

A39

CERRO DE PASCO CORPORATION • NEW YORK
INTER-DEPARTMENT MEMORANDUM

Date: May 19, 1952


To: R. P. Koenig

From: L. Addicks

Subject: Electric Lead Smelting.

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2. I suggest that Mr. Smith be commissioned to make a brief review of the preliminary work we have made along these lines and then present a recommendation as to inaugurating an experimental campaign on the proposed 1500 KW dross furnace to be temporarily assigned to this work after providing it with a bag house. The first answer is merely whether there is a saving in electric power vs. coke when making present products sufficient to warrant such a substitution. Whether or not zinc fuming is to be considered in the operation is for the moment a complicating factor to be evaluated afterwards. I suppose in theory it is possible to push the idea over into a complete Sterling Process conception, but take one thing at a time.


Lawrence Addicks

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Lawrence Addicks

LA:JC

A-389.

51 Maiden Lane.

New York, N. Y., November 25, 1925.

Mr. C. V. Drew,
Cerro de Pasco Copper Corp.,
15 Broad St., New York, N. Y.

Dear Mr. Drew:

ELECTRIC FURNACES:

The questions raised in your letter of November 19th are so large that I think I had better let a comprehensive reply go until the next time I see you. In general I offer the following comments:

There is undoubtedly a field for the electric smelting of copper concentrates. In fact I conducted a series of experiments in conjunction with Fitzgerald at Niagara Falls for the Canada Copper Co. along these very lines several years ago. It is very improbable, however, that any such application could be founded on a heating basis against other fuels with the possible exception of in Norway. The real advantages lie in being able to run at a temperature high enough to permit using types of slag quite unworkable by ordinary methods thereby saving fluxes and in being able to operate economically on a small scale permitting installation directly at the mine with a saving of freight due to the smaller bulk of the product. I am not very familiar with some of your operations in Peru but I take it that you have no such conditions to consider.

The general problem is stated for smelting in the new edition of Hofmann's "Copper Metallurgy" on page 282, and for refining in my

Mr. C. V. Drew, - - - - - Sheet #2. - - A-389. - - Nov. 25, 1925.

"Copper Refining", page 138.

As to the possibilities of refining your blister in the mixer, Chrome is just starting the plant I advocated for producing anodes direct from their blister and you will very shortly be able to get exact operating costs on such work. With your material anodes could not be used on account of the richness in silver, and Chrome to date does not seem very anxious to make any allowance in the contract for a purer bullion. The saving to them at any rate by having you eliminate lead and antimony would probably not exceed a dollar a ton.

On the sampling side I believe we shall be able to come to a satisfactory conclusion without refining in Peru. In any event I do not see that electric operation could show any saving beyond KWH + electrode consumption vs. fuel oil, and I should hesitate to put bullion of your richness in silver in a smothered arc furnace.

However, I shall be glad to discuss this whole matter the next time I see you.

Yours very truly,

L. A.
B.

Adx/B.

CONTENTS NOTED
DEC 4 1925
LAWRENCE ADICKS

Cerro de Pasco Copper Corporation

*Office of C. V. Drew,
Room 4-5th floor,
Telephone Hanover 1684-5-6*

*15 Broad Street,
New York,*

*Cable Address:
"Cerrocop, New York,"
"Cerrocop, Lima."*

November 19, 1925

Dictated by

Dear Mr. Addicks:

Our inquiry for electric furnaces to melt down special sample lots of copper bars has produced a rather formal proposition from one Frank Hodson, who thinks he could design a 60 to 100 ton electric furnace in which we could reheat all of our molten blister copper very cheaply and thereby remove many of the impurities, making a very uniform and homogeneous bar with very little segregation. What do you think of it? Herewith copy of his letter of November 18.

Yours truly,

C. V. Drew

To
Mr. L. Addicks,
51 Maiden Lane,
New York City.

enc.

FRANK HODSON
200 Jefferson Building
Philadelphia

November 18, 1925

Cerro de Pasco Copper Corp.,
15 Broad Street,
New York City.

Gentlemen: Office of Mr.C.V.Drew, 5th floor
 Subject: Electric Furnace

With further reference to our conversation yesterday, I thought you might like me to put on record what I said with regard to use of electric furnaces in copper metallurgy and as applied to your particular needs.

Electricity has not yet been very much used in the preliminary stages of copper smelting - chiefly because other forms of fuel are usually cheaper nearer the mines than electricity. Where electricity can be generated hydro-electrically near the scene of smelting, there is no reason whatever why it cannot compete with any other fuel, and it has considerable advantages over most existing methods. One commercial sized electric furnace has been operating in Norway on Smelting of Copper Matte for a number of years and with cheap power the use of such furnaces could be considerable extended.

My own opinion is that one of the largest fields for the future use of electricity in copper metallurgy will be the large furnace - taking how charges from converters or other furnaces - acting as a Metal Mixer, and into which the smaller charges from the Converters are poured as ready.

The bath would be an open one and refining or other slags could be added. The furnace atmosphere could be regulated at will - also the temperature.

The furnace could either be arranged to empty completely at one operation or to pour continuously.

Such a furnace could be limited in size only by the amount of power available, but a furnace of 60/100 tons holding capacity would be comparatively simple to design.

The electrical heat would only need be sufficient to replace heat losses by radiation, plus any higher temperature that might be needed for the slags or refining operations.

I believe the kilowatts hours used would run between 100 and 200 per ton of copper treated, certainly not more than 200 KW hours. The resultant product would be exceedingly uniform in composition, free from gaseous inclusions, blowholes, etc. The electric furnace would be practically a closed container for heating - all fumes would be collected, and I believe you would produce a much higher grade of raw copper than is possible by any other process. A rough estimate of the cost of a 60-ton capacity furnace, capable of handling say 300 tons of molten copper per day, allowing for 5 complete renewals of the bath contents, would be \$90. to \$100,000. You would need 2500 KW of power available behind the furnace, although this quantity would not be actually used.

Your final product could be a very pure solid copper bar, ingot, slab or whatever other form you desired. I do not think there would be any difficulty in regulating the analysis in the electric furnace, so that you always poured approximately the same analysis material. No doubt you have ores or metals richer in non-copper constituents

available and adjustment of analyses by these could be made to get uniform results from the electric furnace.

I do not know whether any specially selling point could be made of electric furnace copper bar, but I would imagine so. It would certainly give better results than ordinary blister bar for many purposes.

Coming back to your immediate problem of the 60-ton sample lots - the big furnace would do this and give a very uniform product. A continuous casting machine can be designed to produce bars any size we wish and drillings from these would give absolutely uniform results and show what is actually in the metal.

If you do not feel inclined to consider the initial outlay for a large furnace of this kind, a specially designed induced current closed electric furnace of 600 to 1000 lbs. per hour capacity would give absolute uniformity of each cast - by judicious mixing of the 60-ton of samples before electric melting, you should get fairly homogenous finished 60-tons. It could be cast into small ingots or bars, the time taken by 100 lbs. of copper to cool being the only limitation of the size of cast bars. A furnace of this kind would probably cost you around \$15,000 and it would take 200 KW of power.

A simple standard type of melting furnace with direct or indirect arcs would not cost more than half the above, but I do not think it would give you anything like the same results as either the large melting furnace or the induction furnace just mentioned.

What I would very much like to do for your company would be to take on a consulting job to thoroughly investigate present methods of manufacture and then to make you a full report on the possibilities and probabilities of the use of electricity - the cost of type of plant to be employed - working costs and where and how you might expect to benefit. The use of electricity as a fuel has made immense strides in practically every metallurgical industry, except copper. My own 15-years practical experience in design of electric furnaces of all types and for all kinds of purposes, would fit me very well to know what to look for and what to disregard. I am a practical metallurgist, have done research work on a number of steel and electric problems, and can very soon get the scientific and practical aspects of copper industry. The general ones, of course, I know, but my work has not brought me in such close touch with the non-ferrous metals, as it has with the ferrous metals, alloys, etc.

Because cheap electricity is not usually available near copper mines and as electro-metallurgists have been busy on the nearer problems of iron, steel, aluminum, calcium carbide, etc., very little attention has been given to its possibilities, but we are overlooking something that can radically change and improve the industry.

An appropriation to cover such an investigation would, I am sure, be an excellent investment for your company. I am thoroughly interested and keen on the subject and would be prepared to give from 6 to 12 months of my time to it.

If the idea interests you seriously, I would be very glad to work out a more detailed scheme for your consideration and to show you that my metallurgical and electric furnace knowledge and experience fit me to tackle the problem.

If you wish, I could include in this work recommendations for improving the present condition with regard to the sample lots on which your sales and selling price of your product is based. I am quite sure I can save you money on this.

Yours very truly,

(SGD) FRANK HODSON

P.S. The 2500 KW 60-ton holding capacity furnace could, if not working on molten copper from the converters, be used for smelting ore concentrates or copper matte, depending on the analysis of the charge; it is capable of handling 80/100 tons of such smelting from cold material per day. An ore containing 5/6% copper, 12% SeO_2 , 5% CaO MgO - remainder being iron and sulphur, takes 650 KW hours per ton of charge. Electrode consumption is very small. The slag loss usually ran 0.3 to 0.4 percent copper. All fumes can be collected and the furnace operates in a neutral atmosphere.

MINING MAGAZINE
June, 1917

ELECTRIC SMELTING OF COPPER ORES

By W. DEWAR, M. I. C. E. M. M.

The author discusses the possibility of applying electric current to the smelting of copper ores, and gives the results of tests on a silicious ore containing copper as silicate, oxide, and carbonate.

INTRODUCTORY.—Smelting is the art of rendering ores fluid by the admixture of proper fluxes and the application of heat, so as to concentrate the desired metal or metallurgical product in the minimum quantity of valuable material, and eliminate extraneous matter to the greatest possible extent, the new combinations separating in the furnace according to their specific gravities and being drawn off separately. To generate the heat, carbonaceous fuel either solid, liquid, or gaseous, and electric current can be used. With the continual improvement in the construction and design of electro-metallurgical plant, together with the importance and the availability of electric power, especially hydro-electric, the smelting of non-ferrous metals by electric heat is a subject well worthy of attention by those who are interested in metallurgical progress. The van of progress has so far lain with the Continent and America, due no doubt to the greater availability of water power.

Ore deposits may be situated in mountainous districts, or may be inaccessible by railway, the price of coke for smelting may be too high, or the coke unobtainable, or the transport of the ore to a suitable reduction works may be too costly. On the other hand, sufficient water power may be available, from which electric power can be developed at a reasonable figure. It is much cheaper to carry wires over hilly districts than to transport carbonaceous fuel over the same ground. So that, *prima facie*, the field for electric smelting is where deposits are situated in regions in which coke is costly and hydro-electric power comparatively cheap.

The thermal efficiency of the coke-fired blast-furnace is about 50%, and that of the electric furnace 70%; the cost of coke and electric energy are about on a par when the ratio of the price of coke per ton to that of power per kilowatt-year is in the neighbourhood of 1 : 1.8, neglecting the value of electrode consumption per ton of ore treated. In ordinary smelting, two-thirds of the fuel is used for heating, and the remainder for reduction. The electric furnace requires only sufficient fuel to reduce the metal, and is practically only a heat producer. Its value as a substitute for the combustion fur-

nace, where conditions are such as to warrant its use, would depend on the relative cost of carbonaceous fuel and electric power.

The use of electro-thermic means for treating copper ores has been proposed in recent years, and in a number of cases experimentally tested. Speaking generally, outside of smelting iron ores, the manufacture of aluminium, and the production of steels and of ferro-alloys, electric smelting is still in the experimental stage. There are, however, some other successful applications, of which little has appeared in the press, such as the production of zinc fume, which at present commands a record price, the reduction of tin ores and tin-bearing materials, and cyanide precipitates, and in a small way the production of black copper.

Native copper, oxidized, and, to a certain extent, sulphide ores can be smelted just as efficiently as in the reverberatory or blast-furnace, as the reactions desired can be obtained equally as well in either of these furnaces. Success depends not so much on whether certain chemical or electrical results can be obtained, as to whether they can be secured on commercial lines, and that accompanying and undesirable metals can be held back in the resulting product. The electric furnace does not lend itself to an oxidizing smelt, and is particularly unsuited for the treatment of those sulphide ores which can be smelted with a consumption of a small proportion of fuel. The only part likely to affect the reactions chemically is the lining of the furnace so that the action in the furnace would be confined principally to the combining of the silica with such bases as are already oxidized, and, where necessary, those added as a flux to form the desired slag. Higher refractory slags can be produced in the electric furnace than would be possible in blast-furnace work, but in order to get metal or matte free from impurities certain slags must be run, and it is owing to the lack in this particular that unsatisfactory results have been obtained in the past with experimental trials. The worthy pioneers in this class of work have been for the most part engaged in the manufacture of ferro-alloys. The production of such alloys and the furnaces used are in no way analogous to the process and plant for the treat-

ment of ores. The slag produced in ferro-alloy manufacture is to all intents and purposes negligible, practically the whole of the solid product being of a valuable character.

Native copper or oxidized ores can be smelted direct to black copper, and lean sulphides to a high-grade matte, in one operation. Those containing sulphur in excess for that required for the matte might be partly and cheaply roasted to remove the surplus, or mixed with oxidized ores to obtain the same result. Where oxidized, semi-oxidized, and sulphide ores are available, matte production would probably be the best line of procedure to adopt. Modifications and variations in order to achieve the greatest efficiency and profit for any particular ore or combinations of ores are so entirely dependent on local conditions that the subject can only be briefly outlined in a very cursory manner. What might be good metallurgy in one place might be unprofitable or unwise in another.

One important point to be considered is that charcoal is essential for reduction of the metal from oxide, so that it is necessary for the operation to be in close proximity to a well timbered region where charcoal can be produced at a reasonable cost. Coke is not satisfactory as a reducer, being too dense. It makes undesirable electrical conditions, and causes too high electrode losses in the furnaces; it creates crusts between the electrodes, sows the furnace, and with slags high in lime has the tendency to form carbide.

In the design of an electric furnace, the aim is to obtain the highest efficiency compatible with the operating conditions. The amount of heat necessary to achieve the results desired in any metallurgical process can be determined, and in order to obtain and maintain this temperature a definite rate of heating is essential, this rate depending on the speed at which the heat is conducted away through the furnace walls, the dimensions of the furnace, and the cooling effect of the charge that is periodically added. The relative loss of heat through the walls is less for a large furnace than for a small one, so that each unit should be as large as possible, consistent with the nature of the material treated in the furnace. Where possible it is preferable to employ a continuous process, or, if intermittent, to have small delays between charging. The first consideration is to construct the furnace so as to obtain the greatest percentage of heat value from the electric energy supplied. The dimensions of the electrodes, the section of the furnace best adapted for the operations, and the most suitable tension and intensity are points to be worked out.

For a direct current the power supplied to a furnace in watts is the product of the amperes multiplied by the volts, whereas for an alternating current it is the result of the amperes multiplied by the volts multiplied by the power factor, which varies with the furnace construction. Alternating current is used almost entirely in furnace work. The power may be kept constant by low voltage and high amperage or vice versa. Large amperage means large conductors, thereby increasing the cost of cables, transformers, electrodes, etc. High voltage, though economical, is generally not feasible, as most furnaces have a low electrical resistance. There are two ways of regulating current; firstly, raising or lowering the electrodes, and second, these are kept stationary and the voltage varied by means of variable voltage transformers.

Furnace linings may be classed as acid, basic, and neutral. First, with regard to acid linings available. Good silica bricks should contain about 95% SiO_2 . Dinas bricks are somewhat friable and brittle, and do not stand abrasion. These differ from the ordinary silica bricks in the binder employed. Ganister is generally used as a loose material which is tamped into place as a furnace bottom. Fire-clay may be either acid or neutral owing to the different proportions of the silica and aluminous contents; the combination of these should equal about 95% of the clay. Given the same chemical composition, a brick from a fine-grained clay is not as refractory as one made from material in which there is a large proportion of coarse particles.

There are a number of basic materials suitable as lining. Magnesite bricks are manufactured by mixing 90% dead-burnt magnesite with 10% incompletely calcined, the latter being added to effect the setting of the brick. The objections to the MgO bricks are that they expand on heating and flake off when subjected to sudden variations of temperature, and are also liable to crumble when subjected to a heavy load when hot. Bauxite can be used for basic or neutral linings. For basic, the SiO_2 contents should be not much over 10%. It must be thoroughly calcined, otherwise it will shrink at furnace temperatures. It requires Fe_2O_3 to make a firm brick. Lime is not used as a lining, owing to the rapidity with which it slakes when exposed to the air. Of neutral linings may be mentioned carbon, which is one of the most highly refractory materials known. It can be used in the form of lamp black or retort carbon, mixed with tar and rammed in position. Chromite is extremely in-

fusible, but is not suitable for furnace bottoms as it will not stand mechanical erosion. It is, however, suitable for furnace walls above the slag line. Alundum can be used as a lining; and is made by fusing bauxite in the electric furnace. Carborundum is also employed. It is made by mixing coal, saw-dust, and silica and heating in an electric furnace at a high temperature. This material is crushed and mixed with a suitable binder.

THE ELECTRIC WORKS.—The trials on an oxidized copper ore which are recorded hereafter were undertaken at the works of Messrs. Keller, Leleux & Co., at Livet, Isere, France. These are situated about 20 miles from Grenoble, on the main line of the Paris, Lyons, and Marseilles Railway, and are connected to Livet by an electric and steam tramway. Power is obtained from the Romanche river, the water being brought at a head of 60 metres through a tunnel 1,700 metres long. At high water 20,500 h.p. is available, but from November to March this drops to half, when some of the furnaces are shut down and others operated with reduced power.

The installation comprised:

- 3 units 3,000 h.p., 3-phase, 4,000 volts.
- 1 unit 4,000 h.p., 3-phase, 5,000 volts.
- 5 units 1,300 h.p., single phase.
- 4 units 200 h.p., continuous current for general purposes.

The furnaces directly connected to the single-phase generators receive current at 50 volts, 50 cycles, at the furnace. Those on the 3-phase circuit have a single-phase transformer to each furnace and the volts vary from 50 to 70 at the furnace. The 3-phase machines are connected in parallel, and generate current of 25 cycles. The works produce calcium carbide, ferro-silicon, ferro-chrome, ferro-tungsten, and ferro-vanadium. The power cost is £2'18 to £2'75 per kilowatt-year, or 0'059d. to 0'073d. per kilowatt-hour.

THE ELECTRIC FURNACE.—The furnace employed in the copper-ore experiment was of the resistance type such as is used in their regular work of alloy manufacture, and of 600 to 700 kilowatts, with slight alteration of the tap-holes. These tap-holes were two in number, one for the slag, and the other for drawing off the metal. Briefly described, the furnace is a rectangular iron casing lined with refractory brick; at the bottom and along the centre for its greater length a carbon sole exists, made by ramming, while still hot, a mixture of anthracite and tar. This sole is in electrical communication with the exterior of the furnace by means of an iron rod. This connection is

essential to avoid the breaking of the current while casting metal, which would interfere with the working of the furnace. The operation of the furnace is therefore continuous and it is not interrupted in any way by charging or casting. The statistical figures obtained in these experiments are given in Tables I, II, III, the details of these being in reference to the discussion following.

The electrodes, two in number, to which the electric current is distributed, are suspended in the furnace, and can be lowered for any required extent of their length. The regulation of the electric input is effected chiefly by the vertical adjustment of these. This is easily accomplished and requires little attention, the man on the feed floor being responsible for this duty. The current passes through the charge between the suspended electrodes and the carbon sole. The energy of the current is absorbed by the resistance of the charge, which is thereby heated to the temperature required for the reduction. The use of the resistance of the charge as a means of heating makes it possible to control the temperature of the work, within certain limits, by controlling the current. The charging of the furnace was done by hand. The charge was shot on to the feed floor, which was on a level with the top of the furnace. The furnace was kept full by shovelling the charge in, as and when required. It worked with open top, and the gases were allowed to escape and burn. The lining was badly eaten away during the tests, due more to physical than chemical action between the slag and the lining, though the extraordinary quick drying of the furnace for lot 2 was also to a great extent responsible for the deterioration of the furnace. Pieces of lining came away with slag at times, and the tap-holes were badly eaten away by the end of the trials. The wear and tear of a furnace is to a great extent a question of design, and the mechanical defects could be overcome by a furnace more suitable for the purpose; increase in the depth and provision of water jackets at the sides of the furnace would be beneficial. The iron casing showed the following variation during the 2nd campaign: temperature at commencement 19°C; 1st day 65°C; 2nd 76°C; 3rd 89°C; 4th 103°C; 5th 110°C.

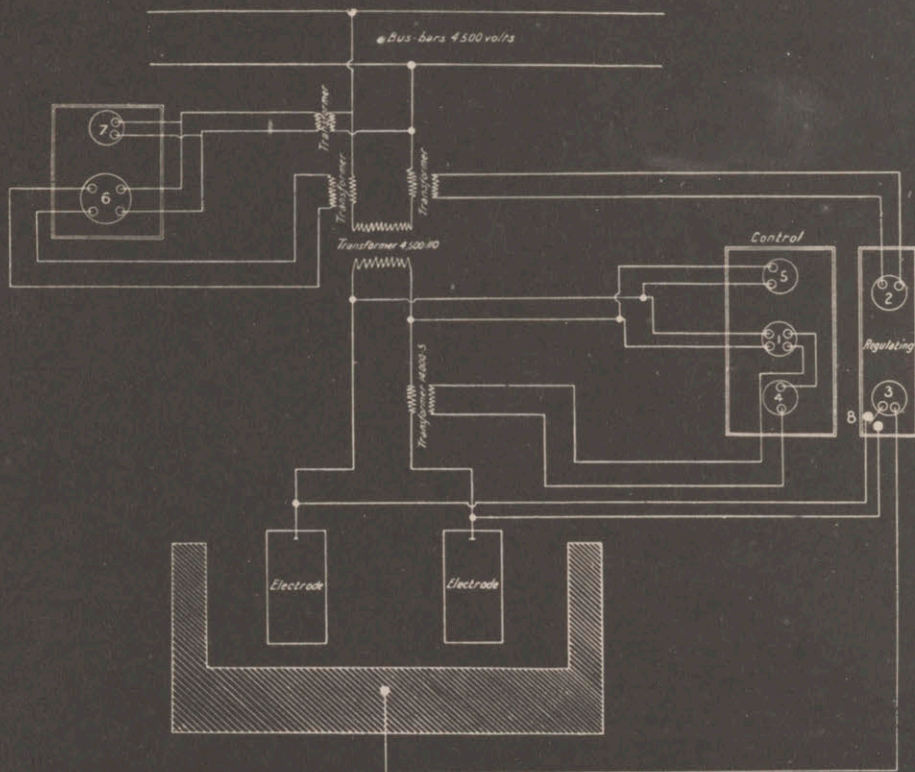
ORE, FLUX, AND FUEL.—The ore was a highly silicious oxidized ore, and was crushed to pass a 1 in. ring. The flux used was lime, with the exception of the last mixture in the second trial of 5,000 kilos, which was run with limestone. The lime used in the trials was somewhat old and had a large proportion of

finer. Extremely fine particles are detrimental, as they tend to block passages, cool the charge, and increase the power required. The fuel used was charcoal in good physical condition, the pieces being about 2 in. in size. In the mixture with limestone, coke was substituted for charcoal. The ore, flux, and fuel after being weighed were mechanically mixed before going to the feed floor. The analysis of these materials will be found in the statement in Table IV.

THE TESTS.—These were two in number. The first was started on a hot furnace, which had been heated up for a couple of days on old slag, but the run was cut short owing to the inability to tap metal, the tap-hole being too high to accomplish this. The run lasted for two days. For the second campaign, the furnace was rebuilt and the metal tap-hole lowered. It was then dried by a wood fire, and the furnace started by adding a small quantity of copper

TABLE IV.—GENERAL ANALYSES.

1. Ore.	Lot 1.	Lot 2.
	%	%
CuO	10.94	12.10
	(= 8.75Cu)	(= 9.66Cu)
SiO ₂	63.58	62.50
Al ₂ O ₃	2.42	3.51
CaO	1.81	0.60
Fe ₂ O ₃	13.28	12.76
MnO	0.51	0.73
MgO	1.05	1.80
S	0.28	0.31
Loss on ignition	5.70	5.40
		%
2. Charcoal.	Fixed carbon	88.35
	Volatile matter	5.55
	Water	3.92
	Ash	2.18
		%
3. Lime.	CaO	91.41
	MgO	0.02
	SiO ₂	0.60
	Fe ₂ O ₃ and Al ₂ O ₃	0.24
	Loss on ignition	8.17



DIAGRAMMATIC SKETCH OF FURNACE AND ELECTRIC CONNECTIONS.

- 1 Wattmeter B.T., 1,000 kw.; on the secondary.
- 2 Ammeter H.T., 0-500 A; on the primary.
- 3 Voltmeter B.T., 0-70; tension between one electrode and sole.
- 4 Ammeter, 0-14,000; on the secondary.
- 5 Voltmeter, 0-150; total tension between electrodes.
- 6 Wattmeter H.T., 1,000 kw.; office.
- 7 Voltmeter, 4,500 volts; office.
- 8 Two-way Switch.

TABLE I.—SCHEDULE OF WEIGHTS.

Ore Mixture	ORE CHARGE						
	WEIGHT		TOTALS		PER CENTAGE		COMPOSITION
	Mixture Kilos	Ore Kilos	Lime Kilos	Charcoal Kilos	Ore %	Lime %	Charcoal %
A	9,165	6,000	3,060	105	65.5	33.5	1.00
B	13,802	9,105	4,540	157	66.5	32.5	1.00
C	4,597	3,075	1,470	52	67.2	31.8	1.00
Totals...	27,564	18,180	9,070	314			
E	15,197	9,700	5,327	170	63.80	35.10	1.1
F	4,197	2,746	1,403	48	65.50	33.40	1.1
G	20,458	13,400	6,847	211	65.55	33.43	1.03
H	12,220	8,170	3,920	130	66.84	32.10	1.06
I	7,830	5,000	2,751	79	63.90	35.10	1.00
J*	10,580	5,000	5,500	80	47.3	52.00	0.70
Totals...	70,482	44,016	25,748	718			

* For lime and charcoal read limestone and coke.

filings; the current passed through, making circuits by contacts or arcs. Immediately the metal had reached the fluid state, the necessary material to form a slag on its surface was added. During this period of formation the current was subject to violent fluctuations. When the slag was formed, the electrodes were placed in contact with it, and the energy of the current largely absorbed by its resistance. Conditions then becoming normal, the charging of the ore mixture began. The furnace was under constant observation throughout the whole campaign, and all materials fed into it were carefully weighed and sampled, as also the slag and metal produced. After tapping, occurrences of blowing out of the material happened, caused by the sudden fall of cool material around the electrodes into the hot zone of the furnace. This blowing out was much more marked when dealing with the limestone mixture, owing to the escape of more gas than under the lime charge.

In the first run there were three different mixtures, and in the second six. Full details as to these and furnace capacities will be found in the schedule of weights, Table I. The second trial lasted five days.

SLAGS.—The assays of the slags as given for the different mixtures represent "cut-outs," and were taken as representative of those produced by the different ore charges. The higher alumina in the first series was due to the greater fluxing of the furnace lining. The slags produced were very clean and flowed freely, and in the majority of cases readily solidified without passing through an intermediate state of plasticity. The average running tem-

perature was about 1,450°C, this being determined by means of a Wanner pyrometer.

The number of slag tappings in the first series was 25, and in the second 72. The amount of slag produced per ton of ore was respectively 1.22 and 1.24 tons, and averaged 0.25% and 0.3% copper. The metallic contents of the slags compare favourably with those of slags produced in matte smelting, and show a considerable improvement over those produced by direct reduction of this class of ore in blast-furnaces, which might be anything between 1 and 1.5%. Though here it might be mentioned, en passant, that to keep slags low in copper in direct reduction to metal in blast-furnaces, it is advisable, in fact imperative, to have some sulphur present in the charge. The SiO_2 varied from 49%–57%, and represented the bisilicate type, having the formula $\text{R}_2\text{O}_3, 3\text{SiO}_2, \text{RO}, \text{SiO}_2$, or $\text{R}_2\text{O}_3, \text{RO}, \text{Si}_4\text{O}_8$. Prima facie the slags would appear economically and metallurgically good, as they used the minimum of flux, and were low in copper contents, and while they were all right from that point of view, the grade of metal was all wrong; though it will be here mentioned that previous experimental trials on the ore had indicated the possibility of producing high-grade metal with the type of slags as shown in the analysis schedule. How this was accomplished will be indicated under the heading "Metal."

The slag is the primary concern of the smelter, and given the proper one the metal will take care of itself. With a suitable slag there is no difficulty in producing good black copper in the electric furnace, and it is being done. The essential is a fluid slag of low

TABLE II.—ASSAYS AND POWER

Ore Mixture	SLAGS									POWER CONSUMPTION	
	ANALYSES									TEMPERATURE	Per 1,000 kilos ore-charge K.W.H.
	SiO ₂ %	Al ₂ O ₃ %	CaO %	FeO %	MnO %	MgO %	CuO %	S %	Degrees Centigrade		
A	54.96	7.35	32.80	3.56	—	0.42	0.20	0.05	1,400—1,460	510	755
B	54.20	8.70	34.12	2.61	—	0.27	0.42	0.04	1,400—1,580	755	1,145
C	53.32	7.21	34.30	4.62	—	0.87	0.22	0.07	1,460—1,540	1,100	1,640
Average Slag	55.01	7.9	33.26	3.57	—	0.38	0.31	0.057			
E	52.16	4.28	36.81	5.93	0.37	0.81	0.46	0.091	1,400—1,550	696	1,088
F	53.10	3.83	36.41	5.68	0.40	1.17	0.32	0.07	1,460—1,540	597	910
G	53.40	4.06	36.48	5.01	0.38	0.27	0.59	0.08	1,480—1,540	625	955
H	56.70	4.88	34.76	3.08	0.40	0.44	0.51	0.044	1,460—1,560	664	992
I	55.18	4.71	35.86	3.64	0.40	0.35	0.26	0.054	1,450—1,540	710	1,113
J*	49.44	3.91	40.58	5.90	0.40	1.24	0.53	0.029	1,400—1,540	600	1,265
Average Slag	52.20	4.97	36.64	5.00	—	1.17	0.38	0.11			

* For lime and charcoal read limestone and coke.

TABLE III.—SUMMARY OF RESULTS. LONG TONS.

Trial Number	MATERIAL SMELTED IN FURNACE.				MATERIALS USED PER TON OF ORE.		POWER CONSUMPTION		ELECTRODE CONSUMPTION		COPPER		
	TOTALS DURING TRIAL		AVERAGE PER 24 HOURS		Lime. Tons.	Charcoal. Tons.	Per ton ore. K.W.H.	Per Ton of Copper K.W.H.	Per ton of Ore. Kilos.	Per ton of Copper Kilos.	Recover'd as Metal. %	In Slag. %	Un-accounted for %
	Ore Mixture. Tons.	Ore. Tons.	Ore Mixture. Tons.	Ore. Tons.									
1	27.12	17.9	14.95	9.84	0.499	0.017	1.122	14.025	12.40	155.00	91.48	5.0	3.52
2	69.38	43.4	15.15	9.52	0.519	0.016	1.028	11.719	5.53	63.04	91.10	5.6	3.74
	96.50	61.3											

* Lot J left out.

melting point. The question to consider is not whether the proper ratio between base and acid exists, but whether the bases are of such a nature that the slag will melt at a suitable temperature. This belongs to a study of slags. Iron is readily reduced from its oxides, and it requires much care even at the temperature of the ordinary blast-furnace for smelting copper ores to prevent its reduction. A resistance furnace when operating adjusts itself to the charge, so that with a refractory slag it gets hotter; this increase in temperature results in accelerating the reaction between the somewhat costly electrodes and the iron oxide of the gangue and reduces iron which enters the copper. The power consumption and the general operation of the furnace are all 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 lower the temperature the more rapid the rate of smelting, and the higher the purity of the metal, and consequently the slag best adapted

to furnace work is one that gives the greatest furnace output of ore, marketable brand of copper, or suitable matte, compatible with economic conditions.

Bi and tri-silicates form at a temperature 80° lower than mono-silicates, but these acid silicates are not melted at the formation temperature, whereas the mono and sesqui-silicates are quite fluid. Complete fusion of ferrocilic silicates is difficult to obtain, when CaO greatly exceeds the FeO, and in this case it is advisable to make slags carrying less than 40% SiO₂. In high SiO₂ slags, the CaO should not exceed the FeO. The specific gravity of the slag has also to be considered. CaO lowers it, thus permitting easier settling of the metal. A bi-silicate slag which has CaO to all intents and purposes the only base, the other bases being so small as to be negligible, is fatal in producing a high-grade metal. It is the same with matte production in the electric furnace. Any attempt to make difficultly fusible slags is attended by a decrease in general economy

and matte value.

The following data (Table V) will give some indication of typical good slags produced in the electric furnace. The first is that obtained on a commercial scale in treating a highly silicious carbonate ore using ironstone and a small percentage of limestone as a flux. The metal produced averaged 98% copper. The others represent experimental work on native copper material, and gave metal containing 96.24%, 97.96%, and 98.59% copper respectively.

TABLE V.—TYPICAL GOOD SLAGS.

	1	2	3	4
	%	%	%	%
SiO ₂	46.2	53.14	51.10	44.88
FeO.....	35.5	11.58	25.50	29.80
CaO.	13.4	12.24	8.78	6.92
MgO.....	—	3.12	2.03	2.32
Al ₂ O ₃ ...	4.2	20.33	10.45	15.34
Cu.....	0.35	0.31	0.23	0.15

METAL.—In the first trial as previously mentioned no metal was tapped, and the fused mass of copper and brick lining removed from the furnace could not be weighed or properly sampled. The metal recovered has been based on the average slag assay, and due allowance has been made for other losses as indicated by the second trial. In the second campaign only 3 tappings were obtained, the first 9 hours after starting, the second 20 hours, and the third 31 hours, giving altogether 400 lb. of metal product. Copper with a high percentage of iron chills quickly, and the formation of crusts of slag or other material in proximity of the tap-hole may in some measure account for the inability to tap more metal. The formation of such crusts in treating an ore charge in an electric furnace is of common occurrence, and can be overcome by adding to the furnace at intervals just previous to tapping and in line with the tap-hole easily fusible material such as CaF₂ or old slag.

The analyses of the three metal tappings were as follow :

	1	2	3
	%	%	%
Copper ...	75.96	61.25	57.74
Iron	23.60	34.75	39.90

Toward the end of the smelting this metal was thrown back into the furnace and remelted with that remaining in the furnace.

After all the ore mixture had been smelted and the slag tapped, the contained metal was run from the furnace by boring a tap-hole lower down and the metal run to specially constructed moulds, in order to segregate the metal

into a high and low grade product. Segregation does take place under certain limits, but as to what these are exactly has not been so far determined.

The following segregated metal was obtained: 2,275 kilos assaying 91% copper and 8.03% iron, with traces of silicon; and 3,535 kilos assaying 51.01% copper and 45.72% iron, with traces of silicon; total 5,810 kilos assaying 66.68% copper and 30.96% iron; the iron equals 45.71% of total present in the ore. In lot 1, the product averaged 57.60% copper and 40.20% iron, and contained 63.4% of the iron in the ore.

The 2,275 kilos of metal represent 39.16% of the total produced, and 48.7% of the total copper in the ore. The 3,535 kilos represent 60.84% of the total produced, and 42.4% of the copper in the ore.

In Messrs. Keller's experimental trials the products of segregation were represented by metal of the following grades: No. 1 Product, copper 93.84%, iron 5.34%, Si 0.699%, and represented 88% of the total copper; No. 2 Product, copper 16.51%, iron 79.0%, Si 3.58%, and represented 6% of the total copper of the ore. In the earlier runs the following represented the ratio type of ore mixture: 1,000 kilos ore, CaO 400 kilos, bauxite 80 kilos, fluor-spar 30 kilos, charcoal 17.5 kilos.

ELECTRODES.—The actual consumption recorded in Table III. was obtained by measurement. To allow for losses by breakages and portions of electrodes that could not be utilized in actual practice, 20% should be added to compute works practice. The electrodes employed were suitably coated to prevent undue oxidation, and were water-cooled, thus preventing the burning of the electrode and heat getting to the transformer. Twenty litres of water passed through each electrode per hour and a rise in temperature of 8°C. was shown. The water became electrically charged. A new set of electrodes was employed for the second trial.

POWER CONSUMPTION.—The current employed for preliminary heating has not been reckoned in the figures recorded. The current used was one of low voltage and fairly high amperage. In No. 1 test the volts varied from 96 to 106 and the amperes from 2,400 to 5,500; in No. 2, the volts varied from 96 to 104, and the amperes from 3,500 to 6,000. The average consumption of energy during the stable periods of the trials (mixtures F and G) was 955 kilowatt-hours per ton of ore, and the consumption per ton of copper was in trial 1, 1.6 kilowatt-year, and trial 2, 1.34 kilowatt-year.