 18-ft. furnace contracted to 48 by 180 in. at the tuyeres kept hotter and in better shape at the bottom. We have, however, as the feeders became more accustomed to the system of charging, usually been able, on all the furnaces, to prevent the formation of such crusts as would seriously retard their running.

In the experiments on No. 6 and No. 7 furnaces the differences in depth interfere somewhat with drawing a conclusion as to the effect of the narrower width *per se*;

while the results on No. 3, though showing a decided increase in tonnage per square foot of hearth area, certainly do not indicate that the narrower width is *per se* a commercial advantage in smelting our materials. However, the results on No. 6, showing a slightly decreased tonnage but an increased grade of matte, and a slightly decreased power expense, owing to the lower blast pressure, may readily be held to indicate a commercial advantage for the 48 by 180 in. furnace of 15 ft. depth in a plant where cost of converting is an important item.

Our experience here leads me to differ from Mr. Van Liew as to the advantage of mechanical charging. Until 1902 we were here running a number of furnaces 45 by 100 in. at the tuyeres and 11 ft. from tuyeres to charging floor. Pretty much the same materials were used as in the 56 by 180 in. furnaces now in use, but in slightly different proportions; the small furnaces using about 38 per cent raw ore, 32 per cent of slag and matte, and 30 per cent of limestone, and making a slag containing about 39 per cent SiO_2, 26.6 FeO, and 22 CaO; while the large furnaces use about 45 per cent of raw ore, 27 per cent of slag and matte and 28 per cent of limestone, and make a slag containing about 42.5 per cent SiO_2, 26.9 FeO, and 21.2 CaO. The small furnaces were fed from small hand barrows, but so skillfully that they were kept practically free from crusts, while the large furnaces are fed from cars, as described above, and are generally somewhat crusted. The large furnaces use a larger coke per cent, though a comparison of the materials used would not indicate that this is entirely due to the mechanical feeding. However, the cost-sheets show that the excess in coke costs on the large furnaces is considerably more than balanced by the excess in labor costs on the small furnaces, while the larger furnaces get a still further advantage in the final cost per ton from the large divisor that their heavy tonnage gives them into the general expenses of the plant.

I think further data from plants smelting the same materials, in different models of furnaces, would be very generally appreciated.

GEORGE W. METCALFE.

Anaconda, Mont., April 10, 1903.

CONTRIBUTION BY JAMES W. NEILL.

The Editor:

Sir—The interesting article by Mr. Van Liew in your issue of the 21st,[1] touches upon a point in smelting practice of vital interest to practical men. The question, "How wide should a blast furnace be to treat a certain ore?" is not so readily answered, nor will mere reference to the workings of other furnaces always be a safe guide. The quality of the ore to be smelted and the results to be obtained must be considered, and the furnace should conform to these. The different furnaces mentioned by Mr. Van Liew are handling very different materials, thus: In Montana the material is crude ore, coarse in size, very silicious, carrying 12 to 15 per cent sulphur. The fluxes are either coarse concentrate or coarse iron ore, and coarse limestone. Briquetted material is included. This charge produces a very silicious slag and matte of converter grade. In British Columbia, Mr. Johnson has been smelting a crude ore consisting chiefly of metamorphosed limestone (containing garnet, epidote, etc.) with much magnetite and very small amounts of sulphides, so that there is no sulphur to be eliminated; the ore is a "natural flux." Moreover, it is broken in surface quarries and arrives at the smelter in large, coarse pieces, ideal for blast-furnace work.

In Tennessee, Mr. Channing is smelting a heap-roasted pyrrhotite, which is low in silica, requiring the addition of quartz for good slag results. This material again is in large pieces, porous from the roasting operation, and in most excellent condition for the blast furnace. Here, again, the amount of sulphur to be driven

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[1] See page 266.

off in the furnace is considerable. The slag is naturally an irony one, but the resulting matte is of converter grade.

At the Globe plant, in Arizona, the regular work of the furnace has evidently been the production of black copper from oxidized ores, and the test made by Mr. Van Liew was for the purpose of producing matte instead. It is evident from the context that the amount of sulphides used in this 5-day run was no more than necessary to cover the copper and produce matte, therefore no sulphur to speak of would have been driven off. I do not know the conditions as to the size of ore particles when fed to furnace.

If instead of the conditions described we consider an ore which contains 35 per cent sulphur (as pyrite), which breaks in the mine so fine that it resembles concentrate (60 per cent will pass a ¾-in. screen), and carries silica sufficient to flux the iron; under *such* conditions, what width of furnace should I use? The data given by Mr. Van Liew do not help me, as the conditions are not similar to any of those cited. We cannot heap-roast; and mechanical roasting will necessitate briquetting, which we desire to avoid.

It is obvious that to charge such fine material into a blast furnace 56 in. in width, will result in a very heavy body of material, through which the blast will penetrate with difficulty and necessitate a high pressure. This will result in the formation of 'blow-holes' and all the attendant troubles known to the practiced blast-furnace man, with the production of immense quantities of flue-dust and flue losses. Furthermore, high pressure is the surest agent toward reducing action in the furnace, but in this very case we desire oxidizing action, or our matte will be too low in grade! Therefore, such conditions, more than the question of buildings or of previous records, must govern our choice of furnace width.

I do not think that the data given by Mr. Van Liew warrant the deduction that a narrow furnace will always smelt more tons per square foot of hearth area; the physical conditions of the ore charge and its chemical contents must be largely factors in the final result. Thus, for the Old Dominion ores making black copper I should choose a round or elliptical furnace; for the porous Tennessee ores, any length and width which my blast would penetrate; in Butte not over 45 in. in width, and for the fine sulphides mentioned above not over 42 in., with a preference for less.

James W. Neill.

Salt Lake City, Utah, March 26, 1903.

REVERBERATORY COPPER SMELTING.

By E. P. MATHEWSON.

Probably the largest installation of reverberatory furnaces built in recent years for the smelting of copper is that of the Washoe Copper Co., at Anaconda, Mont. The plant consists of 14 furnaces, originally 20 by 50 ft. hearth measurement, set back to back in two rows, and housed in two steel buildings substantially built and well ventilated. Between these buildings is a chimney 225 ft. high, 20 ft. internal diameter, constructed of steel and lined with brick, the connections to the furnaces being made by four main flues. One feature of the original construction was an arrangement for pre-heating the air by the heat of the escaping gases and the heat radiated from the bottom of the hearth. The air was admitted first to a brick chamber built around the brick-lined steel pipe, which carried the waste gases to the main flue; thence the partially heated air passed beneath the bottom of the furnace in a narrow channel, passing from the front to the back four times before rising in a cast-iron box to the top of the furnace; thence to a sheet-steel box above the bridge wall, being finally admitted to mix with the gases from the fire-box, through checker work in the roof above the bridge wall, the draft of the furnace being sufficient to draw in the hot air. In remodeling the furnaces, this arrangement was omitted, as it was considered more important to retain the heat in the bottom of the furnaces than to use it for pre-heating the air furnished to the top of the charge, more rapid smelting being accomplished by keeping the matte in the furnace as hot as possible.

In building the plant, every convenience to facilitate the handling of materials was arranged for. The ashes from ash-pits are sluiced away by waste water from the concentrating department, and the slags are granulated and washed away to the dump by the same means. The matte is tapped from time to time into large ladles holding 11 tons and drawn by compressed-air locomotives to the converting department, where the still molten matte is dumped from ladle to converter to be blown to copper. The air necessary for the combustion of the fuel was forced under the grates by fans, the ash-pit being closed by cast-iron doors. Each furnace had an average daily smelting capacity of 100 tons of calcines. The fuel used was obtained from Diamondville, Wyo., and consisted of 'run-of-mine' coal of the following composition: water, 1.6; volatile combustible, 38.7; fixed carbon, 49.2; and ash, 10.5 per cent. This coal gives very satisfactory results under natural draft, but with forced draft it does not act so well.

In the former practice the time lost in grating the furnace averaged three hours per day per furnace, and during the grating (which occupied one-half hour at a time) the front of the furnace became cooled and the slag frequently set near the skimming-door. At the suggestion of Capt. W. M. Kelly, Furnace No. 9 was remodeled on the lines of the best furnace at the old Anaconda works, and an extra large fire-box was constructed in order to give every chance possible for the furnace to work without forced draught. The grates were ordinary bar iron, with open ash-pit. The flues were changed to permit of a more direct connection without sharp bends, and the down-takes for escaping gases were enlarged. The results obtained in the modified furnace were excellent at the start, and the good record has been satisfactorily maintained.

In consequence of the improvements in furnace No. 9, the other furnaces were altered accordingly, and the working under the new conditions made a very excellent showing, as set forth by the following data: Average tonnage of calcines treated per furnace per day was: January, 1903, 106.6; February, 115.7 tons; March, 123.84 tons; and April, 133.53 tons. The fuel consumption averaged one ton of coal to three tons of calcines treated, and it may be stated that this good average will be still further improved. The total supply of coal delivered to the plant is charged up as weighed, and during the months under review (January to May, 1903) much of this coal was consumed in starting up new furnaces and in tapping out old ones. The best record for fuel consumption yet obtained on a single furnace is 1 ton of coal to 4.07 tons of calcines, the output of the furnace being 171 tons. Under the present conditions the furnaces carry an even heat, and no time is lost in grating. The materials sent to each furnace are carefully weighed in charge cars (20 tons to the charge), which are again weighed when empty each trip. The weighing and tramming are done by a set of men entirely separate from the furnace crew and under a separate foreman. The coal weights are checked monthly by the railroad car weights and check within 1.5 per cent of the latter—the variation being overweight. All material for the furnaces is handled by compressed-air locomotives, and is loaded in hopper-bottom cars which discharge into the hoppers above the furnaces.

Under the improved conditions of working, the aggregate cost of coal, labor and repairs has been reduced by more than $1,000 per day. The following details of furnace operations are of interest: Draft in down-take = 1.5 in. of water. Analysis of material charged: Cu, 10.5 per cent; SiO_2, 33.2; FeO, 39.5, and S, 7.8 per cent. Analysis of slag produced: Cu, 0.39 per cent; SiO_2, 41.4; FeO, 45.8, and CaO, 3.1 per cent. Copper content of matte produced 47.44 per cent.

During March, 1903, three of the old-style furnaces were still in operation, which reduced the average tonnage. The modified furnace treated on the average 135 tons of calcines per day, and each of the old furnaces when remodeled will have a capacity of 140 tons per day, with fuel consumption of 1 ton of coal to 3.5 tons of calcines smelted. Another important change now being installed is the placing of a 300-h. p. Stirling boiler between each reverberatory furnace and the main flue, the idea being to utilize the waste heat of the gases escaping from the furnaces. These boilers have been tried once before at the plant, but they were then installed in the same manner as for direct coal-firing, and although the boilers themselves made excellent records, the output of the furnaces to which they were attached fell 20 per cent below those that had no boilers. In the new setting, the idea has been to give free passage through the boiler, and ample down-take beyond, in order to facilitate the escape of gases as much as possible. Furnace No. 11, which was arranged in this manner, has given excellent results, the draft on each side of the boiler corresponding to 1.6 in. of water. The temperature of the gases at the inlet of the boiler was 2,380° F., while at the outlet it was 1,100° F. Allowing 34.5 lb. of water per horse-power-hour from and at 212° F., and feeding the water at 47.2° F., the boiler tested 340 h. p., which corresponded to a saving in coal alone of $70 per boiler per day. The steam pressure was 155.7 lb. on an average.

On account of more careful mixing and fluxing of the charge in the McDougal calciners, the tonnage smelted in the reverberatory furnace averaged 172.3 daily in January, 1904, with a coal ratio of 3.53 to 1. Early in 1904 No. 6 reverberatory was rebuilt with hearth 19 ft. by 59 ft. 1.5 in. This furnace now averages 207 tons daily, with a coal ratio of 4.33 to 1, developing 416 boiler h. p. steam.

As No. 6 was a success, No. 1 and 2 were dismantled, and a new No. 1 built with fire-box 7 by 16 ft., hearth 19 ft. by 83 ft. 2 in., and two Stirling 375-h. p. boilers set tandem. This furnace averages 241 tons daily, with a coal ratio of 4.53 to 1, and develops 710 boiler h. p., its record performance being 305 tons, with coal ratio 5.25 to 1. The next move was to replace the No. 7 and 8 furnaces with one called No. 4, with fire-box 7 by 16 ft., and hearth 19 ft. by 102 ft. 9 in., with two boilers tandem, as in No. 1. This furnace has not been in operation long enough to determine its capacity, but it has smelted 333 tons with a record coal ratio of 5.28 to 1, beside developing 720 boiler h. p. steam. The temperature of escaping gases was 590° F. An experimental jigging plant, to treat the ashes from No. 1 furnace, is in successful operation, saving 5.8 tons of fine coke daily (11 per cent ash), which is being briquetted with slimes from the concentrator, and fed into blast furnaces with a corresponding saving in fuel in that department. A jigging plant situated so as to take the ashes from all the reverberatory furnaces is now being installed.

MATTING OF ORES AT LEADVILLE AND ROBINSON, COLORADO.[1]

BY C. H. DOOLITTLE.[2]

In presenting these notes to the profession it is with the view of counteracting some ideas concerning capacity of furnace, upon which great stress has been laid during the past year in various articles appearing in *The Engineering and Mining Journal.*

The writer has visited some of the works possessing furnaces with such large capacities, and he has been employed at both matting and lead-smelting plants in this country and Mexico, so that he is frank enough to confess that no individual problem has been a harder one to solve than the one at Leadville, and its success was due in a great measure to its vice-president and general manager, the late Mr. Franklin Ballou. The two plants are now closed, the Bi-Metallic having been sold to the American Smelting and Refining Company, while the Robinson smelter is closed down, pending the further development of the mines. Leadville during the year 1900, the period to which these notes refer, was dull, with the market conditions such that suitable ores were difficult to obtain—copper sulphides very scarce and zincy iron sulphides predominating.

The tonnage of a furnace is due both to the physical and the chemical condition of the ores which are treated. The Leadville ores are naturally fine, and it would appear as if too much powder was used in mining, but the

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[1] *The Engineering and Mining Journal,* April 11, 1903, page 558.

[2] Late Superintendent of the Bi-Metallic Smelter, at Leadville, Colo., and the Robinson Construction Mining & Smelting Co., at Robinson, Colorado.

mine owner prefers to ship his ore in that condition, claiming that it helps accurate sampling. One-tenth sometimes, one-fifth more frequently, had to be crushed, rolled and sampled, and for a month at a time our flue-dust and slimes amounted to one-tenth of the total ore. It is likely that Mr. Herbert Lang must have had some experience of this kind, as he assumed that the Tennessee Copper Company was doing similar work, but very much to my surprise, on a visit to that company's works, the writer could find but little flue-dust, and, in fact, they were making less than 0.5 per cent, as nearly as could be figured without actual weighing.

I would like at this point to contradict a statement frequently heard, that "anybody can run a copper-matting furnace, but it takes a good man to run a lead furnace." I have had more sleepless nights over the former than the latter, and with a copper-matting furnace with an ore column of 6 to 7 ft., and the ores as rich in silver and gold as the average lead smelter, and rustling hard to get 1.5 per cent of copper in the charge, the superintendent must be up early and late, and not sleep far from the furnace.

The Bi-Metallic plant was unique in several features, and the system adopted was worked out after much hard labor and expense. The furnaces were three in number, having the dimensions, at the tuyeres, of 36 by 163 in., 36 by 175 in., and 36 by 215 in. The first two were used for concentrating the ore into a low-grade matte, the third and largest one for reconcentrating the matte with the addition of oxidized silicious ores. The air was furnished cold by three No. 7 Root blowers, and so connected to the furnaces that each furnace could have its individual blower, or they could be turned into the general main, and the blast drawn and subdivided for the various furnaces. The individual arrangement was found preferable. Two fans, one 9 ft. and one 6 ft.

diameter, were connected with the dust chamber; the gases, after having traveled 300 ft., were forced through towers, where they were sprayed, and a large part of the fume, rich in lead and silver, was precipitated and saved. The gases then escaping were damp and reduced to a temperature of 100° F., which necessitated a wooden stack. The power was furnished by a Corliss engine developing, by card measurement, 450 horse-power as its maximum load.

Having, by experience, found the capacity of the various furnaces for ore and matte, we endeavored to so run the furnaces that the matte produced by the ore furnaces was just sufficient for the reconcentration furnace, and the slag from the reconcentration furnace, which had to be re-smelted, was not too burdensome for the ore furnaces, but still sufficient for keeping an open charge in the ore furnaces. We thus endeavored to keep our capital out of large surpluses of low matte and high slag.

The characteristic ores treated are given in the following table:

Name of Mine.	SiO_2	Fe	CaO	Zn	Cu (Wet.)
Iron Silver	3.	41.	0.	6.	.1
Ibex M. Co..........	20.	28.	0.	6.	2.7
New Monarch	28.	18.	0.	11.	3.5
Marian	8.	38.	0.	1.	.5
Vinnie	23.	23.	0.	9.	3.
Commodore	85.	3.	2.	0.	0.
Centennial-Eureka ..	77.	5.	2.	2.	6.

During the month of March, 1900, the three furnaces ran 31 days each, and there was treated 9,838 tons of ore, carrying 2,256 oz. Au, 159,811 oz. Ag, 314,690 lb. Cu. This shows an average for the three furnaces of 105.8 tons of ore per furnace per day. The matte shipped averaged 2,249 oz. Au, 146 oz. Ag, 14,328 per cent copper. The recovery, 98.5 per cent Au, 95 per cent Ag, 90 per cent copper.

A charge for the ore furnace consisted of:

	Lb.
Ore (all sulphides)	2,600
Lime rock	250
Bricked flue-dust	300
Slag	1,500
Coke	325
Total	4,975

This shows a consumption of fuel of 12.5 per cent on the ore, but such was not really the case, as the works were supplied with a method of submerging the coke in water, and it carried 20 per cent moisture; hence the actual percentage of coke was 10 per cent.

The flue-dust was bricked and fed in sufficient quantities to keep that objectionable material (and necessarily a large one at this point) from accumulating. The slag specified above came from the re-concentration furnace, and was fed to keep this stock low and the charge open. Under ordinary conditions the ore furnaces averaged 120 tons of ore each. Great care had to be used in feeding the furnace, on account of the fine ore and the heating qualities due to the sulphur. A charge for the concentrating furnace consisted of:

	Lb.
Matte	1,000
Silicious ore	600
Lime rock	100
Wet coke	125
Total	1,825

This shows a consumption of fuel (dry) of 16 2-3 per cent. Taking the whole number of tons of ore treated for the month of March, 9,838, and the actual amount of fuel consumed, 1,417 tons, gives a fuel consumption per ton of ore of 14.4 per cent, which is a proper average for a year, as well as for this month.

The capacity of this furnace, for ore, was 75 tons per day. The average slag was: SiO_2, 36 per cent; Fe, 36 per cent; CaO, 7 per cent; Zn, 5 per cent. A little higher saving might have been made by the addition of more lime and using less iron; but from a commercial standpoint, lime cost us money, and a $9 rate, on a neutral basis, on iron ore, left a good margin for treating that class of ore. The average cost for the month showed $3.645 per ton of ore treated, including all expenses, except new construction. The matte was shipped to the Philadelphia Smelting and Refining Company, at Pueblo, and treated for $3.25 (freight and treatment), allowing us $19.25 per oz. for Au, 95 per cent of the Ag, 4c. off casting brand quotation for copper.

The Robinson Construction, Mining and Smelting Company had but one stack, though a second one was on the ground ready for erection. This is a 36 by 142 in. Loder hot-blast furnace. The hot blast consists simply of a series of pipes carrying the air in a chamber through which the escaping gases from the furnace pass on their way to the dust chamber. The blast is warmed, and the highest temperature obtained from numerous tests was 200° F., but even this is not to be despised if it can be obtained without unnecessary use of over-feeding fuel. During the times of re-concentration the furnace was always run with a cool top to avoid losses in precious metals; then the blast temperature would drop to about 90° F. The conditions at this plant were more favorable to smooth running than at Leadville. We mined our own ores and lime-rock, hence we were able to keep the physical condition of the ores such that slag was not a necessity to keep the charge open.

The chemical conditions were different, in that copper appeared in the ores we mined only as traces, and but a small amount of copper ore was obtainable on the market, and this ran but 4 per cent Cu, so our collector was practically an iron matte; but we made a good saving, which was due partly to the copper and more to a heavy fall of matte. The specific gravity of the slag was lightened by a higher percentage of lime than at Leadville. The market condition for the sale of matte had changed.

There was no competition, and but one purchaser for our matte. The price paid was $9.45 freight and treatment, 95 per cent of the Ag, $19 per ounce Au, 6c. off New York quotation for Cu. This condition necessitated a high concentration, of 30 into 1 as a final shipping matte. This was, of course, accomplished by reconcentration. The ores treated are given in the following table:

Name of Mine.	SiO_2	Fe	CaO	Cu
Washington	5.	42.	0.	Tr.
Robinson	25.	15.	1.	Tr.
Pride	64.	12.	3.	4.
Wintergreen	4.	48.	0.	1.

The iron was all a sulphide, FeS_2, except that from the Wintergreen, which was a pyrrhotite, Fe_7S_8.

The writer had been told by other metallurgists to beware of pyrrhotite in a raw state, but we experienced no difficulty with it, and as it carried 1 per cent of copper, we preferred to use it. A 24-hour run on ore used:

	Tons.
Washington	7.60
Robinson	81.00
Wintergreen	36.98
Total ore	125.58
Lime rock	25.65
Slag	55.50
Total charge	206.73
Coke	16.50
Per cent fuel on ore	13.14

Slag assay and analysis showed:

Ag	1.1 oz.
SiO_2	41. %
FeO	30.8 %
MnO	5.5 %
CaO	17. %
ZnO	4. %
	98.3

Re-smelting matte:

Ag	42.07 oz.
Cu	0.2 %

A 24-hour run on re-concentration:

Name of Mine.	Tons.
Robinson	116.50
Pride	28.42
Total ore	144.92
Matte	121.52
Lime rock	21.90
Tons of charge	288.34
Coke	14.60
Per cent fuel on ore	10.
Matte:	
Ag	200. oz.
Au	0.5 oz.
Cu	5. %

The average percentage of fuel used during the time the smelter was in operation was 13.5, being about 1 per cent less than the fuel consumption at Leadville. After smelting 3,289 tons of ore it was deemed best to close the works and make a clean-up. The flue-dust made was 2½ per cent on the ore treated. The cost of fuel and labor was $2 per ton of ore. Labor in this case does not include management, superintendence, etc.

The results, shown by the data given above, will undoubtedly seem small in tonnage to the copper metallurgist, but the ores are similar to those handled by lead metallurgists, and the tonnage fully equals that of the 42 by 146 in. lead furnace with a 20-ft. ore column and a 3-lb. blast.

All of the above enumerated ores carried some lead and zinc, and notwithstanding the volatilization of the major part of the lead, the resultant matte carried about 3 per cent of lead.

SMELTING AT MT. LYELL, TASMANIA.*

Since the beginning of 1902 a smaller number of furnaces have been kept employed for the same ore-consumption. At the present there are five furnaces in continuous operation, whereas the number formerly used to be seven and eight. The company possesses eleven blast furnaces, arranged in two smelting plants, the older, or No. 1 plant, containing six blast furnaces, and the newer plant, No. 2, five. The latter were built according to experience gained in the operation of the former, and were made considerably larger, the respective sizes, in the plane of tuyeres, being as follows: No. 1 plant, 40 in. by 168 in. (except No. 3, which is 36 in. by 126 in.); No. 2 plant, 42 in. by 210 in.; the height in each case, from the tapping to charge floor, is 20 ft. over all; the height of ore-column above tuyeres is 9½ ft.; the number of tuyeres (3 in. throughout) is 32 in No. 1 plant (24 in No. 3 furnace), and 40 in No. 2 plant. The furnaces are water-jacketed high up, the jackets all being cast iron, with the exception of the small steel-plate tymp-jacket. At the present time three of the large furnaces of No. 2 plant practically smelt into first matte all the ore that is delivered, while two furnaces of No. 1 plant are devoted to the concentration of the first matte into converter matte, and only occasionally treat ore alone. Under these circumstances, each furnace of No. 2 plant sometimes smelts up to 350 tons of Mt. Lyell pyrite per diem, and the average is about 270 tons of pyrite daily. Phenomenal tonnages are occasionally reached; thus No. 6 furnace recently treated a

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**The Engineering and Mining Journal,* May 30, 1903. Abstracted from the report of the Secretary for Mines, Tasmania. Prepared by the members of the staff of the Mt. Lyell Mining & Railway Company.

total of 724 tons of material (matte, silica, limestone and slag) in a single day. This vigorous work is due to the fact that the amount of air formerly distributed to the large number of furnaces is now supplied to the smaller number. Simultaneously, the blast pressure also has risen, until it is now on a level with the pressure made use of in the progressive matte-smelting establishments of the United States and British Columbia, and ranges from 36 to 40 oz. at the blowers, and from 30 to 34 oz. at the furnaces, the difference at the two points being loss by friction in hot-blast stoves, blast mains and furnace connections. To meet this higher pressure no alteration was made to the furnaces, except the raising of the sump; that is, the elevating of the overflow-level of the furnaces by several inches for the proper trapping of the blast, the same as before.

At the same time, important improvements were also made in the motive-power department by the installation, in each smelting plant, of steam-saving appliances, consisting of Green's economizers, with induced-draft fans and a system of steam superheaters. The economy effected by the economizers is determined as 15 per cent, and by the superheaters as 16 per cent, total saving 31 per cent. The waste heat of the fire-boxes of the hot-blast stoves, in addition to that of the boilers, contributes to this high result. The steam plant of the smelting works is, therefore, entirely up to date, and the various improvements have succeeded in lowering the cost of power to what must be regarded as the local minimum. The fuel employed is still chiefly firewood, the use of coal being merely auxiliary. The respective consumptions under the boilers were, for the year, 36,443 tons of wood and 1,794 tons of coal.

The work done in this department for the year was the highest yet performed. There are six 124-h.p. Babcock & Wilson boilers available in each smelting plant, two in each case now being in reserve. The average indicated horse-power was 1,905; total engine revolutions, 310,000,000; air delivered to furnaces, 31,000,000,000 cub. ft., weighing 1,054,000 tons. This air was heated to an average temperature of 580° F., thus absorbing 291,500,000,000 British thermal units out of the combustion of 49,000 tons of firewood in the hot-blast stoves. The latter have continued to give every satisfaction. In point of repairs, the eight stoves (four in No. 1 plant, with 56 cast-iron U tubes each, at 16 cwts., and four in No. 2, with 70 U tubes each) have given practically no trouble, there having been only about a dozen tubes completely replaced since the beginning, on account of burning out. The number of hands employed in the smelter motive-power department is 80, comprising engine-drivers, firemen, cleaners, wood-handlers, etc. It may be mentioned that, for the production of the blast, there are in use nine vertical compound condensing engines, 12 in. and 22 in. by 18 in., set up in marine style, and direct-coupled, by means of a flexible coupling, each to a No. 8 Root's blower of 116 cub. ft. displacement; also one reserve No. 7 Root blower, with vertical engine attached, ordinary style, and two reserve No. 7 Root blowers, belt driven from a horizontal tandem compound engine, 12 in. and 20 in. by 30 in. The power plant thus follows the unit system; that is, each furnace has a blower driven by an individual engine, although the blast is not conducted to each furnace separately, but directed into a common blast main. The condensers are of the surface type (1,000 sq. ft.), also with vertical compound engines 6 in. and 12 in. by 6 in., and air-pumps to suit—the latest with Edwardes' pump.

The following are the statistical figures of ore treated, rendered by the company for each quarter of the year 1901:

Treated.	Quarter ended Sept. 30, 1901.	Quarter ended Dec. 31, 1901.	Quarter ended Mar. 31, 1902.	Quarter ended June 30, 1902.	Total for year 1901-02
	Tons.	Tons.	Tons.	Tons.	Tons.
Mt. Lyell ore.......	69,289	65,688	71,111	79,969	286,057
Purchased ores	21,257	20,731	11,680	4,841	58,509
Metal-bearing fluxes.	5,726	6,480	7,702	8,139	28,047
Total	96,272	92,899	90,493	92,949	372,613

It is thus apparent that the average per day was the treatment of over 1,000 tons of raw mineral-bearing substances, besides which the company has treated in its blast furnaces all the first matte, converter slags and linings, flue-dust and similar middle products formed in the process.

The grade of the first matte does not exceed 15 per cent copper, being a concentration of from 6 to 7, and more, from the original ore into first matte, all of which work is accomplished, as stated, by the furnaces of No. 2 plant. The re-treatment and concentration of this first matte by the furnaces of No. 1 plant raises its grade to about 50 per cent copper, which is the standard required for proper economical bessemerizing into blister copper.

The output of the furnaces in copper, silver and gold, for the year under review, is as follows:

Output.	Quarter ended Sept. 30, 1901.	Quarter ended Dec. 31, 1901.	Quarter ended Mar. 31, 1902.	Quarter ended June 30, 1902.	Total for year 1901-02
Fine copper in blister (tons)	2,630	2,579	2,520	1,870	9,608
Silver, fine (oz.)....	164,932	160,102	182,487	171,133	678,654
Gold fine (oz.)......	5,224	5,002	5,650	5,555	21,451

The total money value of this year's output, calculated upon the changing quotations for the copper and silver, was $3,674,803.

The grand total of ore treated since the beginning of smelting operations on June 25, 1896, up to March 31, 1902, as given in the company's last report, together with average assays, is as follows:

AVERAGE ASSAY.

	Total tons.	Copper. %	Silver. oz.	Gold. oz.
Mt. Lyell Mine	1,160,684	3.00	2.81	0.101
Purchased ores	154,923	....	....	....
Total ores	1,315,607	....	....	....
Total metal-bearing fluxes	33,023	1.64	0.21	0.020
Grand total ores and metal-bearing fluxes	1,348,630	....	....	....

It is interesting to state that the average assay of all of the purchased ores, representing as it does (with the exception of a quantity of rich ore shipped to England by the North Mt. Lyell Company) an approximation to the average value of the silicious bornite ores of the district, is: Copper, 6.70 per cent; silver, 1.93 oz.; gold, 0.002 oz. The Mt. Lyell pyrite, therefore, though lower in copper, is relatively higher in silver and gold than the average bornite ore of the district. The aggregate average assay of all ores treated by the company is: Copper, 3.44 per cent; silver, 2.70 oz., and gold, 0.090 ounces.

The following were the last half-yearly average analyses of the ores purchased and smelted:

	Silica. %	Iron. %	Alumina. %
North Mt. Lyell	63.70	6.22	10.10
Lyell Tharsis	64.00	5.08	17.25
Mt. Lyell Blocks	60.91	7.91	10.60

The method of metallurgical treatment has remained the same as at the start, direct pyritic smelting, with the use of only a very small percentage of carbonaceous fuel, and with a heated blast, and subsequent bessemerizing after Manhes. The type and details of the blast furnaces have required no alteration from the original design. The use of a hot blast for the furnace treatment has been proved by experience to be obligatory and not to be circumvented, as indeed is demonstrated by a careful investigation of the thermo-chemistry of the method. The process, as a whole, is divided into three stages, *i. e.*, two smeltings into matte, and the converting of the enriched matte into blister copper.

The quantity of coke used in the blast-furnace smelting, when calculated on the ore, is about 5 per cent in current work for the two-fold furnacing, and 6 per cent on the ore, including all incidental coke for blowing-in purposes and similar occasional use, while, on all the material fed into the furnaces (less coke itself), the percentage is only 3.25 per cent for the double smelting. It can hardly be maintained that so small an addition of coke has any vital bearing on the furnace reactions, from a heat point of view, and it is commonly remarked that the coke simply serves to keep the tuyeres from becoming too hard. It is fed along the walls of the furnaces with this special object in view. Such low percentages of coke are possible only because of the application of the oxygen of the atmosphere to the ore mixture under conditions which make the oxygen perform a function somewhat different in its effect from that which it exercises in ordinary blast-furnace smelting, where a larger percentage of carbonaceous fuel is necessary. In all three stages of the process the material (ore or matte) is constantly subjected to an oxidizing action, and not a reducing one. The process, therefore, in this respect is the direct inverse of ordinary smelting operations in blast furnaces. A potent factor establishing the feasibility of the furnace smelting is the presence of a heavy percentage of iron and sulphur in the Mt. Lyell pyrites, which elements yield a sufficient quantity of heat, during their combustion or union with the oxygen of the atmosphere, to allow the solid products of combustion to remain in a superheated molten condition. This peculiarity also distinguishes the bessemerizing operation. In fact, it may be said that the Mt. Lyell ore-smelting is nothing more than a bessemerizing of ores, or matte therefrom, direct in blast furnaces, but so managed as to be continuous, and not intermittent, as is the bessemerizing operation, and also under perfect control, both as to the composition of slags and mattes, and the proper mechanical separation of these two products. The company remains to this day the only instance of this idea being utilized, on a large scale, as the exclusive method of treatment. The entire length of time occupied by the transition of the copper from the ore into blister copper—that is, from the moment the ore enters the first furnace until the blister copper issues from the converters—is only from five to six hours, if the time lost through cooling and transportation between apparatus is disregarded. An essential condition of success is rapidity of treatment, for it is only by this means that the necessary heat is concentrated to the required pitch.

The blast-furnace slags vary between the following limits of composition, the furnace charges for ore smelting and matte concentration differing somewhat from each other. Limestone is used only in the latter operation, and not on the ore. The first matte is resmelted by itself, as a rule; only very exceptionally has pyritic ore been used with it.

	Per cent.		Per cent.
Silica	36.66	to	41.70
Iron protoxide	50.67	to	43.14
Calcium oxide	1.20	to	8.16
Barium oxide	1.90	to	0.16
Alumina	7.47	to	5.46
Copper	0.25	to	0.35

The flue-dust is ground up and mixed in a steam-driven mud-mill, with about 4 per cent of common clay, and the addition of a little water, and no lime is used for binding. The briquettes are of the size of ordinary bricks, are made by hand in preference to presses, and weigh about 14 lb. each. They are laid on shelves in a large drying room, and rough-dried by the heat of pots of slag ranged alongside. The quantity of flue-dust so handled was 4,666 tons for the year. Its average assay was: Copper, 3.5 per cent; silver, 2.5 oz.; gold, 0.07 oz. It contains about one-third silica, which comes from the dust and sand accompanying the quartz flux. More elaborate means of preparing the flue-dust, like fusion in a reverberatory furnace, moulding and burning in closed kilns, as well as a briquetting machine of good repute, have been given lengthy trials, but have all been discarded on account of being too expensive, and preference was finally given to the present means, which is primitive in character, but more satisfactory in regard to cost and efficiency. The flue-dust is partly oxidized, but there are not sufficient sulphates in it to act as binding material for machine briquetting, and the cost of the obligatory lime has decided against the method.

The system of getting rid of the slag by granulation has been in use since the start, but of late years it has become necessary to resort to mechanically raising the accumulating dump of each smelting plant to a higher level, to keep the talus of the dump within bounds. This is effected by centrifugal pumps of a special construction—the outcome of the simplification of prevalent types and adaptation to the special work to be performed. The pumps are at present steam-driven, but will shortly be run by electricity. The slag pumps are of small diameter, 10-in. suction by 9-in. discharge, with 18-in. beaters or vanes, and are run at from 540 to 600 revolutions per minute. The quantity of slag elevated during the year was 170,000 tons at No. 2 plant and 40,000 tons at No. 1 plant, together with all the water entering the two smelting plants, with the exception of that used for steam generation, sprinkling, etc. The respective lifts are 15 and 11 ft. at present.

The new sampling works, which were erected in place of the original mill, which was destroyed by fire a couple of years ago, treats the samples of all ores and mineral fluxes, also purchase ores used, and prepares them for the assay office. The proportions taken in transit and delivered to the sampling works are: Every 25th ropeway bucket of open-cut pyrites, every 16th truck of underground ore, every 18th truck of mineral flux, while the proportion on purchased ores varies from all to $\frac{1}{4}$ to $\frac{1}{8}$th. The balance of lots goes direct to the furnaces; none of the ores, etc., are specially crushed, but are smelted in the size as broken at the mines.

Converter Plant.—The converting operations are regulated by the amount of matte produced in the blast furnaces. The bessemerizing plant is a double one; that is, it consists of two independent equal-sized departments, each with facilities for turning out from 25 to 40 tons of copper a day. The total number of vessels is 14 (12 in use), vessel stands, 6; and there are two remelting furnaces with a No. 5 rotary blower each, with vertical engine attached; two high-class Fraser & Chalmers horizontal compound condensing blowing engines, 16 and 24 by 30 in., with air-cylinders 30 in. by 30 in., delivering 3,000 cub. ft. of free air at 60 revolutions; a 14 by $3\frac{1}{2}$ by 10 in. duplex Worthington high-pressure pump; an accumulator; and all other necessary paraphernalia for the handling and tilting of the vessels, including hydraulic cars for shifting same, traveling crane, etc., together with the necessary machinery, grinding pans, etc., for the preparation of the vessel linings; also drilling machines for drilling the blister copper for sample. Steam is supplied by four 70-h.p. multitubular boilers. In the motive-power department the average indicated horse-power developed for the year was 300, and the amount of wood fuel consumed 13,043 tons. The average blast-pressure is 8 lb. per sq. in. at the vessels.

The clay for the vessel linings is a fine white variety of refractory nature, won in the immediate vicinity of the converter plant, and the silica is supplied by the screened fine or refuse from the quartz-crushing plant at the quarries. In these important respects the company is therefore fortunately situated. The composition of the clay is as follows:

	Per cent.
Silica	62.52
Alumina	23.89
Iron oxide	0.26
Calcium oxide	0 25
Magnesium oxide	0.40
Water and undetermined	12.68
	100.00

The number of 'blows' which a lining will stand, until corroded so thin that the vessel has to be removed, is from five to seven.

This portion of the reduction works is also unique, for it is the only case of bessemer works for copper in Australia. The first converter was started on January 14, 1897. The style of vessel is that known as the Stalmann type, which has proved satisfactory; it is slight in repairs, a fact due largely to the square shape of the vessels. The tilting-gear is simple and effective, the rack being on the cylinder itself, which travels horizontally on the hollowed piston-rod, through which the pressure water enters and leaves.

The production of the bessemerizing plant for the four quarters of the year is given above, under the output of the furnaces in metals. Since the beginning of operations to the end of the company's last half year (March 31) the total results are as follows: Blister copper, 40,168 tons, of an average assay per ton of: Copper, 98.83 per cent; silver, 81.36 oz.; gold, 3.124 oz. The total content was: Copper, 39,698 tons; silver, 3,268,016 oz.; gold, 125,498 ounces.

Adding on the production for the last quarter, the figures become the following, from the commencement of work to July 1, 1902: Blister copper, 42,060 tons, containing: Copper, 41,568 tons; silver, 3,439,149 oz.; gold, 131,053 ounces.

The quantity of converter matte treated by the converters from the beginning to March 31, is 87,229 tons, of an average assay of: Copper, 49.48 per cent; silver, 39.34 oz.; and gold 1.480 ounces.

The blister copper is now poured into plates, or cakes, measuring 16 in. by 24 in. and 2¼ in. thick, the former bars or pigs having been abandoned. Five hundred plates constitute a lot or parcel of 50 tons, the sample of which is kept distinct. Sampling is done by drilling two ⅜-in. holes through each plate, with power-driven drill presses. The blister copper is all shipped to the United States, via London, and is refined by the Baltimore Copper Smelting and Rolling Company in Maryland, on a toll arrangement, the products being turned over to the company. The products, *i. e.*, electrolytic copper, as ingots, wire-bars, etc., on the one hand, and fine silver and fine gold on the other, are sold by the company, the two precious metals being disposed of at the United States Mints, and the copper in the open market. It is chiefly exported to Europe, its identity being merged in that of the well-known brand "B.E.R."

A note from Mr. Alfred Miller, appended to the article above abstracted, gives some additional information. It says that, since the article was written, important changes have been effected in the smelting method by bringing it up to what may be called "ideal pyritic smelting." Ore smelting in No. 2 plant has been continued since the middle of November last, without the use of coke (but with heated blast, although of a lower temperature than that formerly used), and at No. 1 plant both ore and matte have been smelted without the use of the hot blast (but with the former addition of a small percentage of coke). These innovations have been attended with no trouble whatsoever, and may now be considered permanently established, the results being favorable, and tending toward an increase of furnace capacity. Needless to say, they effect a considerable annual saving, and it is expected that still further improvements and economies will be achievable in the course of time. Four furnaces are now doing the work referred to in the article as being done by five, three of the four being constantly on ore, and the fourth concentrating the first matte into converter matte.

The treatment costs for the company's half year, ended September 30, 1902, were as follows (reduced to United States currency): Mining, $0.5002; removal of overburden, $0.50; smelting, $3.3648; converting, $0.3612; total, $4.7262.

During the half year mentioned there were treated 159,450 tons of Mount Lyell ore, of an average value of: Copper, 2.36 per cent; silver, 2.23 oz., and gold, 0.069 oz.; together with 18,537 tons of metal-bearing fluxes, of an average assay value of: Copper, 1.70 per cent; silver, 0.24 oz.; and gold, 0.026 oz.; and 5,689 tons of purchased ores, making the grand total for the half year, 183,676 tons, and raising the grand total since the beginning to 1,532,306 tons of ore.

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