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THE SMELTING OF COPPER ORES IN THE ELECTRIC FURNACE

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THE SMELTING OF COPPER ORES IN THE ELECTRIC FURNACE.

By Dorsey A. Lyon and Robert M. Keeney.

INTRODUCTION.

This bulletin is one of a series dealing with the application of the electric furnace to the smelting of ores and the manufacture of alloys, and is published by the Bureau of Mines in the endeavor to increase efficiency in metallurgical processes.

The bulletin presents, first, a critical discussion of the possibility of smelting copper ores in the electric furnace; second, the results of the experimental work of other investigators on the electric smelting of copper; third, the results of experiments by the authors on the electric smelting of native copper concentrates and sulphide copper ores; and, fourth, a comparison of the electric furnace with the blast furnace and reverberatory furnace for copper smelting.

Before considering the electric smelting of copper ores the reader should understand that the electric furnace was not developed as a competitor of the combustion furnace, but for the purpose of doing high-temperature work that can not be done in the combustion furnace, and for the treatment of ores from deposits that are in regions where fuel is scarce and costly but hydroelectric power is comparatively cheap, as in Chile, and in certain parts of Canada, the United States, and Mexico.

Furthermore, it is not the object of this report to try to prove that the electric furnace should replace the reverberatory or the blast furnace, as used at present in smelting copper ores, but to show that the electric furnace may be used with advantage in localities where conditions are not favorable to the use of the reverberatory or the blast furnace.

For example, otherwise valuable ore deposits may be situated in a district so remote from, or inaccessible by, railway that the cost of getting coke for smelting, or of transporting the ore to a smelter, is prohibitive. If there is sufficient water power at hand, from which electric power can be developed at a reasonable cost, and this electric power is used for smelting the ore in an electric furnace, with subsequent bessemerizing, if necessary, the concentrated product may be valuable enough to render treatment of the ore possible. Or, the deposits may be accessible by railroad and yet the cost of fuel may be too great for smelting by the ordinary methods, whereas, on the other hand, hydroelectric power can be developed in the district at a low cost. By the use of the electric furnace such ores may be smelted cheaply enough to make their treatment feasible.

The possibilities being as stated, a brief comparative study of the problem is presented for the purpose of aiding those interested in the subject in determining whether it would be possible metallurgically, and feasible commercially, to use the electric furnace in those localities where, by reason of costly fuel, the cost of operating a reverberatory or blast furnace for smelting copper ores is either excessive or prohibitive.

The main points to be considered regarding the electric smelting of copper ores are whether the ores can be treated as successfully in an electric furnace as in a combustion furnace, and if so, whether they can be smelted at a reasonable cost.

CHEMISTRY OF COPPER SMELTING.

The objects sought in copper smelting may be summarized as follows: (1) To concentrate the copper into metal carrying any precious metals that may be present, if a native copper or an oxide ore is to be smelted, or into a matte, if a sulphide ore is to be treated, and (2) to remove, as slag, the excess of iron and gangue materials.

In general, there are three classes of ores that have to be considered in copper smelting, namely, native copper, the oxides, silicates, and carbonates, and sulphide ores.

SMELTING OF NATIVE COPPER ORES.

From the manner of occurrence of native copper, the smelting of native copper, or a native copper concentrate, resolves itself into the melting of the ore, or concentrate, and the separation of the metallic copper from the gangue through the difference in their specific gravities. After removal of the slag a further refining of the black copper obtained from the melting is necessary. The smelting of native ore is the simplest form of copper smelting. The electric furnace is especially adapted to simple melting in a neutral atmosphere, and appears to have particular application in this phase of copper smelting, especially the smelting of fine concentrates, as shown by recent experiments of the writers.

SMELTING OF OXIDE, CARBONATE, OR SILICATE ORES.

The process of smelting oxide, carbonate, or silicate ores of copper consists simply in melting the ores in the blast furnace with coke, thus reducing them to metallic copper. The separation of the slag and metal is an important feature of the process and the supply of coke must be carefully regulated to prevent reduction of iron by any excess carbon. The furnace atmosphere is reducing and not neutral. Such an atmosphere is easily obtainable in the electric furnace, and it is used in the electric smelting of iron. The experiments of Stephan, to be described later, also show the possibility of using a reducing atmosphere in the electric furnace.

SMELTING OF SULPHIDE ORES.

The smelting of sulphide ores presents additional complications. In order to concentrate the copper and any precious metals into a matte, it is first necessary to eliminate from the ore the sulphur, or as much of it as may be necessary, in order to obtain the desired results, and to cause the resultant oxides to unite with silica, which may either be present in the ore or may be added, preferably in the form of siliceous ore, to make a slag.

REMOVAL OF SULPHUR.

As pointed out by Peters,[a] when fused in a reducing atmosphere certain of the sulphides, such as Cu_2S, FeS, PbS, etc., melt down without change, while the very common and important mineral, pyrite, loses about one half of its sulphur, which is driven off as sulphur in the shape of yellowish fumes, which will deposit a coating of brimstone on a cool surface.

Peters also states that—

> The only two sulphides of much interest to the metallurgist, which lose a portion of their sulphur by heat alone in the manner just described, are (1) pyrite (FeS_2), which is assumed to lose one-half of its contents in sulphur by mere heating without air; and (2) chalcopyrite ($Cu_2S.Fe_2S_3$) which (as generally assumed by metallurgists) is composed of one-third copper, one-third iron, and one-third sulphur, and loses (by melting without air) one-third of its sulphur, or one-ninth of its entire weight.

This quotation is given in order to show clearly and concisely that there would be no object in simply melting sulphide ores in an electric furnace. That is, if this were done, only an altered form of the ore would result, somewhat concentrated, to be sure, through the loss of the elemental sulphur, but not concentrated enough.

RESULTS OF SMELTING RAW SULPHIDE ORE IN A BLAST FURNACE.

Peters gives the following example in order to illustrate the concentration effected by smelting in a blast furnace:

> Assume that you wish to smelt 100 pounds of a sulphide ore containing no earthy gangue rock, and consisting solely of a mixture of pyrite, chalcopyrite, and a little galena. It assays 35 ounces silver per ton and carries 6 per cent copper and 4 per cent lead. Neglect for the moment the inevitable metal losses in smelting, and assume (as the practical metallurgist generally does) that the pyrite consists of one-half

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 109.

iron and one-half sulphur, and that chalcopyrite consists of one-third each of copper, iron, and sulphur.

He then makes the necessary calculations to determine the weight of the matte that would be produced if this ore were to be smelted in a blast furnace, and from the results obtained constructs the following table:

Theoretical weight of matte produced by smelting 100 pounds of mixed sulphides.

Mineralogical composition of ore.	Weight of ore.	Chemical composition of ore.				Loss by smelting (S).	Remaining to form matte.				Total weight of matte.
		Fe.	Cu.	S.	Pb.		Fe.	Cu.	S.	Pb.	
	Lbs.	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*
Pyrite	77.36	38.68	....	38.68	......	19.34	38.68	......	19.34	......	58.02
Chalcopyrite	18.00	6.00	6.00	6.00	......	2.00	6.00	6.00	4.00	......	16.00
Galena	4.64	....	....	.64	4.00	....	....	......	.64	4.00	4.64
Total	100.00	....	....	....	......	21.34	44.68	6.00	23.98	4.00	78.66

As stated by Peters, this calculation shows that if 100 pounds of mixed sulphides were to be smelted in the blast furnace the matte would weigh 78.66 pounds, or a concentration of $100 \div 78.66 = 1.27$ would be obtained, and only 1.27 tons of ore would be smelted into 1 ton of matte, which is an unsatisfactory and unprofitable degree of concentration.

METHODS OF SMELTING SULPHIDE ORES.

Such being the case, the smelting of sulphide copper ores resolves itself into one of the three following methods, or a combination of them, in order to effect the removal of sulphur and a desirable concentration:

(1) Oxides and sulphides are mixed in such proportions that the reactions necessary to effect the desired degree of concentration will take place.

(2) Sulphide ores are roasted, thus converting them into oxides, or partly so, depending on the degree to which the roasting is carried out, and are then melted in a suitable furnace. By the preliminary removal of sulphur to the desired degree, the excess iron and the gangue materials present may then be eliminated in the furnace as slag, for it is to be remembered that sulphur which is present in the charge during the melting will combine with the copper, and that any sulphur in excess of the amount necessary to satisfy the copper will unite with the iron; the remainder of the iron present as iron oxide (FeO) will unite with silica to form slag. The concentration of the copper and precious metals into a matte is thus brought about.

(3) As roasting the ore is an additional step in the process of concentration it is to be avoided if possible, and the desired oxidation of the sulphides accomplished in the blast furnace by means of the

oxygen of the air that is forced into the furnace through the tuyères. This is what the modern copper blast furnace does; it combines the functions of roasting and smelting. In other words, it oxidizes and smelts simultaneously, and also utilizes the heat that is generated by the oxidation of the sulphur and iron. This utilization of the heat generated by the oxidation of the iron and sulphur is termed pyritic smelting, although, as will be shown later, true pyritic smelting involves the smelting of sulphide ores solely by the heat arising from their own oxidation, without the addition of any carbonaceous fuel.

REACTIONS IN THE BLAST FURNACE.

In order to illustrate what takes place in a blast furnace smelting sulphide ores, assume that the charge is made up of silica, pyrite, and chalcopyrite. The shaft may be divided into two zones, namely, the preheating zone, and the oxidation and fusion zone, it being understood, of course, that there is no sharp line of demarcation between the two.

Reactions in preheating zone.—The behavior of the main constituents of the charge in passing through the preheating zone is as follows:

(1) Silica (SiO_2) remains unaltered.

(2) Chalcopyrite ($Cu_2S.Fe_2S_3$) loses about one-fourth of its sulphur as elemental sulphur and becomes a mixture of Cu_2S and FeS.

(3) Pyrite (FeS_2) loses about one-half of its sulphur and becomes FeS and then, finally, Fe.FeS.

Reactions in oxidation and fusion zone.—The reactions that take place in the oxidation and fusion zone are as follows:

(1) Chalcopyrite (Cu_2S and FeS compound) melts as soon as it has descended to a point where the temperature is high enough, a comparatively moderate temperature.

(2) The sulphur in the pyrite (Fe.FeS compound) is partly burned by the oxygen of the blast to SO_2. That which remains unburned from lack of oxygen passes through the oxidizing zone and enters the matte.

REVERBERATORY COPPER SMELTING.

The objects sought and the methods used in the smelting of copper ores have been outlined. The next point to be considered is the manner in which the smelting of copper ores is conducted.

Inasmuch as copper ores are smelted in one of two ways—that is, either in a reverberatory or in a blast furnace—it will first be necessary to ascertain if the conditions essential for the successful conduct of these two methods can be attained in the electric furnace.

As stated by Peters,[a] "the one fundamental and preeminent duty of the reverberatory type of furnace is to produce a smelting temperature so that the ore may become liquid, and thus be able to separate

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 170.

into slag and matte." As regards this discussion, the reverberatory furnace is used for the smelting of ores and concentrates that can not be successfully treated in a blast furnace. As Peters has shown:

> The manner in which the heat from the fuel is applied in the reverberatory method of smelting, though much improved in modern practice, is a peculiarly wasteful one, as the mere rapid passing of a flame over the surface of a layer of ore resting upon a comparatively cool hearth is a particularly incomplete way of transferring heat from the flame to the ore particles, especially as the latter are usually poor conductors of heat.
>
> The very large surface of walls, arch, flue, stack lining, etc., is in just as favorable a position for being smelted as is the ore itself, and in spite of its refractory composition, needs frequent repairs and renewals.
>
> Such a method could only originate in a district possessing cheap flaming coal and inexpensive refractory materials, and Swansea (where the process originated) has these in abundance.
>
> The reverberatory process is but little complicated by the active extraneous agents which exert so marked an effect upon the chemistry of the blast-furnace smelting of roasted ores, and the pyrite smelting of sulphide ores, namely, carbon in the one case and oxygen in the other.
>
> Speaking in a broad general sense the ore in the reverberatory smelting furnace is subjected to neither of these influences to any marked degree, and it is thus enabled to work out its own salvation under the influence of heat alone. Such chemical reaction as takes place in the ore mass during the smelting results almost entirely from the behavior of substances already contained in the ore and is affected by the mutual action of the various constituents of the charge itself. This renders the chemical part of the operation unusually simple and enables the smelter to concentrate his efforts upon developing the high temperature essential to the complete fusion of the ore.

SUBSTITUTION OF THE ELECTRIC FURNACE FOR THE REVERBERATORY FURNACE.

If, then, as Peters states, "the proper function of the reverberatory furnace, in smelting copper ores, is to generate heat as rapidly as possible," there is no apparent reason why copper ores can not be smelted equally well in an electric furnace, and perhaps even more advantageously; for, as pointed out by Peters, and as is well known, the thermal efficiency of the average reverberatory furnace is only 5 to 8 per cent, although an efficiency of about 20 per cent is obtained in the large reverberatory furnaces at Anaconda, whereas an electric furnace for smelting copper ores that are suitable for reverberatory smelting would probably have an efficiency of 70 per cent. Chemically, the only possible disadvantage in the use of such a furnace would be the effect of the carbon of the electrodes upon the iron oxide in the charge with which the electrodes might come in contact, and which would result in (1) the loss of electrode; (2) the production of metallic iron, which, if metallic copper was desired, would cause a low-grade product, and increase the cost of refining, or if a matte was desired might necessitate reconcentration; and (3) the loss of electrical energy, due to the additional heat consumed if there were much reduction of iron by the carbon of the electrodes.

As the reverberatory furnace, whether used for smelting sulphide, oxidized, or native copper ores, is essentially a means of melting, for comparison with the reverberatory, the results obtained in the smelting of native copper ores in the electric furnace will serve to clear up for sulphide or oxide ores the three points above mentioned—namely, the loss of electrodes, the production of metallic iron, and the loss of electric energy.

The slags would be somewhat similar in each case. The atmosphere of an electric smelting furnace charged with sulphide ore would not be as neutral as that of an electric furnace charged with native copper ore, because the liberation of sulphur and the formation of sulphur dioxide tend to dilute the air and to reduce its oxygen content, making it sufficiently reducing to prevent the consumption of the electrodes by oxidation. If the electric furnace was not kept closed, considerable sulphur dioxide might be formed, which would possibly be reduced in part by the carbon of the electrodes. Even with any reduction of sulphur dioxide the electrode consumption would be probably less for sulphide ores than for native copper ores. The results of experiments by the authors confirm this statement. In a furnace charged with native copper concentrates of the grade given in Table 2, the maximum electrode consumption in 14 experiments was 48 pounds of graphitized carbon per ton of charge, the minimum was 22.1 pounds, and the average was 34.5 pounds. In the same furnace, with mixtures of sulphide, oxide, and siliceous ores, a matte being the product, the maximum electrode consumption for 20 runs was 60 pounds, the minimum was 10.8 pounds, and the average was 23.3 pounds.

SMELTING OF NATIVE COPPER ORES.

METALLURGICAL PRACTICE AT MICHIGAN SMELTERS.

Copper smelting practice in Michigan differs considerably from that of smelters in the Western States, in that instead of the production of matte and eventually blister copper from sulphide ore, copper [a] is produced directly by melting concentrates and masses containing native metallic copper. The black copper obtained is subsequently refined, and the slags obtained during the operation are resmelted.

CONCENTRATION METHODS.

The concentrates are obtained from the concentration of conglomerate or amygdaloid ores. Before concentration these ores contain from 1 to 3 per cent of copper and some native silver. The amygdaloid is high in silica, whereas the conglomerate contains more iron than silica. The milling treatment preliminary to smelting consists

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[a] Conant, H. D., Copper smelting in Michigan: School of Mines Quart., vol. 32, 1911, p. 285; Canadian Engineer, vol. 21, 1911, p. 246.

of crushing with breakers and steam stamps or Hardinge mills and concentration on jigs and tables until a coarse concentrate is obtained containing 60 to 80 per cent copper. Most of the coarse native silver is removed in concentration. Large masses of copper are cleaned of gangue and smelted directly. At one smelter these concentrates were separated into three grades:[a] No. 0, mass copper, containing from 65 to 70 per cent copper; No. 1, coarse concentrate and some mortar discharge, averaging about 75 per cent copper; and No. 2, fine concentrate, containing 10 to 35 per cent copper. This research was conducted on material resembling the No. 2 grade.

SMELTING PROCEDURE.

There are two general methods of procedure in smelting these concentrates. In one method the mass copper and coarse concentrate are charged into a reverberatory furnace and melted down without any fluxes; the slag is skimmed off as fast as it is formed and the copper tapped into another reverberatory furnace for refining. In the second method melting and refining are carried on in the same furnace. Formerly the fine concentrate, No. 2, was mixed with lime and briquetted for smelting in the blast furnace, but now at some smelters this material is first agglomerated in a reverberatory furnace and later smelted in the blast furnace.

Up to 1904 the largest reverberatory melting furnaces in operation in the district had a hearth area of 17 by 28 feet and a capacity of 200,000 pounds per charge, which, because of the cycle of operation, is practically the capacity for 24 hours.[b] At present furnaces are in use having a hearth area of 16 feet by 35 feet and a capacity of 300,000 pounds per charge. The largest reverberatory refining furnaces have a hearth 13 feet wide by 17 feet long.

METHOD OF OPERATING REVERBERATORY FURNACE.

The cycle of operation in smelting native ore is as follows:

A reverberatory furnace is charged, say, at 4 p. m., with concentrates and mass copper. After slag begins to form the charge is skimmed every 2 hours for a period of 15 hours, after which the bath is oxidized by blowing air on it or into it with an iron pipe as a carrier or tuyère. This oxidation removes arsenic and other impurities and oxidizes some of the copper to suboxide of copper, Cu_2O. The molten bath at this time contains about 95 per cent of metallic copper and about 5 per cent of cuprous oxide. The oxide is then reduced by poling with green wood and charcoal. Casting is commenced about 1 p. m. and finished about 3 p. m. The furnace is repaired if necessary and is ready for the next charge at 4 p. m.

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[a] The western metallurgical field; copper smelting in Michigan: Met. and Chem. Eng., vol. 11, 1913, p. 12.

[b] Austin, L. S., Recent copper smelting at Lake Superior: Eng. and Min. Jour., vol. 81, Jan. 13, 1906, p. 83.

The completion of the process of oxidizing with air is determined by the electrical conductivity and fracture of a test piece. Copper refined from native copper concentrates must be coarsely crystalline to have a conductivity of 100 per cent, whereas electrolytic copper of 100 per cent conductivity may be only finely granular. Poling is complete when a test button of the metal sets with a level surface.

TREATMENT OF SLAGS FROM FURNACE.

The slags from the reverberatory furnace have a very high copper content, but if an attempt is made to reduce it the purity of the product is impaired. The slags have to be retreated in any case, so the percentage of copper in them makes little difference. Reverberatory slags contain from 10 to 30 per cent copper, the content being higher toward the end of the skimming process. Slag from amygdaloid ore is acid and contains 40 per cent SiO_2 and 20 per cent FeO, whereas slag from conglomerate ore is basic and contains 25 per cent SiO_2 and 45 per cent FeO.

These reverberatory slags are remelted in a low-pressure blast furnace with anthracite coal or coke as reducing agent and fuel, and limestone, iron oxide, or silica as a flux. The furnace is run with a slag containing 33 per cent SiO_2, 33 per cent FeO, and 20 per cent CaO, which is a fusible and liquid slag. The slag runs continuously from the furnace and contains 0.6 to 0.8 per cent copper. If the production of a cleaner slag is attempted the iron content of the product is increased, owing to the extra reducing material and higher temperature used. The metal produced contains 90 to 95 per cent copper and is retreated in a reverberatory furnace.

The blast furnace is rectangular in cross section, being 40 to 48 inches wide at the tuyères and 8 to 12 feet long. The crucible is lined with fire brick incased in a steel shell, and the shaft, which is 7 to 10 feet high, consists of steel water jackets. No forehearth is used. A blast pressure of 3 to 4 ounces is used with a production of about 72 tons of copper per 24 hours. With a higher pressure there is greater loss of copper in the slag, owing to more rapid driving of the furnace.

The refined product from the reverberatory furnace is cast into various sizes and shapes of ingots for marketing, but if it contains much silver it is cast into anodes for electrolytic refining.

EXPERIMENTAL SMELTING OF FINE MICHIGAN NATIVE COPPER CONCENTRATES IN THE ELECTRIC FURNACE.

For the purpose of investigating the feasibility of smelting fine Michigan native copper concentrates in the electric furnace instead of the reverberatory furnace, the following experiments were conducted by the writers.

The objects of these experiments were: (1) To note the effect of variation of the proportions of the three main slag constituents—silica, ferrous oxide, and lime—on the power consumption, the operation of the furnace, and the purity of the copper produced; (2) to ascertain the extent of iron oxide reduction caused by the graphite electrode; (3) to compare the results of intermittent smelting with those of continuous smelting; and (4) to study the feasibility of using the slag produced in copper smelting for the manufacture of ferrosilicon in the electric furnace.

GENERAL DESCRIPTION OF EXPERIMENTS.

The experiments were divided in six series, based on different variables in the slag-forming components of the charge. Fourteen experiments were made in all, grouped as follows: In series 1, the slag was formed from the natural gangue of the concentrate; in series 2, the acidity of the slag was varied by the addition of silica; in series 3, the basicity of the slag was varied by the addition of Lake Superior hematite; in series 4, lime was used instead of iron ore as a base; in series 5, the basicity of the slag was varied by the addition of lime and hematite; and in series 6, the furnace was charged at intervals and kept full during the run, and the metal tapped at intervals.

In all of the experiments the furnace was operated as a resistance furnace as far as possible in order to keep down the volatilization of the copper. There was arcking at the start of a run before the charge became molten, and in several runs arcking occurred at intervals during the experiment, but not for a long period of time. In all experiments, except those of series 6, the furnace was charged intermittently—that is, after starting the charge was added gradually—and all of it was completely melted before tapping. As far as possible, the furnace was tapped cleanly after each run. Any slag left in the furnace after tapping was removed by balling in the furnace in a manner similar to that used in the puddling process. Slag adhering to the walls was allowed to remain there.

In the operation of an experimental electric furnace, except on a very small scale, where tapping can not be done, it is not advisable to go to extremes to obtain an absolutely clean discharge of the molten products. To remove entirely all metal and slag, especially slag, one of two procedures must be followed: Either the charge, after becoming molten, is superheated to remelt any material that has adhered to the cooler parts of the walls, or, when the furnace has cooled off, this material is chiseled off. Either procedure results in a partial or complete destruction of the furnace lining, making a new one necessary. If the charge is superheated, the high temperature results in the brick walls becoming pasty and gradual dripping of the softened brick into the slag. One brick may be more exposed than the others and gradually be completely melted, and, aside from causing injury to the lining, materially alter the composition of the slag.

The prolonged heating may also cause volatilization of the copper, although this is not so likely to occur after the copper has been smelted down and covered with a bath of slag, as during the melting period. The prolonged heating also caused a much higher power consumption than would occur in practice, as in practice the accretions and the hangings would not affect this item after the furnace was running regularly and continuously. Chiseling out the furnace is impractical, because of the time consumed and on account of the destruction of the lining. For these reasons the furnace was built with a high degree of slope on the bottom toward the tap hole, and tapped as cleanly as possible through this hole. As a result, some of the runs show a considerable deficiency in the quantity of metal tapped as compared with the calculated amount in the charge, and some an excess; although the greatest discrepancy is in the slag. In some runs the amount of material tapped checks very closely with that charged.

DESCRIPTION OF FURNACE USED.

An electric furnace of the Siemans type is the best suited for laboratory experimental smelting of ores. A furnace of this type, as shown in figures 1 and 2, was used in these experiments.

The furnace had a square crucible, *m*, lined with fire brick. The current passed into the crucible through an upper graphite electrode, *n*, then through the charge, and then through another graphite electrode, *o*, placed in the bottom of the furnace, which was made of carbon. The exterior shell of the furnace was of fire brick. The bricks were held together by iron bands, which could be adjusted for expansion. The crucible was roofed with fire-clay tiles during some experiments, in others the roof was discarded. The upper electrode was hung so as to extend vertically into the center of the crucible. No attempt was made to keep the joints in the roof tight, so that gas escaped around the electrode. Part of the tile roof was put on before charging and the opening was completely closed when all of the charge had been added. There was one tap hole at the bottom of the crucible, in the middle of one side.

THE CRUCIBLE.

The crucible was 11 inches square and its walls were vertical. The carbon bottom was tapered up about 4 inches around the walls, so that the lowest part of the bottom was about 4 inches square. The depth of the crucible was 15 inches on the average, but varied a little with different linings. The total volume of the crucible was approximately 1,600 cubic inches, but this was varied slightly from time to time as convenient.

WALLS AND CRUCIBLE SUPPORT.

The crucible was supported on an arch of fire brick set on two brick bases, each 2 feet 7 inches by 9 inches in section, and 10 inches high to the base of the arch. The arch was of 1 foot 1 inch span. The outside dimensions of the furnace at the bottom were 2 feet 7 inches by 2 feet 7 inches. These dimensions extended up to a height 1 foot 8 inches above the floor, on a level with the end of the tap hole,

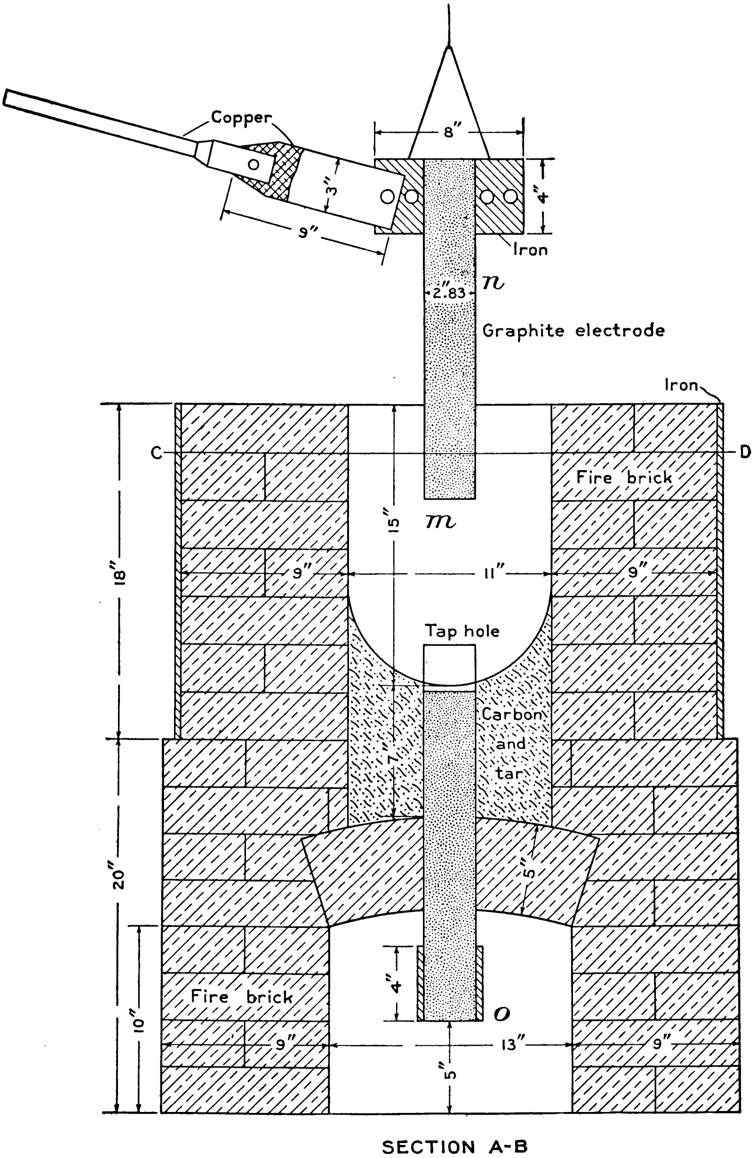

FIGURE 1.—Elevation of experimental furnace.

which was 4 inches lower than the bottom of the crucible. The brick walls surrounding the crucible were 2 feet 5 inches square and 1 foot 6 inches high, making the total height of the furnace 3 feet 2 inches. An iron plate would have been better than the arch of brick, but to have procured one would have involved considerable expense and loss of time.

ELECTRODES AND HOLDERS.

The upper electrode was of Acheson graphite and was 2 inches square and 36 inches long. The lower electrode was of the same cross section and material but was 18 inches long. The part above the top of the arch was embedded in a mixture of ground electrode stumps, or coke, made plastic with tar. This material covered the upper end of the electrode to a depth of 1 inch and extended up on the walls of the furnace. The lower end projected below the arch to within 5 inches of the floor. The electrode was supported at first by an iron clamp embedded in the bricks but later simply by pressure of the bricks and the carbon mass around it. The cross-sectional area of the electrode was, of course, 4 square inches. Current was conducted to the electrode by two insulated copper cables 1 inch in diameter. The cables extended 30 feet to four copper bus bars 4 inches by 1 inch in cross section, which extended 30 feet to the switchboard where all electrical readings were taken.

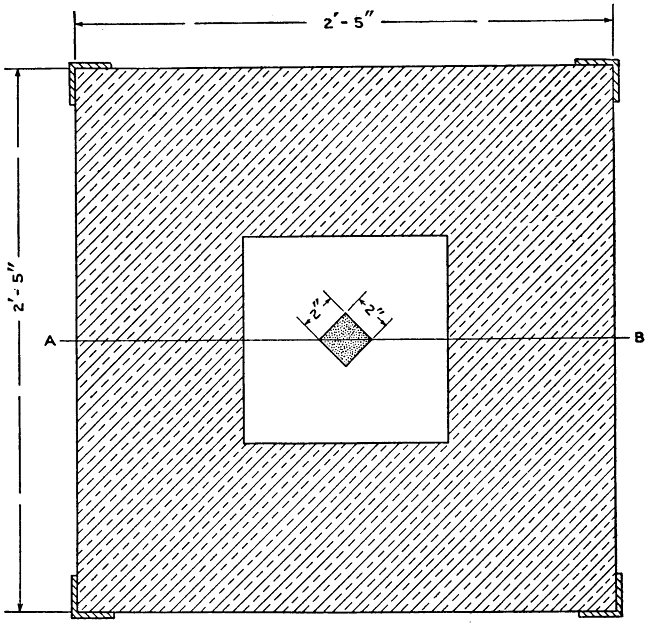

FIGURE 2.—Plan of experimental furnace.

The upper electrode holder consisted of two iron plates 4 inches wide by 8 inches long bolted together around the electrode. During some of the experiments this holder was connected to the circuit by a plate which held the loose wire of the connecting cable against the holder. This connection did not work well with high amperages, owing to short circuits burning out the wires and on account of the heating of the holder from the heat of the crucible. Later a copper plate 3 inches wide and 9 inches long was bolted to the iron holder and connected at the other end with a lug soldered in the cable. This arrangement gave complete satisfaction, as the lug was far enough from the crucible to prevent melting of the solder. The bottom connection was made by clamping the loose ends of the cable between the holder and a plate bolted to the holder, as was first used for the upper holder.

REGULATING DEVICE.

The upper electrode was suspended by flexible copper wire, running over a pulley attached to a steel crosspiece, which was set on a wooden framework 7 feet 6 inches high. The wire was then wound around a loose horizontal pipe, set at the rear of the furnace, and the electrode counterbalanced by attaching weights to the wire. This makes a simple and satisfactory device for hand regulation of electrodes in small furnaces, very close adjustment being possible.

For holding tight the upper section of the furnace, 2-inch channel irons were set on the corners and around them were placed two 1-inch strips of sheet iron.

APPARATUS AND INSTRUMENTS.

Single-phase current was received in the laboratory at a voltage which could be varied on the primary side from 440 to 550 volts with a frequency of 60 cycles. The voltage was stepped down by a 75 kilovolt ampere, single-phase transformer. The primary windings of the transformers were so adjusted and connected with switches and a voltage regulation box that the voltage could be regulated within three ranges, as follows: High voltages, ranging from 120 to 300 volts, by 12-volt steps; medium voltages, 60 to 150 volts, by 6-volt steps; and low voltages, 30 to 75 volts, by 3-volt steps. On the primary side there was a 300-ampere rheostat and a main switch fitted with a spring. On the secondary side there were two main switches and numerous voltage regulation switches.

The instruments used in taking readings were a Thomson high-torque, watt-hour meter set in the primary circuit; a Keystone indicating wattmeter, with a range of 100 kw., connected in the primary circuit; two Keystone alternating-current ammeters, with a range of 200 amperes, connected in the primary circuit, and a Keystone alternating-current voltmeter, the lower scale having a range of 100 volts and the upper scale a range of 500 volts, connected for both primary and secondary circuits. These instruments were set on a switchboard so situated that the current was conducted from 60 to 80 feet by bus bars and cables to the furnace.

Tappings were made in cast-iron molds, 19 inches long by 5 inches wide at the top, 17 inches long by 3½ inches wide at the bottom, and 4½ inches deep, with sloping sides 1 inch thick.

CONCENTRATES SMELTED.

The concentrates treated were practically subdivisions of grade No. 2, previously mentioned, and will be designated as No. 3 and No. 4. The coarser of the two grades, No. 3, is obtained by concentrating ore as follows: The rock is crushed with Blake breakers, stamped under a steam stamp; passed through a trommel with ¼-inch openings to a hydraulic classifier and to two jigs in series, the tailings from the last jig passing over a Wilfley table, the product of the table being grade No. 3. The overflow from the hydraulic classifier is conducted to a settling tank, and the slimes obtained are then treated upon a Card table, which gives grade No. 4. The copper in grade No. 3 was rather coarse and plainly visible; in No. 4 it was so fine that it could be detected only with difficulty by the naked eye, although by panning it was very easily concentrated to an appreciable extent. There was, however, considerable loss by sliming.

The sieve analyses of these concentrates are as follows:

Sieve analyses of Michigan copper concentrates.

Fineness.	No. 3.	No. 4.
	Per cent.	*Per cent.*
Less than 10-mesh	0.80	0.03
10 to 20 mesh	5.33	1.38
20 to 40 mesh	14.20	8.32
40 to 60 mesh	9.90	3.13
60 to 80 mesh	8.37	4.05
80 to 100 mesh	29.14	19.88
Over 100-mesh	32.22	63.19
	99.96	99.98

The great difference in the fineness of the two grades is apparent from these analyses, 87.12 per cent of No. 4 passing through a 60-mesh sieve, whereas only 69.73 per cent of No. 3 passed 60 mesh. The fineness of the material, especially that of the copper, seemed to affect the results obtained in the experiments.

The silica used was a ground quartzite containing 96.82 per cent SiO_2. The lime was from a limestone containing 63.2 per cent CaO and 5.7 per cent MgO. A hematite ore of the analysis given below was used as a flux.

The chemical analyses of the concentrates and of the hematite were as follows:

Chemical analyses of concentrates and hematite.

Constituent.	Concentrate No. 3.	Concentrate No. 4.	Hematite.
	Per cent.	*Per cent.*	*Per cent.*
Cu	37.35	25.35	
SiO_2	33.33	25.92	5.19
FeO	15.40	21.60	
Al_2O_3	8.78	12.40	1.22
CaO	5.18	5.98	0.20
MgO	1.09	1.09	0.16
S	0.056	0.096	
P			0.61
Fe	0.30	0.50	51.68
Mn			0.49

As may be seen from the above analyses, both grades of concentrates were rather high in alumina. The iron, silica, and alumina were so proportioned in each as to give a rather infusible slag. The concentrates were not uniform in composition, the analyses given being the average from several samples. This variation in composition of the concentrate affected the results in some experiments in that the calculated amount of copper in the charge and that tapped do not check well.

PROBLEMS OF ELECTRIC SMELTING OF NATIVE COPPER CONCENTRATES.

Before describing the experiments, some of the problems that arise in connection with the smelting of native copper concentrates in the electric furnace will be discussed.

As previously stated, the treatment of these concentrates consists of a melting operation and the separation of the metallic copper and gangue by reason of the difference in their specific gravities.

LOSS OF COPPER.

The loss of copper by slagging and volatilization depends upon the melting point, fluidity, and specific gravity of the slag. The slag should have as low a melting point and be as fluid as possible in order to prevent volatilization of the copper. In the electric furnace there is always a tendency for the furnace to develop a higher temperature as the melting point of the charge is raised, so that charges which could not be treated in an ordinary metallurgical furnace are easily smelted. For this reason the furnace must be watched to keep the temperature low in treating easily volatilized metals. As long as there is slag covering the metal in an electrode furnace the loss is not excessive, but with an arc and no slag the copper is rapidly volatilized or oxidized.

The loss of copper in the slag depends, as stated above, on the fluidity and specific gravity of the slag. The loss in treating native copper concentrates in the electric furnace is practically entirely due to mechanical loss of minute metallic globules that are carried off because of imperfect settling. A fluid and liquid slag of low specific gravity prevents such loss.

REDUCTION OF IRON OXIDE.

The only reducing material present in the electric furnace during a simple melting operation would be the carbon of the electrode. As a result, the reducing action need not be very strong in operating an electric furnace for this purpose. If the slag has a low melting point, there is no reason why the reduction of FeO to metallic iron should be excessive. This factor would have greater influence in a small furnace than in a large furnace, where the electrode area exposed to the charge would not be so great in proportion to the amount of material charged. In order to lower the reducing action, the temperature should be kept as low as possible and still give a fluid slag. Also in the electric furnace the effect of the base used to flux the silica may result in a decreased reduction of FeO.

DESCRIPTION OF EXPERIMENTS.

SMELTING CONCENTRATE NO. 4 WITHOUT FLUX.

Two experiments were made in which No. 4 concentrate was smelted without the addition of anything to flux the gangue, the results of which are shown in Table 2, experiments Nos. 1 and 2.

EXPERIMENT NO. 1.

Before experiment No. 1 was begun the furnace was heated with gas for an hour and a half to burn the tar out of the lining. The furnace was then filled with concentrate, and the current turned on.

Throughout the experiment the furnace did not run well and it was difficult to keep it from arcking at times. A green flame burned at the top, due to the oxidation of some of the copper. Carbon monoxide gas was also given off, due to the partial oxidation of the electrode.

In about one hour the charge was ready for tapping. Instead of tapping into an iron mold, as was done in later experiments, the charge was tapped into a mold made of bricks. The furnace did not tap well, because the bottom had settled, so that some metal was left in the furnace, and was later removed by chiseling.

No copper could be seen in the slag except at the bottom of the mold, where small pellets had concentrated. The slag flowed out of the furnace readily, but was a little thick, and had a glassy appearance when cold. Some of the carbon bottom came up and mixed with the slag in a semifused mass. There was not enough tar mixed with the carbon to make it set properly.

The metal contained so much iron that it could be seen with the naked eye. Under the microscope, iron, copper, and blowholes were visible, but no cuprous oxide.

EXPERIMENT NO. 2.

The furnace was hot when run No. 2 was started. The furnace tapped cleanly, the slag was glassy and rather thick, but flowed from the furnace. It was free from shots of metal. One brick in the furnace was eaten out by the slag. The metal showed the same characteristics as that from run No. 1.

VARYING THE ACIDITY OF THE SLAG WITH SILICA.

Five experiments were made, using varying amounts of silica for the purpose of noting the general effect of a very acid slag. Slags Nos. 3, 4, 5, 6, and 7 (see Table 2) were made in this manner. In all of these experiments No. 4 concentrate was used with silica as a flux.

EXPERIMENT NO. 3.

In experiment No. 3, the furnace was started cold. The furnace ran well, almost entirely as a resistance furnace. The green flame noted in the two previous experiments was present. The lining was in such poor condition that the metal could not be tapped, but was removed after cooling. The slag was viscous and very glassy, but flowed from the furnace. The metal showed iron and blowholes, but no cuprous oxide.

EXPERIMENT NO. 4.

In experiment No. 4, the furnace was hot from the previous run. The furnace ran well throughout the run, as a resistance furnace. The slag was viscous and very glassy, but flowed from the furnace. No iron or cuprous oxide were visible in the metal, but it contained some blowholes.

EXPERIMENT NO. 5.

Before beginning experiment No. 5 the furnace was relined and heated with the air to burn out the tar in the lining. The charge was slow in melting and shots of metal floated in the slag. On tapping, these seemed to settle toward the bottom of the mass. The slag was very thick and glassy, but flowed from the furnace. During the run, the furnace was operated entirely as a resistance furnace. The metal, on being tapped into two round ladles used for molds, froze and had to be melted out. This accounts for the abnormally high percentage of iron in the metal.

Under the microscope iron and blowholes were clearly visible, but no cuprous oxide.

EXPERIMENT NO. 6.

At the start of experiment No. 6 the furnace was cold. The power supply was very unsteady, being off for 18 minutes throughout the run. On tapping, the slag was thick and glassy, but flowed from the furnace, and no copper could be seen which did not settle out. There was a large amount of iron visible in the metal under the microscope, but it was present in a more dendritic form than in previous experiments. No cuprous oxide was formed.

EXPERIMENT NO. 7.

In experiment No. 7 the furnace was hot before charging. The slag was sticky and thick, but flowed from the furnace and was free from shots. Under the microscope very little iron could be seen in a section of the metal, and no cuprous oxide at all. The metal was also comparatively free from blowholes.

VARYING THE BASICITY OF SLAG WITH HEMATITE.

In experiments No. 8 and No. 9 (Table 2) hematite was used to note the effect of a more basic slag and the value of iron oxide as a flux in comparison with lime.

EXPERIMENT NO. 8.

In experiment No. 8 the furnace was hot before charging. The furnace acted better than in previous runs, probably because of the slag being more fusible. The slag was very fluid on tapping. It was of a dull basic appearance and free from shots of copper. Neither iron nor cuprous oxide was visible in the metal under the microscope.

EXPERIMENT NO. 9.

The furnace was cold at the beginning of experiment No. 9. The slag was dull, basic, and very fluid and was free from shots of metal. Neither iron nor cuprous oxide was visible in the metal.

VARYING THE BASICITY OF THE SLAG WITH LIMESTONE.

EXPERIMENT NO. 10.

In experiment No. 10 limestone was used as a flux. The furnace was hot at the start and operated well throughout the run. The slag was very fluid and basic and was free from shots of metal. A very little iron was visible under the microscope, but no cuprous oxide. The power consumption was very low.

VARYING THE BASICITY OF THE SLAG WITH LIMESTONE AND HEMATITE.

In experiments No. 11, No. 12, and No. 13 the basicity of the slag was changed by using both limestone and hematite and was increased as far as was considered practical.

EXPERIMENT NO. 11.

In experiment No. 11 the furnace was started cold. The slag was very fluid and boiled out of the top of the furnace considerably. Iron was distributed rather evenly throughout the metal as dendrites, but no cuprous oxide could be detected.

EXPERIMENT NO. 12.

The furnace was cold at the start of experiment No. 12. During the run fine globules of copper floated in the slag, which boiled out of the furnace considerably. There was so much of this boiling and slipping of the charge that some of the limestone dust was carried off by the gases. The slag was a little thick and sticky, but flowed readily from the furnace. Very fine dendrites of iron were visible in the metal, but there appeared to be no cuprous oxide.

EXPERIMENT NO. 13.

Experiment No. 13 was started with a hot furnace, but a rather sticky and thick slag was produced, which boiled and slipped so much that there was considerable loss of limestone and ore dust. There were shots of metal in the top of the slag in the furnace, but these settled out in the mold. Under the microscope only a few pieces of iron were visible in the metal and no cuprous oxide.

CONTINUOUS SMELTING.

EXPERIMENT NO. 14.

In experiment No. 14 concentrate No. 3, with lime as a flux, was charged at intervals so as to keep the furnace full, and three tappings were made during a run of 2 hours and 40 minutes. The furnace operated well and very little gas was given off, so long as the top of the charge was not fused.

When the bath was tapped, considerable material hung on the walls. The slags were very fluid, but the first two tapped contained considerable metal because of semifluid material running out of the furnace with the slag. No iron or cuprous oxide could be detected in the metal from any of the tappings.

RESULTS OF EXPERIMENTS.

The results of each experiment were tabulated as shown in Table 1, which gives the results of experiment No. 10 in detail.

TABLE 1.—*Results of smelting concentrate No. 3 with limestone flux, experiment No. 10.*

Time.	Amperes.	Volts.	Kilo-watts.	Power factor.	Remarks.
P. m.					
12.35	500	70	33	94.3	Furnace started hot after experiment No. 9.
12.40	630	72	45	98.8	Charge all added at 12.37 p. m.
12.45	680	71	48	99.2	All fused at 12.46 p. m.
12.50	600	78	46	98.0	Shots of metal in upper part of slag.
12.55	580	80	46	99.0	Tapped. Slag fluid and free from shots.

Charge:	
Michigan concentrate No. 3, pounds	22.0
Limestone, pounds	2.2
Length of run, minutes	20
Mean volts in the furnace	74
Mean amperes on the furnace	598
Mean kilowatts on the furnace	44
Mean power factor of circuit	.97
Kilowatt-hours	14.4
Electrode consumption per pound of metal tapped, pounds	.052
Electrode consumption per ton of charge smelted, pounds	28.2
Calculated weight of copper, pounds	8.21
Weight of metal tapped, pounds	8.13
Weight of slag tapped, pounds	16.09
Loss of copper in the slag, per cent	.33
Copper unaccounted for, per cent	3.17
Total loss of copper, per cent	3.50
Kilowatt-hours per pound of metal tapped	1.77
Kilowatt-years per ton of ore smelted	.15
Kilowatt-years per ton of charge smelted	.135
Analysis of metal:	
Copper, per cent	97.13
Iron, per cent	2.50
Sulphur, per cent	.35
Arsenic, per cent	.02
Analysis of slag:	
Cu, per cent	.17
SiO_2, per cent	35.82
CaO, per cent	15.45
MgO, per cent	7.09
FeO, per cent	25.20
Al_2O_3, per cent	17.10

REMARKS.—On examining a microscopic section of the metal, after etching with picric acid, a few spots of dendritic iron could be seen, but no cuprous oxide.

The complete results of experiments 1 to 14 are tabulated in Table 2.

TABLE 2.—*Results of smelting of fine Michigan native copper concentrates in the electric furnace.*

Item.	Series 1.		Series 2.					Series 3.		Series 4.	Series 5.			Series 6.			Average.
	1	2	3	4	5	6	7	8	9	10	11	12	13	14			
Composition of charge:																	
Concentrate No. 4, pounds	22.0	22.0	22.0	22.0	22.0	22.0	22.0	......	22.0	......	......	......	......		.....		......
Concentrate No. 3, pounds	......	......	......	......	......	......	......	22.0	......	22.0	19.6	19.6	22.0	5.5	97.1		......
Quartzite, pounds	......	......	2.2	2.2	4.4	4.4	4.4	......	......	......	......	......	......		.....		......
Hematite, pounds	......	......	......	......	......	......	......	1.1	2.2	......	1.98	4.18	6.6		.....		......
Limestone, pounds	......	......	......	......	......	......	......	......	......	2.2	1.98	4.18	6.6		9.71		......
Composition of metal:																	
Cu, per cent[a]	87.29	88.62	96.24	97.96	83.82	90.95	95.95	98.59	97.96	97.13	93.73	93.51	96.64	[b]99.49	[c]98.69	[d]97.77	......
Fe, per cent	12.50	11.10	3.50	1.75	15.90	9.25	3.75	1.10	1.90	2.50	6.20	6.30	3.10	[b].30	[c]1.04	[d]2.00	......
S, per cent	.19	.26	.24	.27	.26	.21	.30	.29	.12	.35	.15	.17	.24	[b].19	[c].25	[d].21	......
As, per cent	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	.02	[b].02	[c].02	[d].02	......
Composition of slag:																	
Cu, per cent	.40	.23	.31	.23	.15	.15	.10	.15	.02	.17	.54	.24	.46	[b]1.91	[c]1.60	[d].62	......
SiO_2, per cent	49.80	40.60	53.22	51.10	49.96	42.58	48.00	44.88	31.71	35.82	33.15	33.96	29.37	[b]34.01	[c]34.33	[d]34.62	......
CaO, per cent	13.25	9.07	12.40	8.78	10.10	8.80	9.00	6.92	10.10	15.45	17.60	20.04	19.20	[b]16.80	[c]17.50	[d]20.32	......
MgO, per cent	1.93	3.00	3.12	2.03	4.60	2.71	1.84	2.32	2.45	7.09	3.61	4.46	4.17	[b]2.95	[c]3.14	[d]2.86	......
FeO, per cent	15.25	21.15	11.58	25.50	19.90	29.00	26.25	29.80	37.95	25.20	29.00	27.60	31.25	[b]26.70	[c]27.30	[d]24.70	......
Al_2O_3, per cent	13.70	[e]23.82	20.33	10.45	16.10	16.48	14.99	15.34	14.89	17.10	16.28	11.75	14.10	[b]17.09	[c]14.83	[d]13.35	......
Length of run, minutes	50	30	30	24	25	23	22	23	23	20	25	30	26		100		......
Mean volts on the furnace	59	86	90	79	80	75	82	64	57	74	66	77	70		66		73
Mean amperes on the furnace	......	......	......	532	......	542	444	710	808	598	683	663	670		557		621
Mean kilowatts on the furnace	32	33	40	40	42	38	34	43	45	44	44	49	45		34		40
Mean power factor of circuit	......	......	......	.95	......	.95	.93	.94	.97	.97	.96	.97	.96		.94		.95
Kilowatt-hours	26.7	16.4	20	16	17 6	14.6	12.5	16.4	17.2	14.4	18.4	20.3	19.5		57.3		......
Electrode consumption per pound of metal tapped, pounds.	.08	.16	.043	.12	.16	.12	.12	.052	.052	.052	.071	.071	.071		.036		.086
Electrode consumption per ton of charge smelted, pounds	48	41.9	21.8	33.25	41.9	33.25	33.25	28.20	28.20	28.20	34.20	34.20	34.20		22.10		34.5
Calculated weight of copper, pounds	5.58	5.58	5.58	5.58	5.58	5.58	5.58	8.21	5.58	8.21	7.31	7.31	8.20		36.30		......
Weight of metal tapped, pounds	[f]6.60	3.97	5.45	4.52	2.86	3.07	2.97	7.60	3.63	8.13	6.81	6.81	7.04	[b]11.0	[c]6.28	[g]15.28	......
Weight of slag tapped, pounds	13.58	21.10	16.22	21.30	17.95	15.82	22.20	14.42	16.40	16.09	16.20	16.60	15.03	[b]15.83	[c]16.55	[h]34.30	......
Loss of copper in slag, per cent	.97	.87	.90	.88	.48	.42	.40	.26	.59	.33	.54	.22	.85		2.12		.70
Copper unaccounted for, per cent.	......	36.90	5.03	17.65	56.40	49.20	48.30	7.22	35.25	3.17	11.62	12.03	16.41		10.85		23.10
Total loss of copper, per cent	......	37.77	5.93	18.53	56.88	49.62	48.70	7.48	35.84	3.50	12.16	12.25	17.26		12.97		23.80
Kilowatt-hours per pound of metal tapped	4.05	4.13	3.67	3.53	6.30	4.75	4.19	2.13	4.73	1.77	2.68	2.96	2.77		1.76		3.51
Kilowatt-hours per ton of ore smelted	2,425	1,500	1,820	1,450	1,600	1,320	1,130	1,490	1,565	1,310	1,880	2,070	1,780		1,190		1,430
Kilowatt-hours per ton of charge.	2,425	1,500	1,740	1,320	1,330	1,075	1,015	1,420	1,425	1,280	1,565	1,455	1,100		1,075		1,380

[a] Copper by difference.
[b] First tapping.
[c] Second tapping.
[d] Third tapping.
[e] Fire bricks melted into the slag.
[f] Copper left in furnace from previous run.
[g] Third tapping. Total metal tapped in all three tappings, 32.56 pounds.
[h] Third tapping. Total slag tapped in all three tappings, 66.80 pounds.

DISCUSSION OF RESULTS.

As previously stated, the three most important points for consideration are: (1) The loss of copper by slagging, volatilization, or by dust being carried off in the gases; (2) the reduction of the iron oxide to metallic iron which enters the copper; and (3) the power consumption and general operation of the furnace.

All of these factors are largely affected by the melting point of the slag and the temperature necessary to make it liquid enough to flow readily from the furnace. The results obtained from the experiments just described can be used as a basis for discussion in considering the commercial application of the electric furnace to the smelting of these concentrates.

COPPER LOSSES.

In order to have a uniform basis for comparing the results obtained by varying the silica, lime, and ferrous oxide contents of the slag, the oxygen ratios of the various slags were computed and tabulated. In doing this alumina was considered as a base. Of course the results, considered on this basis, do not show conclusively that a slag of a certain oxygen ratio gives the best results, because the high percentage of alumina in the slag varied considerably, owing to melting of the brick in the furnace lining. That is, the effect of alumina, when substituted for another base, on the fluidity of the slag is not shown completely.

The oxygen ratios of the different slags and the copper losses in the different tests were as follows:

Losses of copper with different slags.

Experiment No.	Ore smelted.	Flux used.	Oxygen ratio of slag. $\left(\frac{\text{Acid O}}{\text{Basic O}}\right)$	Proportion of copper in slag.	Proportion of copper lost.[a]		
					Losses in slag.	Losses by volatilization, in hanging pieces, and in dust.	Total.
				Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
4	Concentrate No. 4.	Silica	1.97	0.23	0.88	17.65	18.53
1	do	None	1.85	.40	.97	....	....
5	do	Silica	1.60	.15	.48	56.40	56.88
7	do	do	1.59	.10	.40	48.30	48.70
6	do	do	1.28	.15	.42	49.20	49.62
2	do	None	1.10	.23	.87	36.90	37.77
9	do	Hematite	.87	.20	.59	35.25	35.84
8	Concentrate No. 3.	do	1.44	.15	.26	7.22	7.48
12	do	Limestone and hematite	.93	.24	.22	12.03	12.25
10	do	Limestone	.91	.17	.33	3.17	3.50
13	do	Limestone and hematite	.82	.46	.85	16.41	17.26
11	do	do	.70	.54	.54	11.62	12.16

[a] Computed on basis of total copper in ore.

EXPERIMENTS WITH CONCENTRATE NO. 4.

From the table it is evident that as the acidity of the slag is decreased for concentrate No. 4 there is a gradual decrease in the percentage of copper in the slag until a slag having an oxygen ratio of 1.59 is reached, when the loss rises slightly and remains uniform with further increase of basicity. Concentrates No. 3 and No. 4 are considered separately because of the higher alumina and lower copper content in No. 4. The oxygen ratio does not vary directly with composition of the charge, because some of the slag from previous runs and some of the lining entered the slag. The ratio 1.59, at which the percentage of copper in the slag reaches the lowest point (0.10 per cent), corresponds to that of a slag containing 48 per cent SiO_2, 9 per cent CaO, 1.84 per cent MgO, 26.25 per cent FeO, and 14.99 per cent Al_2O_3. With this ratio the lowest percentage of losses in the slag from No. 4 concentrate, on the basis of total copper in the ore charged, occurred. The percentage of copper unaccounted for was high, however, though in all experiments, except where noted, the furnace was tapped practically clean, except for what slag stuck to the wall in spots. This unaccounted for loss may be considered as largely due to volatilization, dust loss, and hanging of the charge. To be on the safe side, all unaccounted for loss will be considered as volatilization and dust loss, although in a small furnace, without tearing out the lining, 10 per cent of hanging charge would not be excessive. The total loss of copper in experiment No. 7, including copper unaccounted for—that is, copper lost by volatilization, as dust, and by hanging of the charge, and the copper loss in the slag—was 48.7 per cent.

A further study of the results of lowering the acidity of the slag shows that although there is an increase in the percentage of copper in the slag it is not excessive, and that there is a considerable decrease in the percentage of copper lost in dust or by volatilization. With an oxygen ratio of 0.87, the percentage of copper in the slag is 0.20 per cent, the percentage of total copper charged that was lost in slag is 0.59 per cent, the copper unaccounted for is 35.25 per cent, and the total loss of copper 35.84 per cent. The slag resulting from experiment No. 4 shows the effect of a low percentage of alumina in a marked manner. This slag has an oxygen ratio of 1.97, greater than any of the others, but the loss by volatilization and in dust is less than in any other experiment with No. 4 concentrate, being 17.65. The slag contained 0.23 per cent copper, the total loss being 18.53 per cent of the copper contained in the quantity of ore charged. This slag contained 51.1 per cent SiO_2, 8.78 per cent CaO, 2.03 per cent MgO, 25.5 per cent FeO, and only 10.45 per cent of Al_2O_3. The cause of the decrease in the alumina content was not apparent. This evidently lowered the melting point considerably, so that less copper was volatilized.

The slag loss of copper was low in all of the experiments with No. 4 concentrate and the loss by volatilization and in dust was high. These two losses, the writers believe, can be eliminated. The material used was much finer than No. 3 concentrate, as shown by the analysis on p. 21. This fineness of subdivision accounts in a measure for the oxidization of copper dust, which was plainly visible during the experiments, as shown by the marked green flame. Also, more dust was carried off by the gases from the oxidation of the electrode than in the later experiments with concentrate No. 3. The results of the experiments with concentrate No. 3, in which a more basic slag was used, show that the volatilization loss can be kept low by using a slag with a low melting point. As the oxygen ratio was decreased in the tests with concentrate No. 4 the decrease of this loss was marked. The dust loss would be small in a large furnace. It probably would be necessary to use a slag higher in ferrous oxide and having a lower melting point for concentrate No. 4 than for No. 3, in order to keep down the oxidation and volatilization of fine copper.

EXPERIMENTS WITH CONCENTRATE NO. 3.

The experiments with concentrate No. 3 showed a much less total loss than did those with No. 4. This was undoubtedly due to the fact that the slags used for No. 3 had a lower oxygen ratio, resulting in a much more fluid slag with a lower melting point. Also bases were added to dilute the alumina, so that this constituent had less effect than in the previous experiments. The best results were obtained in experiment No. 10, in which the oxygen ratio of the slag was 0.91, the percentage of copper in the slag being 0.17. The total loss of copper in the slag was 0.33 per cent of the amount of copper in the ore charged. The loss by volatilization, in dust, and in hangings was 3.17 per cent, making a total loss of 3.5 per cent. These results were obtained by increasing the lime content by charging limestone, giving a slag containing 35.82 per cent SiO_2, 15.45 per cent CaO, 7.09 per cent MgO, 25.20 per cent FeO, and 17.10 per cent Al_2O_3. That this slag gives good results was confirmed in the continuous experiment No. 14, in which the same proportions of charge were used as was used in No. 10.

As the oxygen ratio became lower than about 0.91 there was a tendency for increased losses in the slag and a higher loss from other causes. It seems from the results that the most satisfactory slag for these particular ores is one with an oxygen ratio of about 1.0, that is, a monosilicate, and containing 35 per cent SiO_2, 22 per cent CaO and MgO, 25 per cent FeO, and 17 per cent Al_2O_3. The oxygen ratio of this slag corresponds to that of an ordinary copper blast furnace slag, but owing to the nature of the ore the alumina content is very high. Also the use of lime as a base instead of iron oxide seems to give better results with ore of high alumina and iron oxide content as regards copper losses, due probably to the formation of a slag of lower melting point. Increasing the lime content also lowers the specific gravity of the slag, thus permitting easier settling of the metal.

There is no reason why the results obtained with the coarser material No. 3 should not apply to the finer concentrate No. 4. The high losses in experiments with the latter were due largely to the high melting point of the slag. As no more ore was at hand, the experiments could not be continued with this concentrate, and No. 3 was used. The percentage of copper in the slag need not exceed 0.25 per cent with a slag of proper composition. The percentage loss in the slag of total copper in the ore charged should not exceed 0. 5 per cent. The other losses in the best run (No. 10) amounted to 3.17 per cent, but in practice with continuous operation of a larger furnace this item should not exceed 1 per cent, and the total loss should not be over 1.5 per cent of the total amount of copper in the ore charged. The use of limestone is preferable in the case of the material treated, simply because of the production of a slag of easier fusibility and lower specific gravity, which allows better mechanical separation of metallic copper and slag and reduces the loss by volatilization. The dust loss would also be less if the very fine concentrate was mixed with a coarse concentrate.

IRON IN THE METALLIC COPPER.

In some of the runs in which a slag of high oxygen ratio and high melting point was used the percentage of iron in the copper produced was very high. The effect of length of run and of the ratio of acid to basic constituents in the slag is shown in the following table:

Metallic iron in the metal produced, experiments Nos. 1 to 13.

Experiment No.	Ore smelted.	Flux used.	Length of experiment.	Oxygen ratio. $\left(\frac{\text{Acid O}}{\text{Basic O}}\right)$	Proportion of iron in metal.	Weight of iron in metal.	Weight of metallic iron in charge.	Weight of iron reduced.	Percentage of electrode consumption used in reduction.
			Minutes.		*Per ct.*	*Pounds.*	*Pounds.*	*Pounds.*	
4	Concentrate No. 4.	Silica	24	1.97	1.75	0.079	0.110	0.0	0.0
1	do	None	50	1.85	12.50	.825	.110	.715	29.10
3	do	Silica	30	1.68	3.50	.191	.110	.081	7.41
5	do	do	25	1.60	15.90	.454	.110	.344	16.20
7	do	do	22	1.59	3.75	.111	.110	.001	.06
6	do	do	23	1.28	9.25	.283	.110	.173	10.35
2	do	None	30	1.10	11.10	.441	.110	.331	11.60
9	do	Hematite	23	.87	1.90	.069	.110	.0	.0
8	Concentrate No. 3.	do	23	1.44	1.10	.083	.066	.017	.92
12	do	Limestone and hematite.	30	.93	6.30	.429	.059	.370	16.40
10	do	Limestone	20	.91	2.50	.203	.066	.137	6.95
13	do	Limestone and hematite.	26	.82	3.10	.217	.066	.151	6.50
11	do	do	25	.70	6.20	.423	.059	.364	16.10

There was a small amount of free iron present in both concentrates No. 3 and No. 4, which entered the metal to an appreciable extent. This free iron was from the wear of breakers and stamps during milling of the ore. The influence exerted by the carbon electrode as a reducing agent is also shown in the table. Aside from iron little attention was paid in this research to other impurities in the copper, as it would be necessary to refine the copper in any case. The sulphur and arsenic ran about the same in all the samples. However, it was desirable to ascertain whether the ore could be melted and the gangue separated from the copper without reduction of iron by the carbon of the electrode to an extent that would contaminate the product and cause excessive expense in subsequent refining in a reverberatory furnace. In the reverberatory furnace the iron is removed from copper by oxidation and slagging, so that an excessive amount of iron, say over 5 per cent, would increase the cost of the process considerably.

FACTORS AFFECTING REDUCTION OF IRON BY THE ELECTRODE.

The chief factors which might have influenced the percentage of iron reduced by the carbon electrode are the length of time the ore was exposed to the electrode and the melting point of the slag formed. The time of exposure to the reducing action of the electrode did not appear to have a marked effect on the percentage of iron reduced, for in some tests a product high in iron was obtained after a run of the same length of time as an experiment in which the product contained little iron. For example, the copper from run No. 6, which lasted 23 minutes, contained 9.25 per cent of iron, whereas that from No. 9, which also lasted 23 minutes, contained only 1.90 per cent of iron.

A classification of the products of the experiments with both concentrates according to the oxygen ratios of the slags does not show a uniform change in the percentage of iron, but in general the percentage of iron reduced by the electrode decreases as the slag becomes more basic and approaches a monosilicate. This result is probably due entirely to the higher melting point of the more acid slag. The furnace, when operating as a resistance furnace, adjusts itself somewhat to the melting temperature of the charge so that with a refractory slag it gets hotter. This increase in temperature results in accelerating the reaction between the carbon electrode and the iron oxide of the gangue. With a monosilicate slag containing not over 15 per cent of alumina this reduction of iron need not result in a product containing over 2 per cent iron, including any metallic iron in the charge. This was shown by the results of the intermittent runs (see p. 28) and was further confirmed by experiment No. 14, in which the metals from three tappings during continuous operation of the furnace contained, respectively, 0.30 per cent, 1.04 per cent, and 2 per cent of iron.

The proportion of the total electrode consumption used in the reduction of iron varied from zero to 29.1 per cent, the average for 13 experiments being 9.35 per cent. From these results it may be seen that, with a furnace operating at a low temperature, the reduction of iron by the carbon electrode should not be excessive, or enough to result in a product containing less than 95 per cent of metallic copper. The writers believe that in practice the contents of metallic iron and other impurities could be kept as low as 2 per cent, giving a product containing 98 per cent copper.

EFFECT OF BASE USED.

The kind of base used to form a monosilicate slag did not seem to affect materially the reduction of iron oxide. The experiments indicate that there was slightly less reduction with hematite than when limestone was added, but there is so little difference that probably as good results could be obtained with one flux as with the other, the problem being simply one of economics as to which is the cheaper. The essential point is to have a fluid slag of low melting point, which can be obtained with either flux. It was thought in the beginning of the experiments that by the use of a slag high in silica it might be possible to eliminate the reducing effect of the electrode. This result was not accomplished in most of the tests because the slag formed was difficultly fusible, owing to the high alumina content of the ore. The high temperature increased the reducing effect so that any strong tendency of the FeO to combine with the SiO_2 was more than offset.

EFFECT OF ALUMINA.

Here, as well as in the discussion of copper losses, the effect of the alumina, as regards the reduction of iron oxide, should not be forgotten. In considering a series of very acid slags for copper losses and iron oxide reduction, it may be noticed that any diversion from high loss by volatilization was generally due to a decrease in the percentage of alumina in the slags. The adaptability of the electric furnace to smelting slags of high alumina content is clearly demonstrated by these experiments. As before stated, when the electric furnace is operated with the charge as the resistance and no arcs are formed, it adapts itself to the temperature required to melt the charge and produce a fluid slag. Of course there are many slags which could not be melted without undue power expenditure.

CONCLUSIONS.

These experiments show that the quantity of metallic iron reduced by the action of the carbon electrode is so small as to be of no practical importance as a drawback to the substitution of the electric furnace for the reverberatory furnace in the smelting of these concentrates. The above statement also applies to sulphide copper ore. The small amount of iron which is reduced alloys with the copper, or, if the ore is a sulphide, enters the matte and increases its collecting power for gold, especially if the matte consists mainly of FeS. In either case not enough iron could be reduced to form a product so impure that excessive refining or reconcentration would be necessary.

POWER CONSUMPTION.

With the exception of the continuous run, experiment No. 14, the lowest power consumption and in general the best results of all the experiments were obtained in experiment No. 10, in which concentrate No. 3 was fluxed with limestone. For comparing the theoretical power consumption with the actual consumption the charge used in this run is taken as a basis.

For convenience, the results of this run will be reduced to the basis of 1,000 kilograms of ore; that is, 1 metric ton (2,204 pounds). The quantity of limestone required to flux 1,000 kilograms of ore is 220 kilograms. The metal is assumed to contain 2.50 per cent iron, which includes that reduced from the FeO of the ore by the carbon electrode, and any metallic iron present. The gases, CO from the electrode and CO_2 from the limestone, are assumed to escape at an average temperature of 500° C. at the top of the furnace during the melting stage. This temperature is high, but there was no shaft on the furnace. The ore is assumed to be dry and of the composition given on page 22. The loss of copper in the slag is assumed to be 0.33 per cent of the total amount of copper in the ore charged.

The results, computed on this basis, are as follows:

Results of smelting concentrate No. 3 with limestone flux on basis of 1,000 kilograms of ore, experiment No. 10.

	Quantity charged or consumed.	Quantity in product formed.		
		Metal.	Slag.	Gases.
Ore No. 3:	*Kilos.*	*Kilos.*	*Kilos.*	*Kilos.*
Cu	373.5	372.27	1.23	
SiO_2	333.3		333.30	
Fe	122.7	9.53	113.17	
Al_2O_3	87.8		87.8	
CaO	51.8		51.8	
MgO	10.9		10.9	
O	34.3		32.5	1.8
	1,000.0			
Limestone flux:				
CaO	121.0		121.0	
MgO	19.09		19.09	
CO_2	79.2			79.2
Carbon of electrode	14.1			14.1
Oxygen from air, completely burned to CO, with oxygen from iron reduction	17.0			17.0
Total	1,264.69	381.80	770.79	112.1

Results of smelting concentrate No. 3 with limestone flux on basis of 1,000 kilograms of ore, experiment No. 10—Continued.

COMPOSITION OF SLAG.

Constituent.	Calculated.	Analyzed.
	Per cent.	*Per cent.*
Cu	0.16	0.17
SiO_2	43.20	35.82
CaO	22.40	15.45
MgO	3.88	7.09
FeO	18.81	25.20
Al_2O_3	11.37	17.10

The wide difference between the actual analysis and the calculated composition of the slag may be accounted for by several factors which must be taken into consideration. Possibly some of the fire-brick lining was fused and entered the charge, thus increasing the alumina content; or some of the previous charge, which contained hematite, may have been left in the furnace; also during the run considerable limestone was carried out of the furnace by the gases.

In the calculation of the theoretical power required it seems best to consider the melting point of the slag as that of the slag actually obtained. According to Hofman [a] the melting point of a slag containing 32.1 per cent SiO_2, 17.95 per cent FeO, 16 per cent CaO, and 11.37 per cent Al_2O_3 is 1,310° C. and that of a slag containing 32.1 per cent SiO_2, 24.69 per cent FeO, 22 per cent CaO, and 11.37 per cent Al_2O_3 is 1,145° C. The melting point of the slag actually obtained in run No. 10 is assumed to be 1,250° C. and the temperature necessary to make it flow to be 1,350° C. The quantity of heat necessary to melt and superheat the slag is assumed to be 600 calories per kilogram. In oxidizing C to CO; 8,100 calories are developed per kilogram of carbon. The amount of heat necessary to reduce 1 kilogram of Fe from FeO is assumed to be 1,173 calories. The quantity of heat necessary to expel 1 kilogram of CO_2 from $CaCO_3$ is 191 calories. The specific heat of dry carbon monoxide at 0° to 500° C. is 0.256 calorie per kilogram and that of carbon dioxide is 0.245 calorie per kilogram. It is assumed that 204 calories of heat are required to melt and superheat 1 kilogram of pure copper, and that 370 calories are required for 1 kilogram of iron. The metal contained some iron, but owing to insufficient data on the specific heats of alloys of iron and copper, the specific heat of pure copper is used. All of the heat constants are taken from Richards's "Metallurgical Calculations."

The theoretical amount of heat required to smelt 1,000 kg. of ore may now be computed, as follows:

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 395.

Theoretical amount of heat required to smelt 1,000 kg. of ore.

	Calories.
Melting and superheating 372.27 kg. of copper.....$372.27 \times 204=$	75, 943
Melting and superheating 9.53 kg. of iron............$9.53 \times 370=$	3, 386
Reducing 9.53 kg. of Fe from FeO.................$9.53 \times 1{,}170=$	11, 178
Melting and superheating 770.79 kg. of slag........$770.79 \times 600=$	462, 474
Raising the temperature of 32.8 kg. of CO to 500° C.......... ....$500 \times 0.256 \times 32.8=$	4, 195
Raising the temperature of 79.2 kg. of CO_2 to 500° C.......... ....$500 \times 0.245 \times 79.2=$	9, 650
Total..	566, 826
Amount of heat supplied by combustion of 14.1 kg. of carbon ....$14.1 \times 8{,}100=$	114, 210
Amount of heat supplied by electric current..................	452, 616
Total..	566, 826

The theoretical amount of electrical energy required for smelting No. 3 concentrate is $\frac{452{,}616}{857}=529$ kilowatt-hours per metric ton of concentrate charged, or 481 kilowatt-hours per short ton, or 0.055 kilowatt-year per ton. On the basis of total charge the energy consumption is 393 kilowatt-hours per short ton of charge or 0.045 kilowatt-year per ton. The theoretical energy consumption per pound of copper produced is 0.63 kilowatt-hour.

In experiment No. 10 the actual power consumption was 1,310 kilowatt-hours per ton of ore smelted, 1,280 kilowatt-hours per ton of charge smelted, and 1.77 kilowatt-hours per pound of metal tapped. As 1,310 kilowatt-hours per ton of ore smelted were actually used, whereas 481 kilowatt-hours per short ton are theoretically required, without allowing for heat losses or other inefficiency, the furnace used 2.72 times more power than was theoretically necessary, or it had a heat efficiency of 36.76 per cent.

The power consumption in experiments 1 to 13, tabulated according to the oxygen ratios of the slags, is as follows:

Power consumption in smelting tests of native copper concentrates.

Experiment No.	Ore used.	Flux.	Oxygen ratio. $\left(\frac{\text{Acid O}}{\text{Basic O}}\right)$	Power consumption—		
				Per pound of metal tapped.	Per ton of ore smelted.	Per ton of charge smelted.
				Kw.-hours.	*Kw.-hours.*	*Kw.-hours.*
4	Concentrate No. 4.	Silica........................	1.97	3.53	1,450	1,320
1	.....do............	None........................	1.85	4.05	2,425	2,425
3	.....do............	Silica........................	1.68	3.67	1,820	1,740
5	.....do............	.....do......................	1.60	6.30	1,600	1,330
7	.....do............	.....do......................	1.59	4.19	1,130	1,015
6	.....do............	.....do......................	1.28	4.75	1,320	1,075
2	.....do............	None........................	1.10	4.13	1,500	1,500
9	.....do............	Hematite....................	.87	4.73	1,565	1,425
8	Concentrate No. 3.	.....do......................	1.44	2.13	1,490	1,420
12	.....do............	Limestone and hematite...	.93	2.06	2,070	1,455
10	.....do............	Limestone..................	.91	1.77	1,310	1,280
13	.....do............	Limestone and hematite...	.82	2.77	1,780	1,100
11	.....do............	.....do......................	.70	2.68	1,880	1,565

As some of the charges were treated longer than was absolutely necessary, the power consumption varied and the results are not uniform. The power consumption per pound of metal tapped in some of the runs with No. 4 concentrate was high, owing to the low amount of metal tapped, but of course was less when concentrate No. 3, which contained a higher percentage of copper, was charged. A low power consumption per ton of charge was obtained with No. 4 ore in experiments 6 and 7, in which slags with oxygen ratios of 1.28 and 1.59, respectively, were produced, but the volatilization loss was so high as to prohibit the use of such charges. The lowest power consumption obtained in any of the experiments in which copper losses were not excessive was in runs 10 and 13, being 1,280 and 1,100 kilowatt-hours per ton of charge. Limestone flux was used for No. 10 and a mixture of limestone and hematite for No. 13, the oxygen ratios of the slags being 0.91 and 0.82, respectively.

In general it may be stated that the most satisfactory results, as regards both power consumption and copper losses, are obtained when the native copper concentrate is fluxed with limestone or hematite, or both, in such proportions that the slag formed approximates a monosilicate. With the small furnace used, which required 40 kilowatts—the average in 14 experiments—a heat efficiency of 36.76 per cent was obtained. A 750-kilowatt commercial furnace of modern design, equipped with feeding shafts and a covered top, when working on similar ore should have a heat efficiency of 70 per cent and possibly higher, as the process consists of simply melting the ore, but 70 per cent may be taken as a safe figure. In experiment No. 14, in which the furnace was run continuously on the same kind of charge as was used in No. 10, the power consumption was even lower than in No. 10, because of the continuous charging, being 1.76 kilowatt-hours per pound of metal, 1,190 kilowatt-hours per ton of ore, and 1,075 kilowatt-hours per ton of charge.

On the basis of 70 per cent efficiency a large 750-kilowatt furnace would consume only $\frac{36.76}{70}\times 100 = 52.5$ per cent as much energy as the small 40-kilowatt furnace in smelting the same amount of ore. Hence it may be stated that the following power-consumption figures should be obtained in smelting fine Michigan native copper concentrates, containing from 25 to 40 per cent copper, in a 750-kilowatt commercial electric furnace: Kilowatt-hours per pound of metal tapped, $0.525\times 1.76 = 0.92$; kilowatt-hour per ton of ore smelted, $0.525\times 1{,}190 = 625$; kilowatt-hours per ton of charge smelted, $0.525\times 1{,}075 = 564$.

OPERATION OF FURNACE.

No difficulty was experienced in operating the furnace on the resistance principle. The fineness of the material assisted in this considerably, as there was not so great a tendency for arcking with the particles of the charge packed closely. After the first few experiments it was found that the furnace worked best at 70 volts, and the voltage was kept at about 70 during later runs. The average power on the furnace was 40 kilowatts at a voltage of 73 volts and an amperage of 621 amperes. The power factor was high, averaging 0.95. As the furnace was over powered little difficulty was experienced in starting it. The fire-brick walls were not corroded appreciably by the charge, but when superheating was prolonged after the charge had been melted the fire brick melted in some places and mixed with the slag. This accounts for the variable nature of the slag as compared with that of the materials charged. The carbon bottom used in the first three experiments was not properly put in and was replaced by another rammed in harder, which lasted throughout the experiments, and was apparently not at all affected. Ground coke mixed with tar to a paste gives as good satisfaction as ground graphite electrode butts.

ELECTRODE CONSUMPTION.

The average electrode consumption throughout all of the experiments was 34.5 pounds of graphitized carbon per short ton of charge. The highest consumption, 48 pounds, was in experiment No. 1 and the lowest, 22.1 pounds, was in experiment No. 14. This last figure more nearly represents practical working conditions and could undoubtedly be attained, and even bettered, in practice. The atmosphere of the electric furnace is practically neutral during the melting of a material in which all of the iron oxide (FeO) is combined to form a silicate. For this reason the consumption of the electrode, although it would not be as low as in a reducing atmosphere, should not be excessive, because of the combined state of the ferrous oxide. In practice the electrode consumption should not exceed 10 pounds per ton of charge and probably would be much less.

From these results it is evident that the loss of electrodes through contact with the charge would not form a serious objection to the substitution of the electric furnace for the reverberatory furnace in smelting copper ores. If all of the electrode consumption were used in the reduction of oxides, the amount of energy consumed in supplying heat for reactions might be appreciable, but as an average of only 9.35 per cent of the total electrode loss was so used any loss of electrical energy in this way may be considered as negligible.

UTILIZATION OF SLAGS FOR PRODUCTION OF FERROSILICON.

In connection with the electric smelting of both oxide and sulphide ores of copper, the idea has always been prevalent that it is possible to utilize the slag for making ferrosilicon in the electric furnace, but not much actual experimental work has been done to determine the feasibility of the process.

To obtain information on the factors involved a series of experiments was undertaken.

Five experiments were made, using two lots of slag produced in smelting the native copper concentrates. The composition of these slags, as shown by analysis, was as follows:

Analyses of slags used.

Constituent.	No. 1.	No. 2.
	Per cent.	*Per cent.*
CuO	0.22	0.80
SiO_2	46.19	34.63
CaO	9.61	16.54
MgO	2.55	3.57
FeO	24.10	28.24
Al_2O_3	16.58	14.75

The coke that was used contained 84.38 per cent of fixed carbon, 0.34 per cent of moisture, 0.61 per cent of volatile and combustible matter, and 14.67 per cent of ash. The iron ore and silica were of the composition given on pages 21 and 22. The results of the tests are negative, but are given to show that the making of ferrosilicon from copper slags, which has been assumed by some writers to be possible without experimental confirmation, is a difficult matter.

EXPERIMENT WITH SLAG AND COKE.

In the first experiment, No. 15, 61.6 pounds of slag No. 1 and 17.6 pounds of coke were charged. There was some copper frozen in the furnace when it was charged, which partly accounts for the high percentage of copper in the product. An attempt was made to operate the furnace continuously, but both the metal and the slag were too thick to flow from the furnace, although several attempts were made to tap it during the run. At the end of the run 8.3 pounds of metal were tapped, but the slag would not flow. The slag was removed as far as possible by balling with iron rods, and what could not be removed in this way was chiseled out when cold. The total amount of slag obtained was 59.3 pounds. The metal was very brittle, looked like white iron, and had the appearance of ferrosilicon containing 25 per cent of silicon. Dense white fumes were given off during the run. The power consumption was 3.82 kilowatt-years per ton of metal tapped, or about 10 times the power required to produce 1 ton of "25 per cent" ferrosilicon from iron turnings, quartz, and coal in a commercial ferrosilicon furnace. Coke was charged in amount sufficient to reduce all iron and silicon charged. Possibly, however, some silicon carbide was formed, as this compound tends to form if an excess of coke be present. The effect of silicon carbide is to make the slag thick. The amount of copper in the metal was considerably greater than the amount charged, for the reason previously stated.

The product contained 21.23 per cent silicon, 58.01 per cent iron, 19.35 per cent copper, 0.02 per cent phosphorus, and a trace of sulphur. The slag contained 46.25 per cent SiO_2, 22.4 per cent Al_2O_3, 14.6 per cent FeO, 9.81 per cent CaO, and 2.70 per cent MgO.

EXPERIMENT WITH SLAG, QUARTZ, AND COKE.

In experiment No. 16 the charge consisted of 44 pounds of slag No. 2, 14.65 pounds of quartz, and 12.24 pounds of coke. The charge was calculated to give a product containing 50 per cent of silicon and a slag containing 40 per cent of SiO_2, on the assumption that all of the coke was used for the reduction. The results were similar to those obtained in the previous run; the slag fused, but would not flow from the furnace, the yield of metal was low, and the power consumption was high. The ferrosilicon contained 18.38 per cent silicon, 73.02 per cent iron, 7.01 per cent copper, 0.08 per cent sulphur, and 0.04 per cent phosphorus. The slag contained 49.63 per cent SiO_2, 27.24 per cent Al_2O_3, 9.76 per cent FeO, 9.85 per cent CaO, and 2.60 per cent MgO.

OTHER EXPERIMENTS.

Three other experiments were conducted in which even more negative results were obtained. In one, iron ore was added to ascertain whether the presence of iron in an easily reducible form would accelerate the silicate reduction, just as it lowers the reduction temperature of silica not combined as silicate, but no beneficial effect was obtained. In these runs part of the slag would flow from the furnace, but the product was a high-silicon pig iron. Fine grinding was tried, but it caused the material to pack so that the furnace could not be made to arc steadily, and the maintenance of a steady arc is necessary to attain a high enough temperature for the production of ferrosilicon.

DISCUSSION OF RESULTS.

In these particular experiments the production of ferrosilicon from slag in the electric furnace must be accounted a failure because of the low yield of ferrosilicon, due to the difficulty of reducing the silicate and to the tendency of the constituents of the slag other than iron and silica to rob the ferrosilicon of those elements. As a result of this, the power consumption and the difficulty of operating the furnace were too great for practical purposes. The remarks on the results of these experiments apply in general to the production of ferrosilicon from slags. The writers believe that it would be impractical to attempt the manufacture of ferrosilicon from a slag containing more than 10 per cent of oxide other than FeO or SiO_2 because of the clogging up of the furnace with the difficultly fusible slag resulting. With even 10 per cent of oxide present to form a slag with the FeO and SiO_2, it would probably not be possible to make a ferrosilicon containing over 30 per cent silicon. In the commercial production of ferrosilicon from iron turnings and silica in the electric furnace, "30 per cent" ferrosilicon can be made if some slag is present; a "50 per cent" product can be made only with difficulty, and higher grades not at all, if any slag is formed.

Even if the proper slag is available, the process, except under unusual circumstances, does not seem commercially attractive. A copper smelter is usually situated in a district remote from steel centers, where the greatest market for ferrosilicon is found. The slag, in order to compete with iron turnings, or iron ore, and quartz or sand as raw material for making ferrosilicon, must be cheap enough to offset the disadvantages of being possibly farther from the market, and requiring a higher power consumption.

CONCLUSIONS.

The conclusions that were drawn from the experiments described in the preceding pages are as follows:

1. Fine Michigan native copper concentrates can be smelted in the electric furnace with the production of a good grade of black copper and without excessive losses of copper in the slag or by other means.

2. The percentage of copper in slag of the proper composition need not be more than 0.25 per cent. The proportion of total copper charged that is lost in slag should not exceed 0.5 per cent, and all other losses should not exceed 1 per cent, or the total loss should be not more than 1.5 per cent.

3. The loss of copper by volatilization is high if the slag is much more acid than a monosilicate or if it contains much alumina. The lowest losses were obtained when limestone was added to form a slag containing 35 per cent of SiO_2, 22 per cent of CaO and MgO, and 25 per cent of FeO.

4. The difference between limestone and hematite as a flux is not marked, one being about as efficient for this purpose as the other if the slag is not of too high a specific gravity. No more iron was found in the black copper product when hematite was used than when limestone was used.

5. With a furnace operating at a low temperature on a monosilicate slag, the reduction of iron by the carbon electrode should not be excessive or sufficient to result in a product containing less than 95 per cent of metallic copper. With care a product containing 98 per cent of copper could probably be produced regularly.

6. The total electrode consumption greatly exceeded the amount of carbon necessary for reduction of the iron found in the metal, as an average of only 9.35 per cent of the total amount of electrode consumed was used in iron reduction. In practice the electrode consumption should not exceed 10 pounds per ton of charge.

7. The electric furnace operates satisfactorily with the highly siliceous-aluminous slag produced by the natural gangue without flux, but the losses by volatilization are high because of the high melting point of the slag. There was no copper oxide in the product, as was shown by microscopic examinations.

8. The power consumption for smelting fine Michigan native copper concentrates, carrying 25 to 40 per cent of copper, in an electric furnace of 750 kilowatts or greater capacity should not exceed 625 kilowatt-hours per ton of ore smelted.

9. The production of ferrosilicon from the copper slags can not be economically accomplished if the slag contains large proportions of components other than ferrous oxide and silica.

PROPOSED PROCESS FOR ELECTRIC SMELTING OF FINE NATIVE COPPER CONCENTRATE.

The commercial smelting of fine native copper concentrate in the electric furnace, in order to attain the greatest economy, would be a continuous operation. The process would consist of charging the mixture of ore and flux at intervals into the top of an electric furnace having a short stack above the smelting crucible for preheating the charge before it reached the smelting zone, and tapping the metal at intervals, the slag being permitted to run continuously from the furnace through a slag notch above the metal hole.

The possibility of operating a furnace in this manner was shown in experiment No. 14. There was an average of 1.10 per cent of iron in the product, and 1.37 per cent of copper in the slag. The total loss of copper was as low as in the intermittent runs and was largely due to the small scale of the operation. The loss of copper in the slag was high because of imperfect settling and because of the tapping of partly fluid material with the slags, a condition that would not occur in a large furnace properly operated. The power consumption was lower than in the other experiments.

DESCRIPTION OF FURNACE.

The furnace features essential to the success of this process are a crucible, in which the electrical energy is introduced, and in which the metal settles from the slag, and a short shaft above the crucible for preheating the charge before it reaches the smelting zone. As there is no reduction to be performed in the shaft it need not be higher than to give plenty of space between the top of the crucible and the charging floor for head room in moving the electrodes.

A furnace of the type shown in figures 3 and 4 satisfies these conditions. The furnace has a smelting chamber, *a*, lined with fire brick or silica brick; three carbon electrodes, *b*, suspended vertically through the roof; and a feeding shaft, *c*, which is lined with fire brick, over the crucible.

A 750-kilowatt furnace of this type would smelt in 24 hours about 23 tons of native copper concentrates, carrying 25 to 40 per cent copper. This furnace would be operated with three-phase current at from 50 to 100 volts. The internal dimensions of a 750-kilowatt furnace of this type are as follows: The crucible is 14 feet 6 inches in diameter and 9 feet high, the shaft is 1 foot 6 inches in diameter and 18 feet 6 inches high.

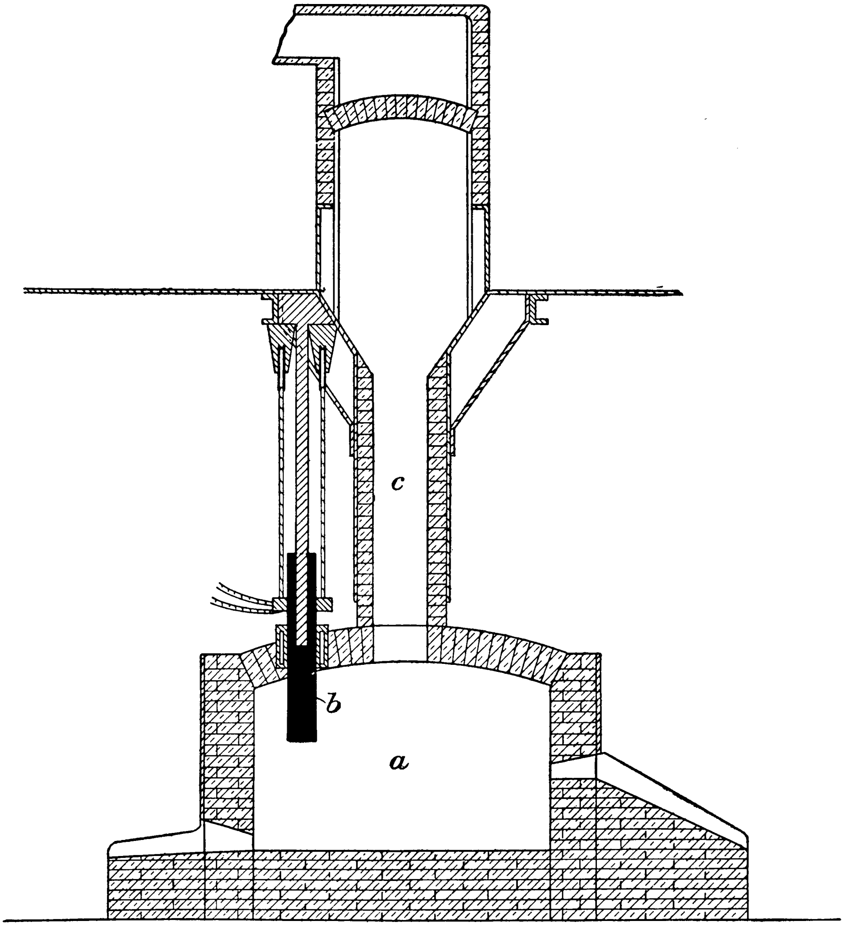

FIGURE 3.—Elevation of electric furnace for smelting native copper concentrates.

As regulation of electrodes would not be necessary in this type of furnace when in regular operation the electrical connections are made close to the roof of the furnace, and regulation is by hand. There is less difficulty in handling the electrodes if they are suspended vertically, and the vertical shaft eliminates the need of suspending the electrodes at an angle of about 55 degrees.

The slag from such a furnace should not contain more than 0.25 per cent copper, and could be discarded at once without further treatment.

It could be granulated with water as discharged from the furnace. The metal would require further refining for elimination of iron and other impurities, and could be charged directly from the electric furnace into a reverberatory furnace and refined as previously described.

COMPARISON OF PROPOSED AND PRESENT METHODS.

The present method of treating these concentrates consists of the following steps:

1. The coarse concentrate is melted and refined in either one or two reverberatory furnaces, from which a slag is produced containing from 10 to 30 per cent copper.

2. The fine concentrate is agglomerated in a reverberatory furnace, or briquetted with lime.

3. The slags from the reverberatory furnace and the agglomerated, or briquetted, fine material is smelted with coke in a blast furnace. The slags from the blast furnace contain 0.6 to 0.8 per cent copper and the metal 90 to 95 per cent copper.

4. The metal from the blast furnace is re-treated in a reverberatory furnace.

There would be the following advantages in using the electric furnace for smelting these concentrates: (1) No agglomeration or briquetting would be necessary; (2) none of the slags produced would require re-treatment; (3) a large proportion of the higher grade concentrates, containing 75 per cent copper, could be mixed with the fine concentrates and treated directly in the electric furnace, thus reducing the amount of slag necessary for resmelting; and (4) if cheap power were available the electric furnace could be used for reducing the copper from the reverberatory slags. The resulting metal would probably be lower in iron, and the slag lower in copper because of the greater ease of regulating the reducing atmosphere in the electric furnace, as the amount of coke is sufficient only for the reduction of the combined copper.

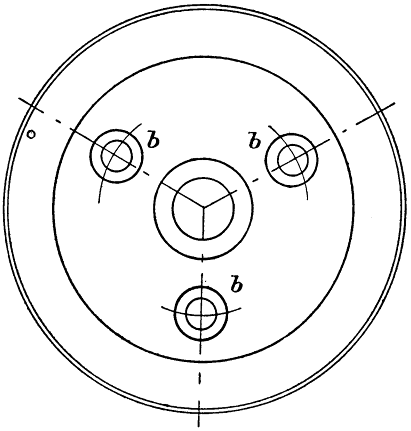

Figure 4.—Plan of electric furnace for smelting native copper concentrates.

The limit to the amount of higher grade concentrates that could be mixed with the fine material would be determined by the electrical conductivity of the mixture in the furnace. In the experiments performed, no short circuits or other difficulties were experienced in smelting a charge containing 37 per cent metallic copper. The point to which the proportion of copper in the charge could be increased would have to be ascertained by experiment.

Technically, the electric furnace does away with the necessity of briquetting or nodulizing the fine concentrates and in general facilitates handling of the materials. The commercial factors influencing the choice of the method to be adopted, as regards cost, would largely depend upon local cost of power and fuel. As regards present practice, the cost of fuel, the cost of briquetting or nodulizing, and possibly a higher labor cost are to be set against the cost of power for electric smelting. Even if the cost of power were equal to these three factors, the electric furnace might still have the advantage because of ease of operation.

In the case of a mine remote from smelting plants it might be cheaper to concentrate by smelting in a small electric furnace the low-grade product obtained by wet concentration and thus reduce the cost of freight. Also to save losses in wet concentration the ore might not be so highly concentrated by wet methods, a low-grade concentrate being made which could be smelted directly in the electric furnace, the high-grade black copper made being shipped to a custom smelter.

COST OF ELECTRIC SMELTING COMPARED WITH COST OF PRESENT METHODS.

Some reliable figures have been obtained by the bureau regarding the cost of smelting fine native copper concentrates in Michigan. These concentrates, containing 35 per cent copper, are agglomerated in a reverberatory furnace. One ton of concentrate yields 0.21 ton of metallic copper and 0.79 ton of slag containing 18 per cent copper. The slag is resmelted in a blast furnace and the copper is refined in a reverberatory and made into marketable shapes. On resmelting in the blast furnace, 0.79 ton of slag yields 0.152 ton of cupola block containing 88.9 per cent copper. The blast furnace slag averages 0.6 to 0.8 per cent copper. These blocks are refined in a reverberatory furnace, the yield from 0.152 ton being 0.129 ton of copper and 0.023 ton of slag. The copper is made into marketable forms and the slag is re-treated in the blast furnace.

Thus in treating one ton of concentrates a total of 0.813 ton of slag must be re-treated in the blast furnace.

The cost of smelting concentrates by the above method is given in the table following. The cost figures include overhead charges, all repairs, and maintenance of plant, but not amortization of investment.

Cost of smelting 1 ton of "35 per cent" native copper concentrate by reverberatory process.

	Per ton of concentrate.
Smelting 1 ton of concentrate in reverberatory furnace	$3.69
Making marketable shapes from 0.21 ton of copper, at $1.96 per ton	.41
Resmelting 0.79 ton of slag in blast furnace, at $4.71 per ton	3.72
Refining 0.152 ton of "cupola block" in reverberatory, at $2.77 per ton	.42
Making marketable shapes from 0.129 ton of copper, at $1.96 per ton	.25
Treatment of 0.023 ton of slag in blast furnace, about	.15
Total	8.64

For estimating the cost of treatment of these concentrates by the combined electric and reverberatory process, the electric furnace is assumed to be a 750-kilowatt furnace with a capacity of 28 tons per 24 hours. The cost of the plant, including cost of buildings, transformers, and furnace, is estimated at $20,000, which is a liberal estimate, as the furnace alone, including transformers, can be erected for $10,000. Reverberatory furnace smelting costs are taken as given in the preceding table. The furnace is assumed to be operating at a smelter where only a portion of the total superintendence, laboratory, and general costs would be charged against it.

The cost per ton of concentrate with power costs of 0.50, 0.625, 0.75, and 1 cent per kilowatt-hour is estimated to be as follows:

Estimated cost of smelting one ton of "35 per cent" native copper concentrate by the combined electric and reverberatory method.

Item.	Cost per ton of concentrate at a power cost per kilowatt-hour of—			
	0.5 cent.	0.625 cent.	0.75 cent.	1 cent.
Smelting in the electric furnace:				
Limestone, 0.10 ton at $1 per ton	$0.10	$0.10	$0.10	$0.10
Electrodes, 1 pound, at 4 cents per pound	.44	.44	.44	.44
Power, 640 kilowatt-hours	3.20	4.00	4.80	6.40
Labor, 3 men at $2 each and a boy at $1 per 8-hour shift	.75	.75	.75	.75
Superintendence, chemist, general expenses	.20	.20	.20	.20
Maintenance and repairs	.25	.25	.25	.25
Amortization and depreciation on $20,000 at 5 per cent each	.18	.18	.18	.18
Interest, on $20,000 at 6 per cent	.11	.11	.11	.11
Total	5.23	6.03	6.83	8.43
Smelting in the reverberatory furnace:				
Reverberatory refining of 0.352 ton of "cupola block"[a], at $2.77 per ton	.97	.97	.97	.97
Retreatment of 0.05 ton of slag from reverberatory	.25	.25	.25	.25
Cost of making marketable copper of 0.345 ton copper at $1.96 per ton	.67	.67	.67	.67
Total	1.89	1.89	1.89	1.89
Total cost of smelting ore per ton of concentrate	7.12	7.92	8.71	10.32

[a] Cupola blocks 98 per cent copper. Total loss of copper 1.5 per cent.

According to these figures the electric process with power at 0.5 cent per kilowatt-hour or $43.80 per kilowatt-year is $1.52 per ton cheaper, at 0.625 cent per kilowatt-hour is $0.72 per ton cheaper, and at 0.75 cent is $0.07 more expensive than the reverberatory process. The commercial operation of such a process does not require power of the great cheapness necessary in iron smelting to compete with existing processes. A rate of 0.5 cent per kilowatt-hour with a high load factor is possible in many districts. The load factor in this case would be above 90 per cent and the power factor not below 0.85. In practice the cost of refining the "cupola block" of the electric furnace would probably be considerably reduced over that of the blast furnace because of the higher percentage of copper in it.

Also by the electric process there would be only 0.023 tons of slag per ton of concentrate to be resmelted instead of 0.813 tons as with the present process.

SMELTING OF SULPHIDE COPPER ORES.

ORDINARY COPPER BLAST-FURNACE SMELTING.

Ordinary copper blast-furnace smelting—that is, smelting the ore with carbonaceous fuel—differs widely from pyritic smelting, which consists in smelting the ore with heat derived by the oxidation of the sulphur and iron contained in the ore itself.

As stated by Peters,[a] "The main characteristic of ordinary blast-furnace smelting is the employment of carbonaceous fuel mixed with ore as the principle source of heat." Such being the case, the problem that presents itself is to determine whether it is metallurgically feasible to substitute electricity for carbonaceous fuel as a source of heat and whether in so doing there will be obtained as favorable results chemically as it is possible to obtain in ordinary blast-furnace smelting.

PRINCIPAL FEATURES OF THE PROCESS.

The prerequisites and distinguishing features of ordinary copper blast-furnace smelting, as stated by Peters,[b] are as follows:

(*a*) The employment of moderately compressed air to effect combustion.

(*b*) The complete, or nearly complete, continuity of the smelting process, due to the fact that the ore and fuel are fed periodically on top of the charge column at the upper extremity of the furnace shaft and withdrawn continuously (or periodically) in a molten state from the lower portion of the furnace, so that the operation of blast-furnace smelting, as understood by copper metallurgists, is not only continuous but, theoretically speaking, perpetual.

(*c*) The actual contact of the ore and fuel, these substances being charged into the top of the furnace in alternate layers.

(*d*) The employment of carbonized or high-carbon fuel, such as coke, anthracite, or charcoal, to avoid the annoyance and danger arising from the presence in the ore

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[a] Peters, E. D., Principles of copper smelting, 1907.

[b] Peters, E. D., Op. cit., pp. 109, 167.

column of the volatile hydrocarbons and various products of distillation which would result from the use of wood or the ordinary coals containing volatile matter.

In ordinary blast-furnace smelting the atmosphere is distinctly reducing, owing to the presence of large quantities of C and CO, which have a greater affinity for O than have most of the constituents of the charge. Under normal conditions, however, this reducing action is not sufficiently powerful to reduce the oxides of iron to metallic iron, although it reduces Fe_2O_3 and Fe_3O_4 to FeO, providing there is SiO_2 present to combine with this base at the instant of its formation.

In the true pyrite blast furnace, however, where very little coke is used, the atmosphere may be regarded as neutral until the focus is reached. Here we first encounter the O of the blast, and the atmosphere is powerfully oxidizing, * * * but as the O has a greater affinity for the iron sulphide than it has for any other important substance that should ever reach the focus, and as there is—or should be—always more iron sulphide present than there is O to burn it, it follows that even the atmosphere of the focus of the pyrite furnace may be regarded as neutral toward a certain proportion of the iron sulphide, as well as toward various other substances that may come within its influence.

REMOVAL OF SULPHUR.

In the smelting of copper ores, sulphide ores in the main have to be dealt with and the problem consists in getting rid of the sulphides other than copper, and of concentrating the latter into a matte containing anywhere from 20 to 60 per cent copper. The first step, therefore, is to free the ore from its sulphur. Consider first how this is done in ordinary copper blast-furnace smelting. As pointed out by Peters,[a] owing to the constant presence of a large amount of glowing carbon in the furnace shaft, extending from the tuyères for several feet up the shaft of the furnace and often to the charging door itself, the oxygen of the air blown through the tuyères is consumed with extraordinary rapidity and thoroughness, and there is consequently but a poor chance for the oxidation of any substance in the ore that possesses less affinity for oxygen than carbon does.

Therefore, in ordinary blast-furnace smelting with carbonaceous fuel, the sulphides of iron and copper, and such small quantities of other metallic sulphides as may be present, stand little chance of obtaining any of the oxygen to combine with, to form sulphur dioxide and metal oxide. The atmosphere is almost always strongly reducing, and the sulphides tend to melt down with almost the same results that they would in a closed crucible, with complete exclusion of air.

It will thus be seen that, owing to the reducing atmosphere which prevails, we can, when smelting sulphide ores, look for the removal of but little sulphur in the blast furnace beyond the portion that will be driven off by heat alone without the presence of oxygen.

Of the two sulphides in the charge which have to be taken into consideration in this connection, namely, pyrite (FeS_2) and chalcopyrite ($Cu_2S.Fe_2S_3$), the former loses one-half of its sulphur content by mere heating without air, whereas the latter, when similarly heated, loses one-third of its sulphur content and one-ninth of its entire weight.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 109.

Continuing, Peters states:

> It is obvious, then, that there would be little object in smelting unroasted ores carrying a high percentage of metallic sulphides in a blast furnace running on carbonaceous fuel.

From what has been stated, it would appear that electric heat could be used as well for the melting down of the ore as heat derived from the combustion of coke.

However, such melting would not, in either case, give the concentration desired. The concentration of the copper and accompanying precious metals into a matte must be by the removal of sulphur and the subsequent oxidation of the iron.

The removal of sulphur in ordinary blast-furnace smelting is brought about in two ways, namely, by heat alone, which drives off a part of the sulphur as elemental sulphur, and from the reaction of oxides and sulphates, present in the ore as such, on the sulphides. The presence of oxides in the charge may be due to the use of natural oxide ores, whereas if roasted ores are used, both oxides and sulphates may be present. These components react with the sulphides present, a partial oxidation of the latter takes place, and some of the sulphur is removed as sulphur dioxide (SO_2). It is to be remembered that the atmosphere of the ordinary blast furnace is reducing, and therefore in this instance these reactions have to take place in a reducing atmosphere, and for this reason do not take place as freely as they would in a neutral atmosphere. In general it may be stated that with no oxides present in the charge practically no sulphur (aside from that removed as elemental sulphur by heat, as above stated) would be driven off except, as stated by Peters,[a] "the small amount that would be oxidized by the blast, in spite of the generally reducing atmosphere of the furnace." He also states that, judging from his own experience, this "loss of extra sulphur in the blast furnace can not be placed higher than 5 per cent of the sulphur remaining in the ore after deducting the sulphur which the pyrite and chalcopyrite lose by direct volatilization."

POSSIBILITY OF SUBSTITUTING THE ELECTRIC FURNACE.

Ordinary blast-furnace smelting of copper ores consists simply of melting down the ore, and the sulphur removed is that which is removed when a sulphide is heated without air. Any concentration that takes place is due to the reactions which take place between the oxides and sulphates present, and which would take place better in a neutral atmosphere than they do in a blast furnace, the atmosphere of which is reducing. Such being the case, a blast furnace operating in this manner is only a melting-down furnace, and the same results

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 116.

would be obtained by melting such an ore in an open crucible. The only advantage which the blast furnace has over the reverberatory furnace in this instance is that it is more efficient as a melting furnace than is the reverberatory, because it affords intimate contact between the fuel of the charge and the resultant gases and permits of continuous smelting. On the other hand the reactions between the oxides and sulphates do not take place as readily as they would in the comparatively neutral atmosphere of the reverberatory furnace, or in the still more neutral atmosphere of the electric furnace. So far, then, aside from the question of costs, there is apparently no reason why the electric furnace could not be substituted for the ordinary blast furnace in smelting the class of ores above mentioned.

LOSSES OF METALS.

According to Peters,[a] the losses which occur in ordinary blast-furnace work are (1) the loss of dust, which contains valuable metals, every time the ore is moved; (2) losses of dust in the flue gases, owing to the blast; (3) volatilization losses; and (4) losses in slag "due to particles of matte or metal which have failed to separate properly from the slag and, to a lesser degree, from the oxidation of the oxides of the valuable metals, especially lead and copper, that have entered into combination with the silica and have thus become chemically part of the slag."

The relative extent to which these losses would occur if an electric furnace (with no blast) were used instead of a blast furnace operated in the manner previously described will now be discussed.

Loss of dust in handling the ore would be common, of course, both to the blast furnace and the electric furnace.

Although there would be more or less fine dust carried away by the gases escaping from the electric furnace, the amount, of course, would not be as great as with the blast furnace.

As regards loss by volatilization, an electric furnace, used for smelting sulphide ore, would be operated as a resistance furnace (the charge acting as the resistor) rather than as an arc furnace, and hence the heat in the vicinity of the electrodes would not be as intense as if an arc were used, and probably not greatly in excess of that which is generated at the tuyères of such a blast furnace as has been considered.

If an electric blast furnace of the type shown in figures 5 and 6 be used, the volatilization would take place at the base of the shaft of the furnace, and hence the volatilized metals would have a chance to be deposited out on the cold part of the charge in the upper part of the stack, and as it would not be necessary to use a blast when operating

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 113.

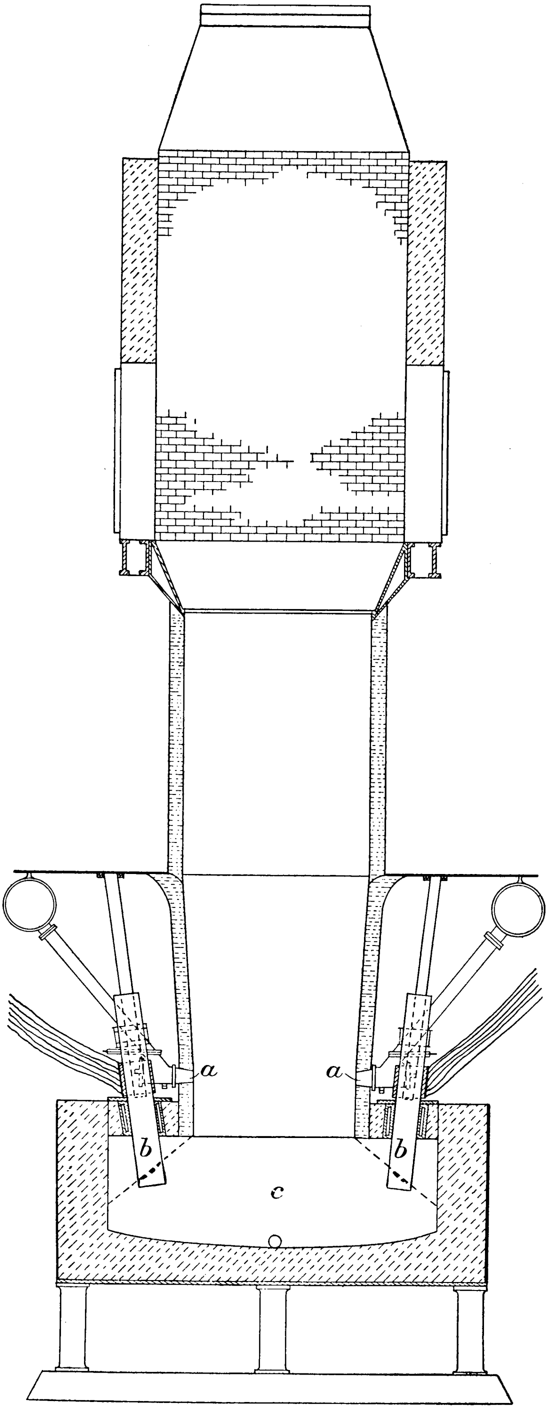

FIGURE 5.—Elevation of electric blast furnace. *a*, tuyères; *b*, electrodes; *c*, crucible.

the electric furnace for the purpose under discussion—the blast of the blast furnace in this particular instance is used only for furnishing air for burning the coke to produce the heat necessary for the process—the volatilization loss should not, and probably would not, be as great as in the blast furnace.

As regards the losses in the slag owing to particles of matte or metal failing to separate properly from the slag, such losses would occur in the electric furnace as well as in the blast furnace, but there is no reason to believe that they would be any greater and on the other hand they should not be as large, because the electric furnace could be regulated to impart a sufficiently high temperature to the bath before it leaves the furnace to insure its being fluid enough to cause perfect separation of matte and slag in the settler, which would thus obviate this loss. As to the oxidation of valuable metals, this loss should be reduced to a minimum in a furnace operating without a blast, for the loss by oxidation is doubtless due to the action of the blast upon the products of the furnace as they pass the region of the tuyères.

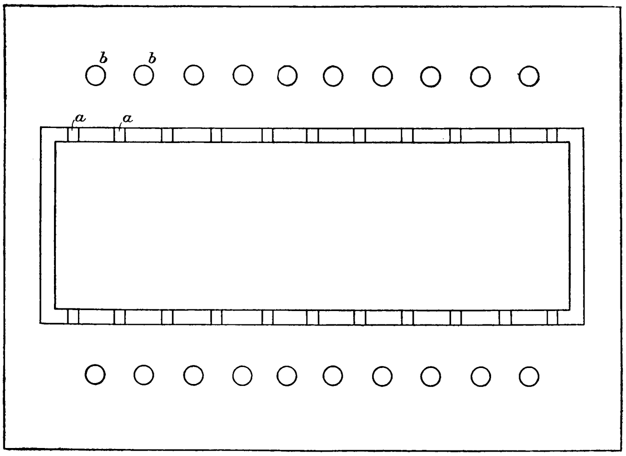

FIGURE 6.—Plan of electric blast furnace. *a*, tuyères; *b*, electrodes.

DATA ON EXPERIMENTAL SMELTING OF RAW SULPHIDE, OXIDIZED, AND CARBONATE ORES.

Having thus made a brief comparative study of the possibility of using an electric furnace for the smelting of mixed sulphide and oxide ores—it being understood, of course, that the object of this paper is not to prove that the electric furnace should replace the combustion furnace, but to determine, if possible, whether it can be substituted for the combustion furnace in those localities which are not favorable to the latter—it may be well at this point to consider some of the experimental work which has been done along the lines above indicated.

EXPERIMENT BY VATTIER.

One of the best known experiments on the smelting of copper ores in the electric furnace was made on April 23, 1903, by Vattier[a] at the works of the Compagnie Electrothermique Keller et Leleux, at Livet, France. The test was made by Vattier for the Chilean Government, which wished to procure such data as it would be possible to obtain in making experiments on a commercial scale. The test was made before a commission which was composed of some of the best known metallurgists of England and continental Europe.

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

METHOD AND RESULTS OF TEST.

The experiment was conducted with two kinds of ore, namely—

(1) An ore carrying approximately 7 per cent of copper which was present principally as copper pyrites, 8 to 9 per cent of sulphur, the gangue consisting mainly of micaceous copper oxide, together with some silicates and a little carbonate of lime; and

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

In the experiment a shaft furnace was used which had a smelting chamber 70 inches long, 35 inches wide, and 35 inches high. The melted material was run into a forehearth 47 inches long and 23 inches high. Two carbon electrodes, 11¾ inches square by 66 inches long, were used in the smelting chamber, and two electrodes 9¾ inches square by 39 inches long were put in the forehearth to reheat the bath for tapping.

The furnace used, so Vattier states, "Had a capacity of 25 tons of ore per 24 hours. The current used for the experiments was 4,750 amperes, 119 volts, cosine ϕ equals 0.9, and the energy corresponded to 500 kilowatts, or 680 horsepower."

The charge used was a mixture of the two ores. About 8,000 kilograms of ore was smelted in eight hours. The composition of the charge was as follows:

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[a] Haanel, E., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe, Mines Branch, Department of the Interior, Canada, 1909, pp. 215–223.

Composition of charge.

	Per cent.
Carbonic acid	4.310
Silica	23.700
Alumina	4.0
Lime	7.300
Magnesia	0.33
Sulphur	4.125
Iron	28.500
Manganese	7.640
Phosphorus	0.046
Copper	5.100
Arsenic	Trace.

Matte and slag of the following composition were obtained:

Analysis of matte and slag.

Constituent.	Composition of matte.	Composition of slag.
	Per ct.	*Per ct.*
Silica	0.80	27.20
Alumina	.50	5.20
Lime	....	9.90
Iron	24.30	32.50
Manganese	1.40	8.23
Sulphur	22.96	.57
Phosphorus	.005	.06
Copper	47.90	.10

DISCUSSION OF RESULTS.

Vattier remarks that the slags at both the beginning and end of the run usually contained a slightly higher proportion of copper. He does not say why this was so, or whether the copper present was free or combined. Uncombined copper would probably be present in larger amounts at the beginning of the run, because of the viscosity of the slag, due to lack of heat, preventing proper settling; also a slightly higher proportion of copper at the end of the run would probably be due to the same cause—that is, improper settling.

Vattier also states that the slag, being high in iron and silica, could be used to manufacture ferrosilicon, but, as has been previously pointed out in this bulletin, the results of experiments by the Bureau of Mines indicate that such utilization of slag is impractical.

Vattier also states that, "For good results it is desirable to use a voltage sufficient to cause the arc or electric current * * * to just clear the surface of the bath, in order to avoid as much as possible their (the electrodes) coming in contact with the bath."

His reason for wishing to keep the electrodes out of contact with the bath is explained by his statement that "Carbon at such high temperature has a tendency to reduce the iron oxide into metallic iron, which gives rise to the following inconveniences: (1) A more rapid wear of electrodes; (2) loss of electrical energy; and (3) decrease of the copper contents of the matte."

It is of course true that, if the electrodes extend down into the charge, as they do when the furnace is operated as a resistance furnace (the charge acting as the resistor) instead of an arc furnace, any iron oxide coming in contact with the carbon of the electrodes will be reduced by the carbon and thus cause a wearing away of the electrodes. On the other hand, in tests which have been conducted by members of the bureau and others, it has been found that more satisfactory results are obtained if the charge be smelted by resistance heating, using the charge as the resistor, than if it be smelted by heat from an arc. With the arc a very high local temperature is obtained and hence volatilization losses are higher than if resistance heating, which heats the charge more uniformly, is employed. The loss of electrical energy and the production of a low-grade matte resulting from the reduction of metallic iron has been previously discussed under experiments by the writers.

EXPERIMENTS BY SCHILOWSKI.

SMELTING MIXTURES OF RAW AND ROASTED ORE.

More recently some experiments were conducted by J. Schilowski [a] for the purpose of determining whether it would be possible to obtain in one operation a reaction between raw ore and roasted ore that would give a usable copper and a clean slag, Schilowski states that in these experiments he used a furnace which "had an upper carbon electrode and a water-cooled lower electrode," but he does not give much information in regard to the furnace beyond this. It was probably a small furnace, holding, perhaps, 25 pounds of ore, of the Siemens vertical type, shown in figures 1 and 2.

Analyses of the raw and roasted ore used in the tests are as follows:

Analyses of raw and roasted ore used in experiment.

	Raw ore.	Roasted ore.
	Per cent.	*Per cent.*
SiO_2	20.30	22.5
Cu	10.10	10.5
Fe	27.10	34.6
Al_2O_3	9.45	11.5
As	9.25	Trace.
Mn	.48	.5
S	15.09	1.5
Bi	Trace.	....
Co		....
Au		....
Ag	(b)	....

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[a] Schilowski, J., La fusion electriques des minerais de cuivre et des produits intermediares de fonderies de cuivre: Revue Met., vol. 9, 1912, p. 205.

[b] 0.116 ounces per ton.

The amount of copper unaccounted for was very large in most of the tests, especially when only roasted ore was used. The unexplained losses are probably due to volatilization and mechanical losses, especially the latter, for the gases arising from a small electric furnace carry with them a large amount of fine particles.

A second group of experiments was made to determine what proportions of raw ore should be used in the charge in order to obtain the largest percentage of copper in the matte with the minimum difficulty of fusion.

Schilowski states that in the experiments he used "two electrodes of carbon projecting into the slag." From this the authors judge that the fusion was done much the same as it would be if a charge were fused in a single-phase Héroult steel furnace.

DISCUSSION OF RESULTS.

As a result of his experiments, Schilowski states that the proportion of raw ore that should be used in making up a charge depends on the consumption of energy and the loss of copper in the slag.

He obtained these conditions with a charge containing 60 to 70 per cent of roasted ore. With such a charge, the loss of copper in the slag was 5 per cent of the total copper present in the charge, and the weight of the copper in the speiss was one-ninth of the weight of the copper in the matte.

He also found that the maximum energy consumption was when the charge was made up entirely of crude ore, and the minimum energy consumption was when the charge contained 64 per cent of roasted ore. He states that in treating a mixture of ores of this nature that 17,400 kilowatt-hours would be required per ton of copper produced.

He also states that the high losses of copper were due to the violence of the furnace reaction. The writers have not noticed that the reactions were especially violent in any of the experiments of this nature which they have performed. It is true that after the charge becomes melted it bubbles or boils considerably, due to the gases which are being given off, and it is also true that, in order to get a satisfactory separation of matte and slag, the fused product must be kept at a melting temperature for some little time, just as is done in a settler in commercial smelting. That this is not always done in experimental work may account for some of the high losses of copper in the slag.

RECOVERY OF ARSENIC FROM RAW ORE.

Another experiment was made by Schilowski in order to determine whether a recovery of arsenic could be made by fusing the raw ore.

METHOD OF TEST.

The fusion was made in a carbon crucible with a bottom carbon electrode. A carbon resistance of carbon block was placed in the crucible between the electrodes. A covering was placed around the crucible in order to prevent loss of heat by radiation. The gases were led off through a tube and then through several condensing chambers.

RESULTS OF TEST.

No speiss was obtained in this experiment. The matte contained 21.37 per cent of copper, 50.27 per cent of iron, and 28.39 per cent of sulphur. The slag contained only 0.15 per cent copper. In treating a quantity of ore corresponding to a ton of copper 27,500 kilowatt-hours were used per ton of copper produced.

An analysis of the fine material obtained as a result of the volatilization was as follows:

Composition of volatilized material.

	Per cent.
As	60
Fe	2
Cu	1
S	7
Miscellaneous	16

Schilowski states that in spite of the imperfections of the condensing apparatus, 65 per cent of the arsenic was recovered, and that it would be feasible to attempt the recovery of arsenic from the raw ore in this manner.

EXPERIMENTS BY WOLKOFF.

Wolkoff [a] made some experiments on the smelting of a copper sulphide ore, containing 8.20 per cent of copper, with an acid gangue. Several experimental runs were made with this ore. The results from one or two of them will be briefly discussed.

SMELTING ORE AND IRON OXIDE TO OBTAIN MATTE.

In one experiment, by smelting 12 kilograms (25.5 pounds) of ore, with 6.2 per cent of hammer scale (iron oxide) added to furnish iron oxide for the siliceous gangue, he obtained a thoroughly fused product, a matte containing practically the whole of the copper, and a slag retaining only 0.15 per cent of the copper. He also states that the volatilization losses were very small.

SMELTING ORE AND MATTE TO OBTAIN METALLIC COPPER.

In another experiment he smelted 10 kilograms (22 pounds) of ore with 1.25 kilograms (2.74 pounds) of roasted matte, containing 75 per cent CuO, 8 per cent Cu_2S, and 15 per cent Fe_2O_3, the object being to determine the extent to which the reaction $Cu_2S + 2\ CuO = 2\ Cu_2 + SO_2$, resulting in the production of metallic copper, would take place. On

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[a] Wolkoff, W., Electric smelting of copper sulphide ore: Metallurgie, vol. 7, 1910, p. 99.

smelting this charge with a current of 400 amperes at 75 volts for half an hour, Wolkoff obtained 1.78 kilograms (3.91 pounds) of metal which was 92 per cent copper, 3 per cent iron, and 1 per cent sulphur. He also obtained 10.9 kilograms (2.39 pounds) of slag with a copper content of 0.10 per cent. In this experiment 96 per cent of the copper was extracted; the slag contained 0.6 per cent of the total quantity of copper; therefore, as stated by Wolkoff, the copper balance is as follows:

	Kilograms of copper.
1.78 kilograms of crude copper (92 per cent copper)	1.638
10.9 kilograms of slag with 0.1 per cent copper	0.111
Losses	0.057
Total copper in charge	1.706

EXPERIMENTS BY STEPHAN.

At the first general meeting of the recently formed society of German Metallurgical Engineers, M. Stephan, superintendent of the Girod electric steel works, at Ugine, France, gave an account of some experiments made by him on the electric smelting of nonferrous ores, one of which was a copper ore. The following is an abstract of Stephan's paper:[a]

SMELTING NONFERROUS ORE IN THE ELECTRIC FURNACE.

The ore came from the Belgian Congo, being mined by a Belgian-English concern. In five different analyses given the CuO varies from 21.01 to 5.73 per cent; SiO_2 from 28.48 to 78.55 per cent; Al_2O_3 from 4 to 13 per cent; Fe_2O_3 from 4 to 16 per cent; and, besides smaller amounts of other impurities, 2 to 7 per cent CoO, and no nickel.

The moisture in the ore varied from 7 to 32 per cent. It was not removed, in order to meet the conditions of practical operation and to carry the experiments out under most unfavorable conditions.

Charcoal, coke, and anthracite were used successfully as reducing agents. Charcoal would probably be the cheapest in this particular instance.

Electric furnaces similar to the Girod ferro-alloy furnace were used, and the dimensions changed in several runs within wide limits in order to secure enough data for the construction of a large furnace for the same work. Electrodes were suspended from the top and inserted in the bottom, also in the sides for heating by radiation, and a system was adopted of heating with a smothered arc, mainly by the resistance of a thick layer of slag over the conducting metallic charge. The temperatures were measured with Le Chatelier and Fèry pyrometers.

A slag of the composition 51.9 per cent SiO_2, 11.31 per cent Al_2O_3, 16.83 per cent CaO, 13.71 per cent MgO, 3.55 per cent Fe_2O_3, 0.94 per cent MnO, 0.46 per cent CuO, and 0.87 CoO begins to melt at 1,250° C. and is liquid enough at 1,400° C. to allow the little copper balls to settle completely. At 1,550° C. the slag is liquid enough to flow freely; 1,920° C. were necessary to render the slag liquid when the highly acid charge was smelted without any fluxes.

The pig copper (Schwarzkupfer) produced in six different runs analyzed from 65 to 95 per cent copper, from 1 to 21 per cent iron, and from 1 to 11 per cent cobalt. The lower the smelting temperature the higher will be the purity of the product.

The lower the temperature the smaller will be the chance for reducing any of its impurities, but at the same time more copper will also be retained in the slag, so that the output is decreased.

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[a] Stephan, M. Einiges über die Erzeugung von Metallen im elektrischen Ofen: Metall und Erz, vol. 1, 1913, p. 11; abstracted in Met. and Chem. Eng., vol. 11, 1913, p. 22.

CONCLUSIONS OF STEPHAN.

Stephan's conclusions as to the results of the experiments were as follows:

> It will be a matter of commercial calculation, according to the composition of the ore and other special conditions, whether a high quality product or a maximum output will be desirable.
>
> A continuous run for several days with the same ore, aiming at slags of about the above-mentioned composition, required 1,000 to 1,200 kilowatt-hours per ton (2,205 pounds) of ore. These figures are high as a result of the amount of heat required to keep the very viscous and abnormal slags in liquid condition.
>
> With an easily fluxible ore the power consumption was only 500 kilowatt-hours per ton.
>
> The electrode consumption averaged 8 kilograms, or 17.6 pounds per ton of ore. It (the electrode) was operated with 4 amperes per square centimeter (25.8 amperes per square inch).
>
> Coal for reduction was used at the rate of 25 per cent of the copper in the charge.
>
> The best lining to withstand the severe conditions of the furnace was tamped from fire clay with 80 per cent SiO_2 and 15 per cent Al_2O_3.

EXPERIMENTAL SMELTING OF SULPHIDE ORES IN THE ELECTRIC FURNACE.

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

ORES TREATED.

The ores used in the experiments were a low-grade sulphide copper ore high in sulphur, a gold and silver bearing siliceous ore, and some roasted ore; the analyses of the ores are given below. Limestone containing 63.2 per cent CaO and 5.7 per cent MgO was used to flux the ore.

Analyses of ores used in electric smelting of sulphides.

Constituent.	Raw pyritic ore.	Nodule.	Siliceous ore.
	Per cent.	*Per cent.*	*Per cent.*
SiO_2	2.50	4.00	74.53
Fe	44.07	65.60	10.71
S	48.20	....	8.36
Al_2O_3	.24	1.76	.40
CaO	.08	.45	....
MgO	.73	.46	.26
P	.02	....	....
As	.20	....	....
Cu	1.30	.07	....
	Ounce per ton.	*Ounce per ton.*	*Ounces per ton.*
Au	0.01	0.03	1.28
Ag	.14	.05	3.20

METHOD USED AND RESULTS OF TESTS.

The ores were smelted in the electric furnace that was used in smelting the native copper concentrates. The furnace is described on pages 17 to 21. The top was roofed and kept closed tightly to prevent escape of sulphur and admission of air. A condenser, which consisted simply of three rectangular horizontal chambers with several baffles in them, was attached to the furnace to condense the sulphur and catch any escaping dust. Twenty experiments were made in all. The results of five are given in Table 3.

TABLE 3.—*Results of smelting copper sulphide ores in the electric furnace, experiments 1 to 5.*

	1	2	3	4	5	Average.
Charge:						
Raw pyritic ore, pounds	12.10	8.80	7.93	6.18	5.28	....
Nodule, pounds	9.90	13.20	11.90	19.22	7.81	....
Silicious ore, pounds	3.10	5.76	8.86	10.75	12.50	....
Limestone, pounds	....	2.73	3.01	3.02	3.03	....
Matte produced:						
Copper, per cent	1.03	1.22	1.33	1.05	1.00	1.12
Iron, per cent	63.99	64.18	64.18	56.21	62.49	....
Sulphur, per cent	24.87	22.38	25.47	20.28	24.03	....
Gold, ounces per ton	.28	.72	.80	1.60	2.33	....
Silver, ounces per ton	.68	.96	2.00	4.00	3.60	....
Slag produced:						
Copper, per cent	.05	.05	.24	.13	.29	.15
SiO_2, per cent	47.88	35.25	41.75	53.00	53.05	....
FeO, per cent	40.30	41.30	39.05	25.90	26.20	....
Al_2O_3, per cent	9.24	4.40	5.09	4.01	3.57	....
CaO, per cent	1.33	10.00	10.21	10.37	10.37	....
MgO, per cent	.10	1.64	.10	.14	.11	....
S, per cent	3.84	3.66	3.21	2.23	2.36	....
Gold, ounce per ton	.02	.03	.08	.04	.08	.05
Silver, ounce per ton	.16	.32	.17	.29	.34	.25
Length of run, minutes	35	26	27	20	18	....
Mean volts on the furnace	53	51	56	70	80	62
Mean amperes on the furnace	656	635	920	792	726	745
Mean kilowatts on the furnace	31	32	51	54	56	46
Mean power factor of circuit	.89	.98	.98	.97	.96	.95
Kilowatt-hours (by meter)	18.00	13.5	25.5	17.0	15.0	17.8
Electrode consumption per short ton of charge, pounds	20.2	10.8	18.6	18.6	18.6	17.3
Weight of matte tapped, pounds	[a] 13.45	10.50	7.00	8.42	9.69	....
Weight of slag tapped, pounds	4.74	10.90	16.90	17.84	19.70	....
Sulphur volatilized, per cent	65.9	63.2	57.0	65.1	54.6	61.7
Concentration ratio	1.57	2.84	4.16	3.60	3.85	3.2
Loss of copper in slag, per cent	1.21	4.26	(b)	(b)	(b)	2.73
Copper unaccounted for, per cent	8.48	....	(b)	(b)	(b)	....
Total recovery of copper, per cent	90.31	95.74	[b] 90.30	[b] 100.00	[b] 100.00	95.3
Loss of gold in slag, per cent	2.16	4.18	11.40	5.07	(b)	5.7
Gold unaccounted for, per cent	11.75	....	10.5	0	(b)	....
Total recovery of gold, per cent	86.09	95.82	78.1	94.93	95.2	90.0
Loss of silver in slag, per cent	6.28	17.20	9.40	....	16.20	12.2
Silver unaccounted for, per cent	15.40	32.90	12.30	....	8.37	17.2
Total recovery of silver, per cent	78.32	50.10	78.30	94.50	75.10	75.2
Total value of copper, silver, and gold per ton of charge,[c] dollars	4.96	6.76	8.95	11.14	12.90	....
Value of copper, silver, and gold recovered per ton of charge, dollars	4.29	6.34	7.21	10.62	12.17	....
Total value of copper, silver, and gold recovered, per cent	86.7	94.0	80.8	95.8	94.2	90.3
Kilowatt-hours per short ton of charge	1,435	890	1,610	1,064	1,045	1,209
Kilowatt-years per ton of charge	.164	.101	.184	.13	.12	.174

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[a] From previous run.

[b] In this and subsequent experiments the copper content of both matte and slag is high, probably because of a concentration of copper in the furnace from the numerous experiments made previously.

[c] Copper at 15 cents per pound, silver at 50 cents per ounce, gold at $20 per ounce.

DISCUSSION OF RESULTS.

In these experiments the furnace was charged and, when fusion was completed, tapped as cleanly as possible. Hanging of the charge and accumulation of material in the furnace are responsible for some results not being uniform. An attempt was made to make the results as quantitative as possible, but with a small furnace the losses affect the final calculation much more than with a furnace of commercial capacity.

SMELTING CONDITIONS, SULPHUR VOLATILIZATION, AND RATIO OF CONCENTRATION.

If the electric furnace acts simply as a melting agent in the smelting of sulphide ores, the fact will be apparent in the amount of sulphur volatilized and the ratio of concentration. In simple melting of a charge of pyrite one atom of sulphur would be driven off from the FeS_2, or 50 per cent of the sulphur in the charge would be volatilized. To determine the extent to which this reaction could be carried in the electric furnace, the crucible was roofed and was kept closed, as far as possible, in order to prevent oxidation. In the five experiments given in Table 3 an average of 61.1 per cent of the sulphur in the charge was volatilized, showing that a little more than one atom of sulphur was driven off. Hence without the introduction of an oxidizing agent the concentration ratio in the electric smelting of copper sulphide ores can not be increased much by volatilization of the sulphur in the charge.

The low ratio of concentration of 3.2, the average for five runs, verifies this statement and shows that no concentration is to be expected beyond a simple separation of gangue and matte and the loss of slightly more than one-half of the sulphur. As had been stated, there is no slagging of iron oxide beyond that of the iron oxide present in the ore as such, because there is no oxygen present to oxidize the iron sulphide formed by volitilization of the one atom of sulphur.

The smelting conditions in an electric furnace treating sulphide ore without introduction of air are then similar to those in a reverberatory furnace or a blast furnace in which the heat is supplied almost entirely by coke. The electric smelting of a copper sulphide ore under these conditions is merely a melting operation, in which reactions occur between the oxides and the sulphates of the charge and a removal by heat of a little more than one atom of sulphur.

The amount of sulphur driven off in the blast furnace, running with a reducing atmosphere, over and above that accounted for by the loss of one atom per molecule is not more than 5 per cent, whereas in the electric furnace, as these experiments show, it is about 10 per cent.

The condensation of the sulphur driven off as elemental sulphur was qualitatively possible, as was shown by analysis of the fume in the condenser, but was not quantitatively possible, because the condenser was not long enough for complete condensation of the fume, and some of it came out of the end. The uncondensed fume consisted chiefly of elemental sulphur. An average analysis of the condensed material was as follows: SiO_2, 13.96 per cent; Al_2O_3, 3.85 per cent; Fe, 9.11 per cent; CaO, 2.06 per cent; S, 46.6 per cent; Cu, 0.61 per cent; gold, 0.48 ounces per ton; and silver, 4.60 ounces per ton. A very small amount of dust from the ore was deposited at the mouth of the condenser.

COPPER, GOLD, AND SILVER LOSSES, AND LOW-GRADE MATTE AS A COLLECTOR.

The average copper content of the slag was 0.15 per cent. With proper means for settling the matte instead of tapping into a mold, the low copper content should be reduced to a still lower figure. The average percentage of copper recovered was 95.3 per cent. This shows that there was no material volatilization of copper, for the condensed fumes, which amounted to only 4.75 per cent of the total ore charged, contained only 0.61 per cent copper.

The percentage of gold recovered in the matte was 90 per cent of the total quantity of gold in the ore charged. The remaining 10 per cent was not all held by the slag, as the average loss of gold in the slag was 5.7 per cent of the total amount of gold in the ore charged. The other 4.3 per cent was probably lost mechanically in handling the charge and products. The slags contained 0.03 to 0.08 ounce of gold per ton. This indicates the collecting power of a low-grade copper matte for gold, as the copper content of these mattes varied from 1 to 1.33 per cent and averaged only 1.12 per cent. The charge contained from 0.31 to 0.83 per cent copper, the average being 0.46 per cent.

These results agree with the results of experiments by Dr. Carpenter[a] at Deadwood, S. Dak., where for several years the plant was run with mere traces of copper in the matte and never with more than 1 per cent. These slags carried from 0.025 to 0.075 ounce of gold per ton. As at Deadwood the experimental runs conducted by the writers show a lower gold loss in the slag with a greater matte fall. The results of these experiments seem to indicate that a matte containing 1 per cent copper will serve as a good col-

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 426.

lector of gold in the electric furnace and that a charge containing as low as 0.31 per cent copper may be successfully used.

The silver recovery was not as high as that of the gold, the total percentage of silver recovered being 75.2 per cent. A large part of the silver loss must be charged to the unaccounted loss, which includes mechanical losses and loss by volatilization. The proportion of silver in the fume was higher than the proportion of gold, a fact which, with the greater unaccounted loss, indicates that there was volatilization of silver. The average silver content of the slag was 0.25 ounce per ton, which can not be considered as very high, so that the low-grade matte seems to collect the silver as well as the gold.

In order to ascertain the actual saving of values, copper is figured at 15 cents per pound, gold at $20 per ounce, and silver at 50 cents per ounce. On this basis the average value recovery for five experiments was 90.3 per cent. This would indicate that it is feasible to use a matte containing 1 per cent copper in the smelting of an ore for recovery of gold and silver. With a higher matte fall the recovery would be greater. The loss in the slag seems to depend mostly on the proper separation of slag and matte. With an acid slag this separation is easy, but with a basic slag, as the specific gravity of the slag and that of the matte approach each other, separation becomes more difficult. As was stated by a recent writer on the subject, "Copper is not essential in mattes which accompany acid slags, but with basic slags it may be necessary."

ELECTRODE CONSUMPTION.

The average electrode consumption was 17.3 pounds per ton of charge, the lowest figure obtained being 10.5 pounds. In practice, with a large furnace the electrode consumption should be easily reduced to 5 pounds per ton of charge. The reduction of iron by the carbon electrode has been shown, in the experiments previously discussed, to be of small account.

POWER CONSUMPTION.

The power consumption was abnormally high in all of the experiments given in Table 3, with the exception of No. 2, because the runs were continued longer than was necessary, owing to the difficulty of telling when fusion was complete with the furnace top roofed. A power consumption of 890 kilowatt-hours per ton was attained several times in experiments not given in this bulletin and may be taken as being about the best it was possible to obtain with the furnace used. The efficiency of this furnace had been determined to be 36.76 per cent. On the basis of a properly designed furnace of several thousand kilowatts capacity in which an efficiency of 70 per cent should be attained, the power consumption would be (890×36.76)÷70=480 kilowatt-hours per ton of charge smelted, or 0.055 kilowatt-years. From the general trend of 20 experiments on the smelting of charges similar to those given in Table 3, there seems to be a slight increase in power consumption with a higher proportion of pyrite in the charge.

CONCLUSIONS.

1. The smelting of sulphide ores in an electric furnace without admission of air consists simply of melting the charge, volatilization of about 60 per cent of the sulphur as elemental sulphur, and separation of the slag and matte.
2. The ratio of concentration is simply that possible from separation of matte and slag, reaction of oxides with sulphates and sulphides, and the volatilization of about 10 per cent more than one-half of the molecular sulphur, with no formation of iron oxide to enter the slag, because there is no oxidizing agent present to oxidize the iron sulphide.
3. Qualitatively, it is possible to condense some of the elemental sulphur driven off.
4. The loss of copper by volatilization and in the slag is low.
5. A matte containing about 1 per cent of copper from a charge containing 0.30 per cent copper makes a good collecting agent for gold and silver, if a clean separation of slag and matte is obtained.
6. There is very little loss of gold by volatilization in the electric furnace, but some silver is lost in this way. In a larger furnace with closer temperature regulation there would probably be no loss.
7. The electrode consumption in the smelting of a sulphide ore is low and in practice need not exceed 5 pounds per ton of charge.
8. In a large commercial furnace the power consumption for most ores would be about 480 kilowatt-hours per ton of ore, or 0.055 kilowatt-years.

COMPARISON OF ELECTRIC FURNACE WITH REVERBERATORY FURNACE AND BLAST FURNACE AS A MELTING AGENT.

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

1. Native copper, oxide, or sulphide ores of copper can be melted just as efficiently, and perhaps even more efficiently, in the electric furnace than in either the reverberatory furnace or the blast furnace.
2. The reactions desired in reverberatory smelting, or ordinary blast-furnace smelting, can be obtained just as well and perhaps better in the electric furnace than in either of those furnaces.

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

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

5. A matte containing as low as 1 per cent of copper can be used as a collecting agent for gold and silver in the electric furnace as well as in the combustion furnace.

6. The cost would depend entirely on the nature of the ore to be treated and the relative cost of coal or coke and electrical energy. In general, from the work done in the experiments, it may be said that from 500 to 700 kilowatt-hours per ton of charge, depending on the nature of the ore, would be required for smelting copper ore.

SEMIPYRITIC SMELTING OF SULPHIDE ORES.

PRINCIPAL FEATURES OF THE PROCESS.

In discussing the smelting of copper ores by ordinary blast-furnace smelting, the fact has been noted that the process consists essentially in melting down the charge, and that during the melting certain reactions take place between the constituents of the charge which bring about a concentration of the copper and precious metals (if the latter are present) into a matte and the excess iron and the gangue materials into a slag, that in this method of smelting the heat necessary for carrying out the process is obtained by the combustion of coke at the tuyères, and that the only purpose in using the blast is to supply the oxygen required for combustion of the coke. In other words, the oxygen entering the tuyères plays no part in the reactions which take place in the furnace so far as the constituents of the charge other than coke are concerned.

Such being the case, the essential feature of the process is the melting down of the charge and, so far as the reactions of the process are concerned, the charge could be melted down just as well in an electric furnace as in a blast furnace. Those processes of copper smelting in which the air entering at the tuyères not only furnishes the oxygen necessary for producing heat, but also that required for bringing about the reactions which take place during the melting, and which result in the production of a matte and a slag, as in ordinary blast-furnace smelting, will now be considered. In this connection it is to be remembered that no matter what process is employed in obtaining metallic copper (as a final product) from a sulphide the three essential features of the process are: (1) The removal of sulphur; (2) the oxidation of the iron with which the sulphur was combined; and (3) the removal of this iron oxide by causing it to unite with silica to form a slag.

Hence, as stated by Peters[a], the first step in the smelting of a sulphide ore of copper must be to oxidize it. In the process that will now be considered the object is to bring about the required degree of oxidation and the melting at practically the same time, and to obtain either the whole or a part of the heat required for the process from the oxidation of the iron and sulphur contained in the ore. As is well known, if practically all the heat necessary for melting down the charge is obtained in this manner, the process becomes what is known as pyritic smelting, and if only in part, as semipyritic smelting. However, Peters[b] states that, so far as he is aware, in all pyrite furnaces of the world a small amount of fuel (usually coke) is added to the charge, and this amount may vary anywhere from 0.5 per cent of the weight of the charge up to a proportion that might be actually enough to melt the ore without any heat at all being derived from the sulphides.

SUBSTITUTION OF ELECTRIC HEAT FOR HEAT DERIVED FROM COKE.

The question, therefore, that presents itself in connection with the smelting of copper sulphide ores in the electric furnace is to determine whether electric heat may be used to replace the heat which is derived from the combustion of coke in pyritic and semipyritic processes. In other words, whether it would be possible, and if so, whether it would be commercially feasible, to carry out these processes in an electric furnace so constructed as to permit the use of a blast, thus obtaining as much heat as possible from the oxidation of the sulphur and iron, any additional heat required being supplied by electric energy.

To begin with, let it be assumed that the construction of the furnace is similar to that of a modern copper blast furnace and is practically the same for the upper part of the furnace, including the tuyères. Below the tuyères, *a*, the furnace could be constructed as shown in figure 5. The electrodes, *b*, extend down into the crucible, *c*, and are arranged along the sides of the crucible as shown in figure 6.

At this point will be considered briefly the behavior in the electric furnace of a charge that would be suited to semipyritic smelting; that is, one in which so much coke would have to be used that "its influence upon the oxidizing power of the focus begins to be plainly discernible," which, as stated by Peters,[b] is "the division line be-

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 220.
[b] Peters, E. D., loc. cit., p. 214.

tween true and partial pyrite smelting." Inasmuch as no coke would be used in a charge smelted in an electric blast furnace, the point to be determined is how the charge would act when such a furnace was used, and if the desired results could be obtained, as before stated, in a feasible and economical manner.

BEHAVIOR OF A CHARGE SMELTED WITHOUT COKE.

The object aimed at in operating a copper blast furnace in semipyritic smelting is to supply enough blast to completely oxidize the coke, and also to oxidize as much of the iron sulphide for slag-forming purposes as can be spared from the matte. If, therefore, there were no coke to be oxidized, only enough air need be forced through the tuyères to effect the required degree of oxidation of sulphur and iron.

In general, it may be assumed, as does Peters,[a] that in semipyritic smelting the furnace is "provided with but a scanty supply of pyrite" and that it produces a slag "rather high in silica and earths" and "consequently would require a considerable amount of coke." In the electric blast furnace, as before stated, coke would not be used, the equivalent heat value of the coke being obtained from electric energy instead. If, therefore, the electric blast furnace be supplied with a charge that is suited to semipyritic smelting, what difference may be expected in the behavior of the charge as compared to the behavior of a similar charge when smelted in a copper blast furnace with coke? In making the comparison the behavior and effect of the coke itself will first be considered.

EFFECT OF COKE IN THE BLAST FURNACE.

Peters[b] discusses the action of coke in the blast furnace as follows:

> The amount of coke present carries the melting process high up above the proper zone of oxidation, and to regions where there is yet no formation of FeO. Half-fused masses of acid earthy silicates are formed, and much, in some cases all, of the free silica is combined with the alumina, lime, magnesia, manganese, alkalis, and already oxidized iron, all of which substances are likely to be present in the ore mixture. It is not that the affinity of the silica is satisfied in forming these prefiminary, temporary, refractory acid silicates, but the edge of its appetite is blunted and the eventual formation of ferrous silicate seems to proceed somewhat sluggishly, even when ample air is blown into the furnace.
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> As may be imagined the main endeavor of the metallurgist in this type of smelting is to keep the proportion of coke to ore as low as possible, not merely because coke costs money, but still more because any excess of coke causes a lowering in the grade of matte, due to its interference with the oxidation of the sulphides. A furnace in proper condition and running on a suitable charge is extraordinarily sensitive on this point. The increase of the coke from a standard charge of 60 pounds per 2,000 pounds of ore up to 65 pounds per 2,000 pounds of ore may be followed—as has come under my personal observation—by a dropping of the grade of matte from 35 per cent copper and a raising of the silica contents of the slag from 41 per cent. This results, of course,

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 326.

[b] Peters, E. D., loc. cit., p. 333.

from the consumption by the new coke of a certain amount of oxygen which previously had been employed in burning iron sulphide. Under this new condition this iron sulphide entered the matte at the same time, robbing the slag of just so much FeO.

REACTIONS IN THE ELECTRIC FURNACE.

If such a charge be smelted in the electric blast furnace without coke and with just enough blast to supply the oxygen required for oxidation of the sulphur and iron, there would doubtless be no "half-fused masses of acid, earthy silicates, formed," as stated by Peters, and since, as shown by him, the control of the coke is a matter of great importance, due to the "extraordinary sensitiveness of the furnace on this point" and its resultant great irregularities, it would seem that it would be a great advantage to do away, if possible, with coke. If coke can be dispensed with and the smelting be done in some such furnace as the one shown in figure 5, the charge would probably enter the tuyère zone of the furnace practically unaltered except, perhaps, for loss of elemental sulphur. Inasmuch as the sulphides are easily melted (950° C.), it may be assumed that there would be enough heat present from the oxidation of the sulphur and iron and radiated from the crucible to cause the iron sulphide to melt and to become oxidized to FeO and then be instantly seized by SiO_2 to form a slag. However at a temperature of 950° C. the result might be the formation of a silicate high in SiO_2, due to the fact that at low temperatures the saturation point of silica for iron is low, and this silicate would perhaps "freeze," so to speak, at once and descend into the crucible and be gradually melted in the neighborhood of the electric current. In other words, there would be formed above the tuyère zone the artificial boshes observed by Sticht, Freeland, and others, and described by Peters,[a] thus contracting the smelting and slag forming area "to a comparatively narrow opening running along the middle of the furnace shaft." Sticht states that these boshes "are the contact lines between the active combustion zone and the relatively dead part of the shaft above" and that the absence of coke (speaking of pyrite furnaces) accentuates the sharp division line between the active and stagnant regions of the furnace. Reasoning by analogy, it would seem that this division line would be as pronounced, if not more so, in an electric blast furnace, such as the one under discussion, as in the pyrite furnace.

COMPARISON OF ELECTRIC BLAST FURNACE WITH THE COPPER BLAST FURNACE FOR SEMIPYRITIC SMELTING.

METALLURGICAL CONSIDERATIONS.

To recapitulate, there does not seem to be any metallurgical reason why the chief objects of the pyrite smelter can not be carried out in

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[a] Peters, E. D., Principles of copper smelting, 1907, pp. 234, 235.

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

MECHANICAL CONSIDERATIONS.

By referring to figures 5 and 6 it will be noted that the chief difference in the construction of the electric blast furnace from that of the ordinary blast furnace would be in the part below the tuyères; in other words, the crucible of the furnace. Due to the fact that electric furnace construction has received the attention of some of the very best mechanical and electrical engineers, and because a crucible based upon the principle of the design shown in figure 5 is now extensively used in the electric iron reduction furnaces of Norway and Sweden, the writers believe that no difficulty would be experienced in this respect. Various methods and arrangements can be used for connecting up electrodes. If this be true, there remains to be considered only the matter of costs.

COMPARISON OF COSTS.

Inasmuch as it has already been stated that the use of the electric furnace as a competitor of the blast or combustion furnace, or that it should replace the copper blast furnace, is not advocated by the writers, it will be evident that a comparison of costs is made simply for the purpose of giving some idea as to the outlay that would be necessary to erect and operate an electric furnace plant for the purpose of smelting copper ores.

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 334.

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

COST OF PLANT.

The cost of an electric-furnace plant would be about the same as the cost of a regular plant plus the cost of electrical installation but exclusive of the cost of the generating plant, as it is assumed that it would be possible to purchase electric power from some other company, or, if not, that the generating plant be considered as a separate organization selling power to the smelter at so much per unit. Therefore by electrical installation is meant the cost of transformers, bus bars, cables, instruments, etc. In order to have a basis for computing this cost, it is assumed that the furnace is to smelt 384 tons of charge a day, the composition of the charge being as follows:

Composition of charge.

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

This is the average composition of the entire blast-furnace charge that was smelted at the Washoe smelter, Anaconda, for several months.[a] At the Washoe plant 8.2 per cent of coke was used. The amount of electrical energy that would be required to smelt a charge of this nature in an electric furnace will now be considered.

At the Washoe smelter the furnaces formerly used were 56 by 180 inches in cross section (hearth area 70 square feet) and smelted on an average a little less than 400 tons of charge daily, or 5.6 tons per square foot of the hearth area.[b]

AMOUNT OF ELECTRICAL ENERGY REQUIRED TO REPLACE COKE.

Since 8.2 per cent of coke was required at the Washoe smelter to smelt 1 ton of the above charge, the consumption of coke per ton of charge was $2,000 \times 8.2 = 164$ pounds.

The coke used contained 80.24 per cent of fixed carbon; therefore, the number of pounds of carbon in 164 pounds of coke is $164 \times 80.24 = 131.59$. If 1 pound of carbon be completely burned to carbon dioxide, 8,100 pound-calories are obtained. Therefore the number of pound-calories that would be obtained from the 131 pounds of carbon contained in the 164 pounds of coke is $131 \times 8,100 = 1,061,100$. As 1 kilowatt-hour is equivalent to 1,897 pound-calories, it would

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[a] Peters, E. D., Practice of copper smelting, 1911, p. 264.

[b] Peters, E. D., Principles of copper smelting, 1907, p. 156.

require 1,061,100÷1,897=559 kilowatt-hours to replace the theoretical calorific value of the 164 pounds of coke. However, as a copper blast furnace under the most favorable conditions probably does not have an efficiency of over 50 per cent, the amount of energy really obtained from the coke, so far as work performed in the furnace is concerned, is only 530,550 pound-calories, which is theoretically equivalent to $\frac{530,550}{1,897}$, or 280 kilowatt-hours per ton. Inasmuch as the efficiency of the electric furnace of this type may be as much as 85 per cent and as that of an open-top ferrosilicon furnace is said to be about 60 per cent,[a] it is fair to assume that the efficiency of an electric furnace, such as is shown in figure 3, would be 20 per cent greater than that of an ordinary blast furnace, or about 70 per cent as compared with 50 per cent for the blast furnace. If this be true, then the theoretical 280 kilowatt-hours needed to smelt 1 ton of charge would have to be increased 1.42 times (280×1.42=397), or, say, in round numbers, 400 kilowatt-hours would be required to smelt 1 ton of charge. Considering that part of the heat in this case is supplied by combustion of sulphides, the figures agree closely with the 480 kilowatt-hours calculated from the experiments on melting only. As 16 tons of charge per hour are to be smelted, the constant load on the furnace would be 400×16=6,400 kilowatt-hours, and as the power supply would doubtless be three phase, this would mean that the electrical equipment should consist of three 2,500-kilowatt variable voltage transformers. Transformers of this capacity would allow ample margin for sudden overloads, instruments, bus bars, cables, and miscellaneous supplies.

The cost of this equipment at the plant would, of course, depend upon the location of the plant and would be the factory costs plus transportation charges.

As to the total cost of the plant, that would also, of course, depend, as in every instance, upon the location of the plant. In short, as before stated, it would be the cost of an ordinary copper blast-furnace plant plus the cost of the electrical installation. The cost of the furnace itself would be no more than that of a coke blast furnace. As only enough air is blown in to oxidize the iron and sulphur, the blower capacity would not need to be so large as for a regular blast furnace.

COST OF SMELTING.

It may be stated that the cost of coke and the cost of electrical energy are about on a par when coke costs $7 a ton and electrical energy costs $0.15 per kilowatt-hour, or $13 a kilowatt-year; that is, when the ratio of the cost of coke per ton to the cost of power

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[a] Conrad, W., and Piels, W., La fabrication du ferrosilicium à haute teneur au four electrique: Revue Mét., vol. 9, 1912, p. 362.

per kilowatt-year is about 1:1.8. Any advantages which there may be in the use of an electric blast furnace instead of an ordinary blast furnace, such as the saving in blowing by reason of not having to furnish air for burning the coke, will be disregarded, and it is assumed that the cost of electrodes and of the electricity on the one hand, and the cost of its equivalent in heating value of coke on the other, would be equal; in other words, that the cost of electrodes and electricity would balance the cost of the amount of coke that would be necessary to smelt the same quantity of ore.

Cost of electrodes.—The average consumption of electrodes in smelting iron ore at Trollhättan, Sweden, is about 10 pounds per ton of iron produced. Assuming that 1 ton of iron is equivalent to 2 tons of iron ore, the consumption of electrodes per ton of charge smelted in a furnace such as the one under discussion would not be over 5 pounds per ton. If it is assumed that the electrodes do not cost over 6 cents per pound, which allows a fair margin for transportation over their cost at the factory, the cost of electrodes would be about 30 cents per ton of charge smelted. The consumption would probably be considerably less than 5 pounds per ton, as in an experiment performed by the writers in which air was blown into the furnace, the electrode consumption was only 2 pounds per ton of ore treated. Such being the case, if the combined electrical energy and electrode cost is not to exceed the equivalent coke cost, with coke at $9 a ton, the cost of electrical energy would be 9×1.8=$16.20 a kilowatt-year, minus the cost of electrodes, or about $16. Compared on the basis of calories, if the coke contains 80.24 per cent fixed carbon, it would contain 1,604.8 pounds of carbon per ton of 2,000 pounds, and this carbon would, if completely burned to CO_2, give a calorific value of about 13,000,000 pound-calories. As one kilowatt-year of electric energy is equivalent to 16,616,000 pound-calories, it would require something like 1.2 tons of coke containing 80.24 per cent fixed carbon to equal in calorific value a kilowatt-year, if it be assumed that all the carbon of the coke is completely burned to CO_2, a condition which rarely occurs in practice. It has been determined from actual practice that from 1.5 to 1.8 tons of coke are required to yield the same number of useful calories as may be obtained from 1 kilowatt-year of electrical energy.

THE USE OF THE ELECTRIC BLAST FURNACE IN THE SMELTING OF PYRITIC ORES.

As was previously stated, in all true pyritic smelting a certain amount of fuel, usually coke, that varies with the nature of the ore is added to the charge. Assume, for the purpose of this discussion, that the ore being treated is entirely suited to pyritic smelting and that it is not necessary to add more than 0.5 per cent of coke to the charge, could such an ore be treated as well in an electric blast furnace as in an ordinary furnace, assuming that the costs would be the same in each case? As what has been stated in regard to the construction of the furnace and the costs of applying the process is as applicable to true pyritic as it is to semipyritic smelting, the discussion will be limited to the metallurgical and chemical features involved.

The nature of the atmosphere in a pyrite furnace will first be considered. As the air enters the tuyères practically all of its oxygen enters into immediate combination with sulphur and iron to form SO_2 and FeO. Although some oxygen escapes combination, the amount is not sufficient to support combustion within the furnace, as has been shown by Sticht and others.[a] So much, then, for the nature of the atmosphere within the furnace.

The changes which take place in the charge as it descends from the charging door to the fusion zone are few and may briefly be stated as follows:

1. The iron sulphide loses one-half of its sulphur as elemental sulphur and becomes FeS.
2. The chalcopyrite loses about one-fourth of its sulphur and becomes practically a mixture of Cu_2S and FeS.
3. The silica is unchanged.
4. The coke is for the most part consumed in the upper zone of the furnace, not by the oxygen of the air, as no free oxygen exists in that part of the shaft, but by the O from SO_2; that is, the carbon of the coke reduces the SO_2, the products of the reaction being S and CO, and this CO reacts with more SO_2, forming S and CO_2. Aside from these reactions, which are of no great importance in the chemistry of the process, the chief office of the coke, as pointed out by Sticht,[b] "is apparently to heat up the sulphides and the quartz in preparation for their active oxidation deeper in the furnace." If, therefore, coke was omitted from the charge, this preparation of the sulphides and the quartz would not take place, and it becomes necessary to determine how detrimental to the process this lack of preparation would be.

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

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[a] Peters, E. D., Principles of copper smelting, 1907, p. 225.

[b] Peters, E. D., loc. cit., p. 227.

one-third of the heat required to melt the entire pyrrhotite contents of the charge." This extra heat "doubtless furnishes just the necessary aid to bridge the operation over some critical point." From this it is evident that the office of the coke, then, is solely for the purpose of furnishing heat. Inasmuch as the nature of the charge in true pyritic smelting is such as to demand the oxidation and removal of a large excess of iron as silicate; and as it is necessary to cause the silica present to unite with as much FeO as possible, due to the fact that as a rule silica-bearing materials, which often are barren and hard to obtain, have to be added for slag-forming purposes, and as the saturation point of silica for iron oxide is greater, the higher the temperature attainable in the pyritic furnace; it is quite necessary that the reactions in the melting zone of the furnace proceed as energetically as possible in order to meet these requirements and to provide enough heat to keep the process going. If, therefore, pyritic smelting be attempted in an electric blast furnace it would be necessary to supply the heat from the electric current in such a manner as to insure there being enough heat at the focus to carry on the reactions at that point in the same manner as if coke had been added to the charge. The writers are of the opinion that it would be possible and perfectly feasible to do this in a furnace of the type shown in figure 5. Moreover, the use of electric heating would permit of a more uniform operation of the furnace than is now possible, for with the electric current it would be an easy matter to quickly increase or decrease the heat required for the successful operation of the furnace. In this way there could be maintained the "degree of concentration at a constant" which, as pointed out by Sticht, is very important, but is extremely difficult to do. Also the troubles could be avoided that arise from the constant fluctuations "which are bound to occur and which at present demand unbroken attention and frequent changes in the charge, especially in its proportion of silica, due to the fact that in true pyrite smelting there is no heat to spare," because with the electric blast furnace it would be possible as just stated, to increase or decrease the heat whenever it would be necessary to do so in order to meet these fluctuations.

CONCLUSION.

The electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the combustion of carbon. Inasmuch as the carbon which is used either in the reverberatory furnace or in the blast furnace plays no important part in the reactions that take place in these furnaces, there is no reason metallurgically why electric heat may not be substituted for the heat derived from the combustion of carbon. In fact, as the authors have attempted to point out, in some cases the reactions would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partly reducing atmosphere of the combustion furnace. Therefore, the practicability of using the electric furnace for the smelting of copper ores would largely depend on the relative cost of coke and electric power. As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it, in those localities where it is not advisable, because of the high cost of fuel, to use the combustion furnace, there is no apparent reason why the electric furnace may not be developed as a substitute for the combustion furnace where the conditions are such as to warrant its use, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the development of the electric furnace in the iron industry for the reduction of iron from its ores was due to necessity. As a matter of fact, the field for the electric furnace in the reduction of iron from its ores is a limited one. Perhaps the same is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs, as shown in the preceding pages, it would seem that the possibilities of the electric furnace for the treatment of copper ores are greater than those for the treatment of iron ores, because there is not so great a difference between the cost of coke and of electric power in copper-mining districts as there is in iron-smelting centers. Also the cost of electric power is constantly becoming less, through improvements in gas engines and steam turbines, so that in districts where water power is not plentiful but cheap fuels unsuited to coke making are available, it may be found more advantageous to use electric heat than the heat derived from the combustion of coke.

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

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