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J. W. POWELL, DIRECTOR

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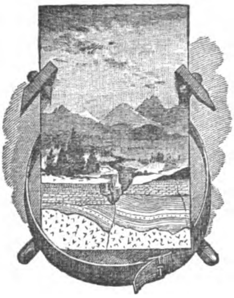

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UNITED STATES GEOLOGICAL SURVEY,
DIVISION OF MINING STATISTICS,
Washington, D. C., July 25, 1885.

SIR: I have the honor to transmit herewith a paper by Mr. Henry M. Howe, on copper smelting, for publication as a bulletin of the United States Geological Survey. This paper is supplementary to the contributions on the metallurgy of copper which have appeared in the reports of this division, and is of great interest as embodying a most complete review of present practice.

Very respectfully, your obedient servant,

ALBERT WILLIAMS, JR.,
Geologist in charge.

Hon. J. W. POWELL, *Director.*

COPPER SMELTING.

By Henry M. Howe.

INTRODUCTION.

In the following paper a somewhat detailed account is given of the methods and the apparatus employed in smelting copper ores and in bringing forward the resulting products to the condition of crude metallic copper by the reverberatory or Welsh and by the cupola or shaft furnace method. Since the subject of roasting[1] copper ores and mattes is ably treated by Dr. E. D. Peters, jr., in the "Mineral Resources of the United States, 1882," published by the United States Geological Survey, it is comparatively briefly passed over in the present paper. The reader is referred for details to that volume, while the valuable information there given is here supplemented by some remarks on the expulsion of arsenic and antimony in roasting, since Dr. Peters does not dwell on this important point, and by several tables giving the more important economic and technical results of the roasting of ores and mattes as practiced in various places. These tables, as well as two showing the corresponding results of smelting in reverberatory and in cupola furnaces, have been compiled from a great number of sources, largely from private communications from the superintendents of copper smelting works,[2] expressly for this work. The writer is not aware that so extensive a compilation of these features of copper metallurgy as is given in the tables has heretofore been published.

The chief problem which is presented in copper smelting is to separate the copper contained in the ore—

1. From the accompanying silicious or earthy gangue.

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[1] In copper metallurgy in Great Britain, and also in general in this country, true roastings (*i. e.*, oxidizing, chloridizing, and similar operations carried on at elevated temperatures without fusion or with at most incipient fusion) are spoken of as "calcinations," the word "roasting" being applied to a particular strongly oxidizing smelting of the Welsh reverberatory process. In this paper the writer uses these words in their strict sense, designating the so-called "calcinations" by their true name, "roastings," and the oxidizing fusion of the Welsh process as a "roasting-smelting."

[2] Where these figures have appeared improbable, the writer has called for corroboration and explanation of them, but he has not, in general, been able to verify them by personal observation.

2. From iron, lead, zinc, nickel, etc.
3. From arsenic and antimony.
4. From sulphur.

The general plan of the separation is the same in the reverberatory and the shaft furnace methods.

1. The gangue is removed by fusion, the copper, in the case of oxidized ores, being previously reduced to the metallic state. On melting, the copper is collected in a matte or in the metallic state, the siliceous and earthy components of the ore uniting in a slag which is specifically lighter than the matte or copper, thus affording a ready means of separation.

2. In the case of sulphureted ores the separation of copper from iron and most of the other heavy metals is based on the fact that, while the affinity of these metals for sulphur is little if at all stronger than that of copper, they have a much stronger affinity for oxygen, so that by a series of oxidizing roastings, each followed by a fusion, the iron and the other heavy metals are gradually scorified and removed, while the copper unites with the remaining sulphur to form a matte, whose tenor in copper of course increases at each fusion with the expulsion of the iron, sulphur, etc.

In the case of oxidized ores and native copper the greater affinity of iron and most of the other heavy metals for oxygen has to be relied on for their elimination. We may have a fusion in which the balance between the oxidizing and the reducing tendencies is such that all the iron and a little of the copper remain oxidized and pass into the slag, while most of the copper is reduced to the metallic state. That portion of the copper which passes into the slag may be recovered by precipitation on iron bars inserted into it, or by submitting the slag to a subsequent reducing smelting. Or this balance may be such that all the copper and part of the iron are reduced to the metallic state, the latter element being subsequently removed by an oxidizing fusion ("roasting"-smelting and refining). Or we may expose oxidized ores to a mixture of reducing and oxidizing gases capable of reducing copper (but not iron) to the metallic state and, by a subsequent nearly neutral fusion, scorify the still oxidized iron.

3. The elimination of arsenic and antimony is chiefly based on the volatility of their sulphides, of metallic arsenic and of arsenious acid (As_2O_3) and trioxide of antimony (Sb_2O_3).[1] In each of the roastings, the

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[1] The volatilizing point of metallic arsenic is stated by some authorities as at a dull red heat and by others as at 180° C. Arsenious acid (As_2O_3) volatilizes at about 218° C. Arsenic acid (As_2O_5) uncombined with bases is decomposed by heat alone at a full red heat into free oxygen and arsenious acid; but the metallic arseniates (compounds of arsenic acid with the oxides of the metals) do not in general appear to be decomposed by heat alone. The sulphides of arsenic, both realgar (As_2S_2) and orpiment (As_2S_3), are readily volatile. But when arsenic is combined with sulphur and a great excess of copper, iron, etc., in a matte, it is not volatilized by heat alone. The same is true

arsenic and antimony are partly oxidized. As they oxidize, part is expelled as they pass through the intermediate volatile state of arsenious acid and trioxide of antimony; while part, becoming fully oxidized, combines with the metallic oxides to form arseniates and antimoniates, which are non-volatile. In the subsequent fusion, part of the still unoxidized arsenic and also (though to a smaller extent) of the antimony is expelled as sulphide; while the portions previously fully oxidized to the non-volatile condition of arseniate and antimoniate are again reduced: part to metallic arsenic and antimony, which combine with the resulting matte, while part is expelled while they are in the intermediate volatile state of arsenious acid and trioxide of antimony. Thus in the alternate oxidizing and deoxidizing operations, the roastings and fusions, much of these noxious elements passes alternately from the state of sulphide, in which they are quite strongly held by the matte, to the non-volatile completely oxidized condition and back again, and on each of these passages a portion is expelled in the intermediate volatile state of arsenious acid and trioxide of antimony.

These elements are sometimes, when present in great quantity, eliminated in a speiss, and they are frequently concentrated in a small portion of the copper (in copper bottoms) by reducing a little of that metal to the metallic state, the rest remaining in a matte which is nearly free from arsenic and antimony.

In roasting pyritous ores a considerable amount of arsenic and even of antimony may be expelled as sulphide, in combination with the sulphur volatilized from the pyrites.

4. In the series of roastings and fusions whose aim is the expulsion of iron, of arsenic and antimony, and of the other heavy metals, much of the sulphur is inevitably expelled. By prolonging this series after the expulsion of the iron the last portions of the sulphur are finally expelled as sulphurous and sulphuric acids (anhydrides).

It is desirable to retain a considerable portion of the sulphur until the iron, arsenic, and antimony have been expelled; on account of the power which sulphur possesses of removing copper from its oxides without deoxidizing iron, its presence enables us to prolong and push further the oxidizing operations by which the iron, arsenic, and antimony are expelled, without incurring, in the subsequent smeltings, a heavy scorification of our copper. It is moreover much easier to expel these elements (iron, arsenic, and antimony) while they are combined with

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of antimony. Metallic antimony volatilizes, though slowly, at a white heat. Its trioxide (Sb_2O_3) melts below a red heat and volatilizes when heated to a higher temperature. Antimonic acid (Sb_2O_5), when uncombined with bases, is decomposed at a red heat into free oxygen and tetroxide of antimony (Sb_2O_4), which appears to be unalterable by heat alone; but it does not in general appear practicable to decompose the antimoniates by heat alone. The trisulphide of antimony volatilizes at a white heat, but some other combination of antimony and sulphur appears to volatilize at a lower temperature. The sulphides of antimony are much less volatile than those of arsenic.

the sulphur in a matte than when, by the expulsion of the sulphur, they become alloyed with metallic copper.

Where sulphureted ores are to be treated with a large proportion of oxidized ores, the roastings may advantageously be abridged or even dispensed with, the oxidized ores furnishing the oxygen required for the expulsion of the foreign elements.

Where oxidized ores alone are to be treated the problem is much simplified; the iron being already oxidized and arsenic and antimony having been for the most part removed by processes of nature similar to those which have removed the sulphur and oxidized the iron, little remains to be done in general beyond submitting the ore to a fusion, too mildly deoxidizing to reduce any great amount of iron to the metallic state yet sufficing to reduce the copper, the iron passing into the slag.

Thus we may say that the reverberatory and the cupola methods each consist of a series of oxidations (roastings), in which iron, sulphur, arsenic, antimony, and to a smaller extent the copper itself are oxidized, alternated with smelting operations in which the copper thus oxidized is again deoxidized, be it by carbon or carbonic oxide, or at the expense of some remaining portion of hitherto unoxidized sulphur or iron, and in which the oxidized iron is removed in a slag. The oxidized arsenic, antimony, and sulphur escape in the gaseous state.

The roastings may take place in heaps, stalls, kilns, reverberatories, or shaft furnaces, according to the mechanical condition and chemical composition of the ore treated, the price of the fuel available, and its quality, the practicability and expediency of recovering the sulphur expelled as sulphuric acid, etc. The fusions may take place either in cupola (*i. e.*, shaft) furnaces or in reverberatories, the choice depending, primarily, on the relative prices of coke, anthracite, and suitable charcoal on the one hand, and of bituminous coal, wood, peat, sawdust, etc., on the other hand; and, secondarily, on the character of the gangue and the character and cost of the available fluxes, on the constancy of the quality of ore supplied, and to a certain extent on the class of workmen obtainable, the size of the establishment, etc. In some establishments all of the fusions are performed in cupola furnaces; in others they are all performed in reverberatories. Although under all ordinary circumstances it is more economical to use a mixed process (performing the fusion of the ore in cupolas, and the last fusion of matte, whose product is mainly or wholly metallic copper, in reverberatories, the choice of furnace for the intermediate operations being dictated by considerations which will be given at length in a later paragraph), it is more convenient to consider and describe as two distinct and separate methods the smelting and bringing forward to metallic copper, employing on the one hand cupolas alone and on the other reverberatories alone for all the fusions. The relative advantages of the two methods and the cases in which each is specially applicable can be more intelligently discussed after describing the processes themselves.

REVERBERATORY METHOD FOR SULPHURETED ORES.

The procedure in this process varies endlessly, according to local conditions and traditions, not only as regards the details of the several operations, but as regards the number of operations employed. In general the number of operations depends chiefly upon—

1. The ratio of the copper to the sulphur, and especially to the sulphide of iron, in the ore. Where this ratio is high the matte obtained in the first smelting will be comparatively rich in copper, and hence may be brought forward to the metallic state in a small number of operations.

2. The importance attached to the expulsion of arsenic and antimony. These elements are expelled more completely by a large number of comparatively incomplete roastings (alternated, of course, with smeltings) than by a smaller number of extremely thorough roastings; and in order to increase the number of roastings, with a view to increase the expulsion of arsenic and antimony, it is sometimes desirable to add raw pyrites in a comparatively advanced stage of the process to throw the grade of the product back.

3. The cost of labor and fuel as compared with the value of the copper per pound at the smelting works. If each of the roastings is extremely thorough, higher grade products will be obtained in each smelting, and consequently the copper will reach the metallic state in a smaller number of operations and at a considerably smaller cost than if the roastings had each been less thorough, but the loss of copper in the slags will also be greater. Where labor and fuel are dear and where the copper is of comparatively low value (either owing to long distance from market or to depression of the price of the metal), it is naturally advisable to push the early roastings farther, so as to lessen the number of operations and thus the total cost of extraction, even if the loss of copper be thereby increased, than would be expedient if fuel and labor were cheap and the copper worth more per pound at the smelting works.

4. Finally, if oxidized ores of copper are to be had to mix with our sulphurets, their use may enable us to abridge the roasting operations or even to dispense with them altogether.

It would be not only tedious but useless to attempt to describe all the methods of procedure. To fix our ideas, let us select for description two important methods followed at Swansea and sometimes designated as the "ordinary" and the "extra" or "selecting" process respectively. The first consists of six operations, the second of seven. As far as the first three and the last two operations are concerned the two processes are nearly identical, the chief difference being that the fourth operation of the ordinary process is in the extra process replaced by two operations.

The ordinary process is as follows:

1. Roasting the ores of class I (ordinary sulphurets).[1]
2. Smelting the roasted ore of class I with the raw ore of class II (lean or impure oxidized ores), obtaining coarse metal (matte), which is roasted in (3), and a clean slag, which is thrown away.
3. Roasting the coarse metal.
4. Smelting the roasted coarse metal with ores of class III (chiefly rich pure oxidized ores), obtaining white metal.
5. Roasting-smelting the white metal for blister copper.
6. Refining the blister copper.

The extra or selecting process is as follows:

1. The same as in the ordinary process.
2. The same as in the ordinary process.
3. The same as in the ordinary process.

4*a*. Smelting the roasted coarse metal for blue metal.
4*b*. Roasting-smelting the raw blue metal for regule and bottoms.
5. Roasting-smelting the regule and bottoms for blister copper.
6. Refining the blister copper.

1. ORE ROASTING.

ROASTING IN HEAPS OR PILES.

Where the ore contains enough sulphur or bituminous matter to keep up an active combustion after the wood employed for firing it has burnt out, this method is much the cheapest, requiring the minimum of handling and of plant. It, however, ties the ore up sometimes for many months, and certain precautions have to be taken to prevent loss of copper, as soluble sulphate of copper is formed which is liable to be washed out by the rain.

Nearly all the sulphur contained in the ore is inevitably lost in pile burning, though a very small fraction of it may be condensed as sulphur on the tops of the piles, as at Agordo, the Lower Harz, etc.[2]

Some examples of the economic features of heap or pile burning are given in the following table:

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[1] See classification of ores and slags, pp. 70-72.

[2] Kerl, Grundriss der Metallhüttenkunde, I, p. 139.

TABLE I.—*Examples of heap roasting.*

	Point Shirley, Mass.	Agordo. Kernel burning. Pyrite.	Mansfeld. Bitu minous schists.	Mansfeld. Bituminous schists.	Manila, Pyrite, enargite, and covellite.[a]	Capelton, Canada. Pyrite.	Imaginary.	Strafford, Vt. Pyrrhotite.	Ely, Vt. Pyrrhotite.	Ore Knob, N. C. Pyrrhotite, chalcopyrite.	Ione, Cal. Cupreous pyrite.[b]
Weight of each pile in 2,000 pound tons	100 to 150	210	150	80 to 200 Fuder	25	100	250	350	300	150	150 to 200
Composition of ore:											
Non-metallic gangue ... per cent								22.1		30	*c* 20
Sulphur ... do		50.0				38		33	32.6		*c* 39
Iron ... do		Fe and Zn 42.0				33		37.31			*c* 36
Copper ... do	17 to 20	1.6	3 to 5			4		6.35	8.2	4.26	*c* 5
Pyrite ... do	10 to 15	Say 95.			50	75	60				
Sulphur in roasted ore ... do								4.6	7.4	7.04	*c* 6.5
Days' labor, per ton of ore	0.15	0.147	0.3			0.3	0.25	0.132		0.14	0.25
Wood, cords, per ton of ore	0.021	0.007	0.04			0.05	0.03	0.024		0.03	0.01
Weeks occupied in roasting	9	40	10	4	5	6	9 to 10	11	11	6	12 to 20
Roasted ore smelted in cupola yields matte containing—											
Sulphur ... per cent	32.5							22.33		22	
Iron ... do	31									47	
Copper ... do	35.5		45.5			25				23	
Authority	R., p. 273	R., p. 314	R., p. 378	K., p. 138	K., p. 138	P. N.	P.	G.	P.	E.	P. N.

[a] Arsenious acid and sulphide of arsenic copiously evolved.

[b] Roast preliminary to weathering and leaching.

[c] Estimated.

E. = T. Egleston, Transactions of the American Institute of Mining Engineers, X, p. 33.
G. = W. Glenn, private communication.
K. = Kerl, Grundriss der Metallhüttenkunde, 1881.
P. = E. D. Peters, jr., in Mineral Resources of the United States, 1882.
P. N. = Private notes.
R. = Rivot, Principes généraux du traitement des minerais métalliques, 1871, I.

ROASTING IN STALLS.

Roasting in stalls requires somewhat more labor than heap burning, and a considerable amount of plant. Except where fuel is very dear and labor comparatively cheap, these drawbacks will nearly or quite counterbalance the advantages it offers over heap burning in effecting a small economy of fuel and a somewhat more complete roasting. Beyond this, there is less mechanical loss in stall roasting than in heap burning, which becomes a matter of great moment in rich ores.

KILN ROASTING.

Kiln roasting gives a more complete control over the operation and more thorough oxidation than either pile or stall roasting, requires no fuel, permits the nearly complete recovery of the sulphur as sulphuric acid, and requires but as many hours as pile roasting requires weeks or even months. On the other hand, kiln roasting requires considerably more labor and an extensive and costly plant, so that, except where the recovery of the sulphur is aimed at for the manufacture of sulphuric acid or for the protection of the neighborhood from its fumes, kiln roasting is rarely as advantageous as pile or stall roasting. When ores lean in sulphur are treated the output of each kiln is very much greater and the first cost of the kilns per unit of daily capacity (their chief drawback) consequently very much less than in the case of ores rich in sulphur.

Only lump ore can be treated in the ordinary kilns, and it is not in general found practicable to have more than 8 per cent. of fine ore mixed with the lumps.

The following table gives some examples of the size and the output of kilns and of the expulsion of sulphur in them :

TABLE II.—*Size, output, &c., of pyrites kilns.*

Locality of ore.	Per cent. of sulphur.		Grate.		Thickness of ore on grate bars.	Pounds ore roasted per square foot of grate per twenty-four hours.	Pounds ore roasted per kiln per twenty-four hours.	Days of labor per ton of ore.	Authority.
	Before burning.	After burning.	Length.	Width.					
			′ ″	′ ″	′ ″				
	48		5 8	4 6	2 3	30	787		Lunge.
	40 to 42					35	900		
	38 to 40					40	1,012		
Davis, Mass.	49	3	(*a*)	(*a*)		32	800		Private notes.
Westphalian	41 to 42					50 to 64			Bode.
Norwegian	41 to 42					38.3			
Oker	40				1 6½	30.5			Bräuning.
Capelton, Can.	40	4.5	4 6	4 6	2 6	49	1,000	0.41	Private notes.
Freiberg	Very small.				(*b*)				

a Twenty-five square feet grate. *b* About 7 feet.

FURNACES FOR FINE ORE.

Three of these, Gerstenhöfer's,[1] Hasenclever and Helbig's,[2] and Maletra's,[3] have come into extensive use for roasting ores where the manufacture of sulphuric acid is aimed at. Each effects a thorough roasting with a considerable saving of fuel and labor as compared with roasting in reverberatories. The first two, however, are very costly in construction.

Hasenclever and Helbig's burner consists of a vertical series of zigzag shelves inclined at an angle of 38°; on these the ore lies at its natural angle of repose, 33°, and gradually slides down from each shelf to the next lower as that on the lowest shelf is withdrawn. This burner, though capable of good work, apparently cannot compete with Maletra's or Gerstenhöfer's under any ordinary conditions.

Gerstenhöfer's beautiful furnace hardly needs description. Briefly, it consists of a vertical shaft, across which run a great many (say 100) parallel horizontal fireclay bars, staggered so that the ore falls from one to another as it passes from the feeding hopper at the top to the discharge at the bottom. This seems to be an excellent apparatus for treating ores and mattes lean in sulphur, especially where the manufacture of sulphuric acid is aimed at and where the material to be treated would be pulverized at any rate. Where the material is in lumps it will not often be expedient to pulverize or granulate it for this furnace. It is used in Freiberg for roasting ores and in Swansea extensively for matte-roasting, the matte being granulated especially for the purpose. The first cost of the furnace and the cost of repairs are very heavy per unit of output, and it is stated that the ore must be reduced to a very uniform size, the largest grains not exceeding 0.04 inch in diameter, a condition hard to comply with.

Maletra's (or Schaffner's) furnace is better adapted to ores very rich in sulphur, such as cupreous pyrites, but less applicable to ores lean in sulphur than Gerstenhöfer's. It consists of a rectangular firebrick chamber some 7 feet 6 inches high, 9 feet long, and 4 feet 6 inches wide, all inside measurements. In this chamber are seven shelves, each extending the whole width of the chamber, but each 6 inches shorter than the inside length of the chamber, so that spaces are left between the ends of the shelves and the end walls of the chamber, and through these spaces the ore is transferred from shelf to shelf. The shelves are staggered so that these spaces come alternately at the front and rear ends of the chamber. The fine ore is charged on the upper shelf and after four hours moved to the next lower shelf, when a fresh charge of ore is introduced on the upper shelf. After four hours more each charge is

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[1] Sulphuric acid and alkali, Lunge, I, p. 205, and Kerl's Grundriss der Metallhüttenkunde, I, p. 147.

[2] Lunge, op. cit., p. 196.

[3] Ibid., p. 218.

moved to the next lower shelf, and so on, each lot of ore remaining four hours on each of the seven shelves, and thus having a total exposure of twenty-eight hours. Here, as in the two furnaces last described, the compact arrangement allows so little radiation of heat that the heat generated by the combustion of the sulphur and iron of the ore suffices to keep the apparatus up to the proper temperature without the employment of any extraneous fuel; the removal of the ore from shelf to shelf exposes fresh surfaces to the air.

This burner has a great advantage over Gerstenhöfer's in not requiring the ore to be either uniformly or very finely crushed. In some cases dust and pea size are roasted together, leaving it is said only 1 to 1.5 per cent. of sulphur in the ore. For fine ores, rich in sulphur, this is probably by far the best apparatus yet devised,whether for acid making or as merely preliminary to smelting — simple in construction, repairs, and operation, and highly efficient. It is already extensively used in this country and in Europe for burning cupreous and other pyrites for acid making.

As compared with Gerstenhöfer's, Hasenclever and Helbig's, and other automatic furnaces, it has the serious disadvantages that the doors must be frequently opened, allowing a great excess of air to pass into the acid chambers, with loss of acid and increased consumption of niter, and that considerable labor is required in charging and working the ore.

Spence appears to have removed both these drawbacks of the Maletra furnace by making it automatic. He attaches to it an ingenious arrangement of automatic traveling rakes[1] and an automatic charging and discharging apparatus, the rakes which stir the ore and transfer it from shelf to shelf passing through stuffing boxes and thus preventing the entrance of air. This modification of the Maletra furnace is coming into use in this country, and appears to the writer to be the most promising mechanical roaster yet proposed for cupreous pyrites. The corrosion of the iron moving parts by the acid fumes, a fatal stumbling block to many mechanical roasters, has certainly been here reduced to a minimum. Still, it can hardly be considered as fully established that the advantages which it offers more than compensate for its increased first cost and its probably heavy additional wear and tear.

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[1] The Spence furnace is described by W. H. Adams in Transactions of the American Institute of Mining Engineers, Vol. XIII, p. 345; also, in The Engineering and Mining Journal, April 11, 1885.

TABLE III.—*Showing some economic features of roasting pyrites in Hasenclever and Helbig's, Gerstenhöfer's, Maletra's, and Spence's furnaces.*

Name of furnace.	Kind of ore.	Labor, days per ton ore roasted.	Output per furnace per twenty-four hours, in 2,000-pound tons.	Per cent. sulphur in ore.		Authority.
				Initial.	Final.	
Hasenclever and Helbig.	Pyrites	......	0.5 to 0.8	....	3.7 to 6.5	Lunge, op. cit. p. 200.
	Cupreous pyrites	......	0.75 to 1.0	....	....	
Gerstenhöfer	Cupreous pyrites	......	2.0 to 5.0	....	....	Lunge, op. cit. p. 205.
Maletra or Schaffner	do	0.66	0.75	48	1. to 1.5	Lunge, op. cit. p. 215.
Do	Cupreous pyrites. Capelton, Canada.	0.62	1.00	40	4	Private communication.
Do	Pyrites. Charlemont, Mass.	0.56	0.65	49	4	Do.
Spence (mechanical)[a]	Pyrites. Louisa County, Virginia.	[b]0.375	3.6 to 4.75	46	1.25 to 2.25	W. H. Adams, private communication.

[a] 1.25 horse power are used in working each furnace.

[b] When working only two furnaces. The labor per ton should be much less if a larger number of furnaces were used.

Revolving cylinders, such as Brückner's, Oxland's, etc., have not in general been successful in roasting copper ores. Ores rich in sulphur are so plastic during the early part of the roasting that the surfaces of the individual pellets of ore are closed, or, as it were, welded up by the series of impacts produced by their fall through the cylinder. This objection does not apply with as much force in the case of ores lean in sulphur, and further experience may show these furnaces to be desirable for such ores.

The *Stetefeldt furnace*, for the same reason, has generally been considered inapplicable to copper ores; but it may yet be found applicable to ores containing little sulphur.

Reverberatory furnaces.—1. Open reverberatories employ at least as much labor per ton of ore treated for a given amount of oxidation as Maletra's shelf-burners and considerably more than heap, stall, or kiln burning; in addition, they involve a pretty heavy consumption of fuel, say 0.12 tons fuel and upward per ton of ore. They are therefore rarely desirable for ores rich in sulphur, which, if in lumps, are best roasted in piles, stalls, or kilns, and if fine in Maletra's (or perhaps Spence's) furnace. Reverberatories are, however, desirable, especially when fuel is cheap, for ores too lean in sulphur to furnish the heat needed for their own roasting and for ores containing much arsenic or antimony, since they afford special means for the expulsion of these elements.

One great advantage which reverberatory roasting furnaces offer as preparatory to other processes (viz, enabling the oxidation to be carried farther than is practicable in heaps and kilns) here in general counts for little, since an oxidation sufficiently thorough for the purposes of copper smelting can be obtained in heaps and kilns where the ore is rich enough in sulphur to be appropriately treated in them.

TABLE IV.—*Matte and ore roasting in reverberatory furnaces.*

	Swansea, ore, 1848.	Swansea, matte (coarse metal), 1848.	Swansea, ore, 1880.	Swansea, matte, 1880.	Briglia, cupreous pyrite.	Freiberg, matte.	Freiberg, matte for vitriolization.	Argo, Colo., ore, 1874.	Elizabeth port, N. J., kiln-burnt Spanish pyrite with salt for Longmaid process, 1883.	Imaginary, ore, 1883.	Lean matte, locality withheld. *a*
	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.
Authority	L. P., p. 170	R., p. 217	V	V., p. 21	C., p. 320	K., p. 175	K., p. 175	E	P. N	K., p. 207	
Dimensions of furnace:											
Length of hearth	17′ 2¾″	17′ 2¾″	28′		13′			30′	75′	30′	60′
Width of hearth	14′ 1½″	14′ 1½″	13′		15′ 5″			11′	10′	12′ 3″	12′
Length of grate	4′	4′						5′	12′		
Width of grate	3′	3′						2′ 8″	1′ 4″		
Ratio hearth to grate	19. 4	19. 4						19			
Depth of grate below bridge	4′	4′								3′ 3″	
Height of roof above bridge	1′ 7¾″	1′ 7¾″								1′ 9″	
Height of roof at velvetry	1′ 4″	1′ 4″						1′ 6″		1′	
Height of roof above hearth at bridge	3′ 2″	3′ 2″						2′ 4″		2′ 6″	
Chimney height	46′ *b*	46′							140′		
Chimney diameter inside	3′ 6″	3′ 6″									
Work done:											
Weight of charge in 2,000-pound tons	3. 8	4. 96	15. 68	4. 5	5. 88	0.55 to 0.77	0. 055	1			
Thickness of charge on hearth	4¾″							3″			
Hours roasted	12	36	12	36			12 to 16	24	12		
Tons output per furnace per twenty-four hours	7. 65	3. 3	28	8		4. 4 to 4. 96		3	20	7. 84	12
Composition of material roasted:											
Sulphur	24	29. 5		23					4		
Copper	10	33. 7		33		45 to 55	72				30
Iron	20	34. 2									
Sulphur in product, per cent	12, approx					6 to 8	1	0. 5 to 4	0. 7		
Smelted in reverberatory yields matte containing—											
Sulphur	30	21. 23	23								
Iron	30 to 33	5. 9									
Copper	30 to 33	65. 72	30 to 35	72 to 75				25 to 30			
and slag containing—											
Protoxide of iron	28. 5	58									
Silica	60. 5	35									
Copper	0. 5	1 to 1. 7									

Outlay of labor and fuel:											
Kind of fuel employed	An. 72 pts. Bit. 28 pts.	Bit. 28 pts. An. 72 pts.			Wood			Wood			Bit.
Men employed per shift	1							1	2		4
Days' labor per 2,000 lbs. of material roasted	0.35	0.70						0.67	0.18		0.66
Fuel, tons	0.123	0.41			0.34 cord	0.3 to 0.36		0.5 cord			0.208
Brick, tons	0.0036										

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a Roasted matte, smelted in a cupola, yields two parts of black copper and one part of matte containing 80 per cent. copper.
b Serves ten roasting furnaces.

C. = Capacci, Revue Universelle, 1881. X.
E. = Egleston, Transactions American Institute of Mining Engineers, IV, p. 421.
K. = Kerl, Grundriss der Metallhüttenkunde, 1881. I.
L. P. = Le Play, Description des Procédés Métallurgiques, etc.
P. N. = Private notes.
R. = Rivot, Principes Généraux du traitement des Minerais Métalliques.
V. = Vivian, Copper Smelting, its History and Processes, 1881.

Table IV gives the dimensions of several reverberatory roasting furnaces, the work done in them, and the outlay of fuel and labor, both in ore and matte roasting. A comparison of the first and second with the third and fourth columns shows how greatly the output, not only per furnace, but also per square foot of hearth area, has been increased in modern practice.

2. Muffles afford a somewhat more strongly oxidizing atmosphere than open reverberatories, and a more uniform temperature, and there is less loss from flue dust. This latter is not, however, a very weighty consideration, except in the case of rich ores and those carrying considerable silver or gold besides copper, and the considerably greater consumption of fuel in muffles generally renders them less desirable for ordinary ores than open reverberatories. At Lambert's works,[1] and at Polvereda, and many other points in Chili excellent and highly economical work is done in muffles heated with the waste heat from ore smelting reverberatories. As the fuel item is thus entirely eliminated, the muffle becomes under these conditions perhaps the best of appliances for fine ores lean in sulphur.

3. Of mechanical reverberatories, from lack of space, I can only mention Gibbs's[2] mechanical furnace, in which the hearth revolves, the ore being stirred by fixed knives which plow through it as it travels around on the hearth. Though of great promise it is said to have been abandoned. A description of Park's,[3] MacDougall's,[4] and O'Harra's[5] furnaces would carry us beyond the limits of this paper.

The preceding figures of the consumption of fuel and labor in roasting a ton of ore are briefly summed up in the following table. While these figures are by no means strictly comparable, since different ores are treated in each case with different degrees of thoroughness and of skill and with somewhat different objects, yet they are so nearly comparable as to be instructive. It should be remembered that the roasting in reverberatories, which aims only at the partial oxidation needed as a preliminary to smelting, is intentionally less thorough than that in kilns and in maletra-burners, in which the most complete possible expulsion of the sulphur, for conversion into sulphuric acid, is sought.

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[1] Copper Smelting, its History and Processes, Henry Hussey Vivian, Scientific Publishing Company, New York, 1881, p. 17.

[2] Lunge, op. cit., p. 208.

[3] Roasting Gold and Silver Ores, G. Kustel, 1880, p. 81.

[4] Lunge, op. cit., p. 222.

[5] Kustel, op. cit., pp. 82–84.

TABLE V.—*Showing the consumption of fuel and labor in roasting one ton of ore by different methods.*

Mode of treatment.	Labor.	Fuel.	
		Quantity per ton of ore.	Kind.
	Days.		
In piles	0.13 to 0.30	0.007 to 0.05 cords	Wood.
In kilns	0.25 to 0.41	Insignificant	
Maletra's shelf burner	0.56 to 0.67	do	
Reverberatory furnaces	0.35 to 0.70	0.123 to 0.5 tons	Coal.

OBJECTS OF THE ROASTING.

1. *An oxidation of the iron and incidentally of the sulphur as complete as is possible without involving an undue loss of copper in the slags of the following smelting.*—If the oxidation be very imperfect, the resulting matte will contain so much iron that its bringing forward will be unduly costly. If the oxidation be too thorough, an undue loss of copper will occur on smelting the roasted ore, first, from the individual shots of the resulting matte carried out by the slag being very rich in copper; and second, sometimes from actual scorification of copper. Of course, what an undue loss of copper is depends on local conditions: if labor and fuel are dear, if heavy freight is to be paid on the copper produced, and if the price of copper itself is low, as at present, we should aim in the first smelting for a product much richer in copper than when the opposite conditions prevail, since under the former supposition the saving of assumedly high-priced fuel and labor due to having a rich instead of a lean matte to "bring forward" to the condition of ingot copper would weigh more heavily against the increased loss in the slags of assumedly low priced copper.

It is rarely for the interest of the proprietors to produce the cleanest possible slags.

Thus, at Swansea,[1] where fuel and labor are very cheap, and where copper is of great value, owing to the immediate neighborhood of the market, they aim at a matte with from 30 to 35 per cent. copper; in the Copiapó district in Chili, however, where fuel and labor are costly and copper comparatively low in value, owing to the great distance from market, they aim at obtaining a 50 per cent. matte in the first smelting.

2. *The expulsion of arsenic and antimony.*—As already stated, these elements may be volatilized as sulphides, and in the partially oxidized condition of arsenious acid and trioxide of antimony. Mildly oxidizing conditions, which will tend to form these imperfectly oxidized compounds, rather than non-volatile fully oxidized arseniates and antimoniates, and which may even allow the arsenic and antimony to be attacked by the volatilized sulphur, are thus more favorable to their expulsion than are strongly oxidizing conditions, which would hurry them through

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[1] Vivian, op. cit., p. 14.

the intermediate volatile state to the stable condition of arseniates and antimoniates, and would rapidly oxidize the volatilized sulphur, and thus prevent it from volatilizing the arsenic and antimony as sulphides.

(*a*) In pile and stall roasting, the *exterior* portions of the individual lumps in the middle and upper layers of the piles are indeed exposed for a long time to the passing sulphur, which is volatilized from the lower layers, and to air whose oxidizing power is greatly weakened, part of its oxygen having been absorbed and fixed by the oxidation of the iron and part having been converted into sulphurous acid. The *interiors* of the lumps, however, especially in compact ores, are not so fully exposed to these mildly oxidizing conditions and to contact with volatilized sulphur, both so favorable to the expulsion of arsenic and antimony. What sulphur volatilizes from the interior of the lumps generally escapes through a few fissures, from the immediate vicinity of whose walls it may indeed drag away a portion of these hurtful elements; but the arsenic and the antimony in the greater portion of the interior of the lumps are to a certain extent protected from removal. Hence we see that their expulsion is favored by having as large a proportion of exterior to interior as possible; (that is, by having the lumps as small as is compatible with allowing regular burning and proper draft.

For the same reason, in matte roasting it is desirable to cast the matte in thin plates rather than in irregular thick lumps, since the proportion of surface is much greater in the former. The expulsion of arsenic and antimony as sulphides is favored by mixing pyrite with the ore as intimately as possible, so as to increase their exposure to the volatilizing sulphur; and Rivot[1] recommends bricking pulverized arsenical ores and cupreous pyrite together in intimate mixture, so as to allow the sulphur volatilized by the pyrite to drag off the arsenic.

It is evident, also, that the expulsion of arsenic and antimony as sulphides is in pile roasting much more favored in ores where the minerals carrying these elements (such as mispickel, tetrahedrite, enargite, etc.) are naturally intimately mixed with pyrite than when the reverse is the case.

(*b*) In roasting in reverberatories, during the intervals between the rabblings, and while the ore is lying still on the hearth, we may distinguish three horizontal zones: First, there is the upper surface, directly exposed to the atmosphere of the furnace whose oxidizing action is slightly weakened by the presence of sulphurous and sulphuric acids evolved from the underlying layers and of the products of the combustion of the fuel; second, the layers immediately underlying the surface, in which a small portion of free oxygen enters by diffusion, and in which sulphur and sulphurous acid evolved from the undermost layers oppose the oxidizing action of what free oxygen there is present; third, the undermost layers, to which little, if any, free oxygen can penetrate, and where

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[1] Op. cit., p. 142.

each pellet of ore is simply exposed to the action of the other pellets with which it is in contact, of volatilized sulphur and of sulphurous and sulphuric acids (anhydrides) generated by the action of sulphur on previously formed metallic oxides.

The expulsion of arsenic and antimony as sulphides is favored in the middle and lower zones by the presence of volatilized sulphur, mixed with sulphurous acid and at most a very limited supply of free oxygen and sulphuric acid. In the upper part of the middle layer, to which a small amount of free oxygen penetrates, we have the gently oxidizing conditions favorable to the formation of arsenious acid and trioxide of antimony. In the upper zone the stronger oxidizing conditions rather favor the formation of fixed arseniates and antimoniates, though even here part of the arsenic and antimony may volatilize and escape while passing through their intermediate volatile condition of arsenious acid and trioxide of antimony.

Now, when we come to stir the mass up and to transfer particles, whose arsenic and antimony, while in the upper zone, had passed to the fixed acid condition, down into the middle and lower zones, it is probable that we afford these metals a fresh opportunity to volatilize by reducing them to the volatile conditions of arsenious acid and trioxide of antimony through contact with volatilized sulphur and with the still unoxidized metallic sulphides of the surrounding ore pellets.[1] The low oxides (protoxide of iron and suboxide of copper) of these ore pellets and the sulphurous acid present in the middle and lower zones also exert a reducing action on arsenic acid, with the formation of higher oxides of iron and copper and of sulphuric acid.[2] Thus each individual atom of arsenic (and of antimony?) may travel back and forth many times through the volatile condition, being oxidized to and perhaps past it at the upper surface, and again reduced back to if not past it as it is again drawn down below the surface, seesawing back and forth to and from the volatile state. And every time that it arrives at this volatile condition an opportunity is offered it to volatilize and escape. This opportunity for reduction from the fixed acid state is, of course, only offered to these noxious metals as long as a considerable amount of unoxidized sulphur and iron or of low oxides of iron or copper remains to effect their deoxidation. But, even after the nearly complete oxidation of the sulphur and iron, similar opportunities may be, and indeed often are, offered to the arsenic and the antimony to pass once more from the fixed acid state towards the metallic state and back again, by mixing a small quantity of coke or coal dust with the roasting ore, and then excluding the air as completely as possible so as to strengthen the reducing conditions. The carbon plays a role similar to that already ascribed to the sulphides and low oxides in reducing arsenic and anti-

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[1] The behavior of antimony under these conditions is not so clear as that of arsenic.

[2] Balling, Compendium der metallurgischen Chemie, p. 51.

mony to and perhaps through the volatile state. By prolonging the roasting after the combustion of the coal dust has been completed we reoxidize any arsenic and antimony which may have been carried back past the volatile state, and we again pass them towards it.

Of course, the expulsion of arsenic and antimony is favored by the presence of a large proportion of pyrites, both because the sulphur distilled from the pyrites tends to drag them off as sulphides and because the presence of the pyrites prolongs the roasting, and thus increases the number of times which the arsenic and antimony pass back and forth past their volatile conditions; hence it is sometimes desirable to mix pyrites with impure ores to further the expulsion of their impurities.

(*c*) In comparing the heap and the reverberatory treatment, it may be said that this alternation of oxidizing with reducing conditions in the reverberatory should offset the advantage as regards the expulsion of arsenic and antimony which pile burning offers in its prolonged exposure to gentle oxidizing conditions. The further advantages in this regard which the reverberatory offers—in the enormous surface exposure (owing to the fine state of division of the ore, which in piles is in large lumps, whose interior may be comparatively little acted on), in the uniformity of its action (pile roasting being necessarily very irregular), and in the power of reducing arsenic and antimonic acids with coal dust—render it probably much more efficient for the expulsion of these metals than pile roasting is. Exact comparative tests are, of course, needed to enable one to speak with absolute certainty on this head.

The advantage of the reverberatory over pile roasting as regards the expulsion of arsenic and antimony is probably considerably less in the case of ores in which the arsenical and antimonial minerals are intimately mixed with pyrites than when the reverse is true, since in pile roasting in the former case the sulphur volatilized from the pyrites has a far better opportunity to drag off these metals as sulphides than when the antimonial and arsenical minerals are well separated from the pyritous portions. In the reverberatory, on the other hand, the whole is so mixed together that the original mechanical condition of the ore in this respect matters little.

2. SMELTING THE ROASTED ORE IN REVERBERATORIES.

FURNACE CONSTRUCTION.

Fire-box.—The distance from the bars, which are of wrought iron, to the top of the fire-bridge is generally very great, often upwards of 3 feet, to allow a thick bed of clinker on which the coal rests. The clinker both preheats the air and prevents fine fuel from falling through the bars. The area of the grate is generally about one-fifth of that of the hearth, though it should be larger than this in case the fuel be either unusually bulky or of low calorific power, as in the case of wood, peat, lignite, etc. Gruner[1] gives the following proportions of hearth area to grate area:

Copper refining furnaces, 4:1.

Copper matte smelting furnaces, 5 or 6:1.

Black copper furnaces, 4.50:1.

For exceptionally refractory ores the area of the grate should be more than one-fifth that of the hearth; for exceptionally fusible ones, proportionally less.

If we base the area of the grate on the weight of fuel which is to be burnt on it, then we should allow about 25 pounds of coal per square foot of total grate surface per hour. The fuel consumption sometimes falls as low as 16 pounds of coal per square foot of grate per hour and sometimes rises to 36 pounds, but about 25 pounds appears to be the best practice. It should, of course, be influenced by peculiarities in the nature of the fuel, strength of draft available, etc.

Special varieties of coal may, of course, require special forms of grate, but the deep rectangular grate with a deep bed of clinker is nearly universally used, even for fine coal.

Fire-bridge.—The longitudinal thrust of the hearth, due to its expansion on being heated, has to be met by inserting in the fire-bridge a very strong iron plate, or, better, a cast-iron box-girder, called a bridge-plate. The bridge-plate carries the stress either to the longitudinal walls of the fire-box, which in turn carry it to the buckstays at the rear end of the fire-box, which are held firmly to buckstays at the front end of the furnace by longitudinal tie-rods; or, better, the bridge-plate rests directly on the buckstays at the rear of the main body of the furnace, which are also firmly held by longitudinal tie-rods. To protect the bridge-plate from the heat the fire-bridge must be very thick, and to the same end, as well as to retard its corrosion, a current of cold air should pass through it to cool it. An arrangement adopted by the writer is to make the arch on which the furnace bottom rests hollow, so that a current of air may pass through it and the bridge-plate and by means of a damper be either admitted beneath the grate-bars or, in case it is desired to increase the oxidizing action of the furnace, directly into the laboratory. This arrangement intercepts and utilizes the heat which would otherwise escape from the bottom of the furnace. The same thing can of course be effected in many other ways.[1]

The inwalls rest directly on the lower arch. They should be laid all headers.

The roof is also laid all headers, and in a very flat arch, whose generatrix is parallel with the major axis of the furnace. The arch is made as flat as is consistent with stability; for ordinary widths a rise of one-half inch for each foot of span suffices. The roof bricks are laid in courses

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[1] Traité de Métallurgie, I, p. 269, 1875.

2½ inches thick, the courses being parallel with the major axis of the furnace; that is to say, the length of the brick (9 inches) is normal to the soffit or intrados, the bed joints (the 4½-inch dimension of the brick) parallel with the major axis of the furnace, the side joints (the 2½-inch dimension of the brick) parallel with the minor axis, and consequently perpendicular to the major axis. The side joints break joint at each course. This manner of laying the roof gives such stability that it enables us to use extremely flat arches. It is no uncommon thing for the old thin roofs of copper-smelting reverberatories, when cooled down, to be actually lower in the middle than at the sides in certain places; that is, for large portions of the upper surface of the roof to be actually concave. When such a roof is again heated the strength of the bond just described is so great that when adjoining portions which had remained convex again become heated, expand, and rise, they raise the portion which had been concave.

The plan employed in puddling and heating furnaces of laying the roof bricks in 4½-inch courses parallel with the minor axis of the furnace, instead of in 2½-inch courses parallel with its major axis, while facilitating repairs, would, if used in the broad, flat arches of copper-smelting reverberatories, allow the separate courses to fall down, one by one, as has been shown by actual experience, since this manner of laying the roof does not give stability.

It is a usual rule to make the lower edge of the velvetry level with the upper front edge of the fire bridge.

Binding.—The transverse stresses to be resisted by the binding are the thrust of the roof and the lower arch and their expansion, and more especially that of the bottom, on being heated. Of these the expansion of the bottom is the most serious, because its mass is very great, and because the stress due to its expansion comes in the middle of the length of the buckstays. These stresses are resisted by a set of vertical buckstays at the sides of the furnace. Each buckstay is held in place by two tie rods extending from its top and bottom to a buckstay at the opposite side of the furnace. Each buckstay is thus a vertical beam supported at both ends (by the tie-rods) and loaded with an irregularly distributed load (the thrusts of the arches and the pressures due to the expansion of the arches and the bottom). The buckstays are rarely stiff enough for the stresses they encounter and generally bend a great deal, thus shortening the life of the furnace and greatly increasing the cost of repairs. Old rails are generally used, but the writer strongly recommends the use of light I beams. The buckstays should be of wrought, not of cast, iron, unless there is an unusual difference in cost in favor of the latter material. Cast-iron buckstays should be made of much greater strength than is required in case of wrought iron, since the latter material is so comparatively flexible that

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[1] Transactions of the American Institute of Mining Engineers, IX, p. 689.

if an undue stress comes on one buckstay it can safely yield enough to cause its excess of stress to be transferred to its neighbors, while if all are unduly strained their bending gives timely warning, and the tie-rods can be relaxed. With rigid cast iron, however, each buckstay has to uphold the load which happens to fall on it without bending, and its sudden snapping is the first intimation that it has been overloaded. This disadvantage of cast iron, however, may be met by having the tie-rods hold the ends of the buckstays not rigidly but through powerful spiral springs.

The load on the buckstays, of course, varies greatly in different furnaces, but it is probably safe to assume that, in furnaces of ordinary widths, the total transverse stress is equivalent to a load of 5,000 pounds per running foot of the length of the furnace, concentrated at the middle of the length of the buckstays. From this may be calculated the dimensions of tie-rods and buckstays required to withstand it. The tie-rods beneath the furnace should be much thicker than this load requires, to allow for corrosion. They should be placed beneath the lower arch, so as to lessen their expansion from heat. Many engineers contend, and not without reason, that the lower ends of the buckstays should be held, not by tie-rods, which inevitably expand greatly from the heat, but by placing large flagstones against them bedded firmly in the ground. Expansion from heat becomes a still more serious thing in the case of the much longer longitudinal tie-rods which retain the lower ends of the buckstays at the ends of the furnace.

The side buckstays should be so long that the tie-rods at their upper ends may stand several inches above the roof when the latter is cold, as it rises greatly when heated, sometimes as much as 6 inches.

The chief longitudinal stress is that due to the expansion of the hearth or bottom. Like the transverse stresses, this is resisted by a system of tie-rods and buckstays, the stress at the fire-bridge end being carried from the hearth or bottom to the buckstays by the powerful bridge-plate already described.

These several stresses may be advantageously concentrated on the buckstays, both side and end, by bedding 3″ and 5″ angle-irons in the brick-work horizontally, and bearing directly against the buckstays, the 5-inch angle-irons opposite the bottom and the 3-inch ones opposite skewbacks of the roof and of the arch beneath the bottom. The thrust of the bottom at the front end is generally carried to the buckstays on either side of the skimming-door by means of a heavy cast-iron plate, greater stiffness for a given weight being sometimes obtained by making the plate very thick indeed, say 4 or 5 inches, but open-worked, like the cane bottom of a chair, the minimum of material thus being concentrated in a series of deep, stiff ribs.

The tie-rods may either be threaded at the ends, passing through holes in the ends of the buckstays, or formed into a loop and passing around the ends of the buckstays. In the latter case the tie-rods should be tightened with turn-buckles in their middles rather than with wedges at their ends; the extra cost of the turn-buckles is trifling, and the wedges are at best slovenly and liable to slip. To make the threaded portion as strong as the unthreaded, the diameter of the rods at the bottom of the threads should be equal to (or better, slightly greater than) that of the unthreaded portion.

The furnace is much neater in appearance and probably considerably more durable if it be all encased in wrought or cast iron plates reaching down to the ground level. This plan enables us to place the buckstays farther apart, concentrating their strength in a smaller number of stiffer buckstays, with stronger tie-rods; the total weight of material required for the buckstays is considerably reduced by this condensation, and the cost of the system of buckstays and tie rods decreased in a still larger proportion, thus partly compensating for the extra cost of the binding plates.

Erection of binding.—The rule should be to build the masonry against the iron work; and it should not be attempted to fit the iron work to the masonry. Thus, when the walls have reached the springing of the lower arches the buckstays and tie-rods should be put in their proper places and tightened up permanently. The angle-irons and the binding plates should be put in position as soon as the masonry reaches the level destined for their lower edges. In this way the masonry can be rammed firmly against the iron work and the greatest extent and uniformity of bearing obtained. All the iron work, but more especially the lower tie-rods, should be smeared with hot tar, and the tie-rods should be heated before applying the tar, so as to increase its adhesion.

The masonry.—All the brick work above and including the lower arch should be laid in fire-clay; it is a first rule to make all joints as thin as possible, since the clay joints are more fusible than the bricks themselves. For the roof and inwalls cutting of bricks should be avoided as far as possible; bricks molded to the proper shape (wedge, split, soap, jamb, etc.) being much cheaper and far more durable than chipped bricks, as the chipped surfaces are not, it is said, so refractory as the kiln-burnt surfaces.

The outer walls may be made of inferior fire-bricks, or even of red bricks. The roof should be of the most refractory bricks obtainable, such as the Dinas, Allentown, or Mount Savage. Many prefer to simply dip the roof bricks in thin fire-clay slurry on laying them, as the thinnest possible joints are thus obtained. Too great care cannot be bestowed on laying the roof. The key-bricks should be driven so tight as to raise the whole roof off the center on which it is built.

For the fire-box and the inwalls, at least for the portion of the inwalls which comes in contact with the charge, Dinas and similar silica bricks are not suited, since here it is not so much the heat as the corrosive action of the clinker and slag that is to be resisted. These parts should therefore be made of more basic or aluminous bricks. The Stourbridge bricks, which have as high as 45 per cent. of alumina, and the Garnkirk, are greatly esteemed for inwalls.

The brick-work of the roof, inwalls, fire-box, and bridge should be laid all headers. When laid in this way it lasts and holds together till the greater part of the length of the bricks has been worn away or melted off.

The bottom or hearth.—Its preparation is a most difficult matter. It must be sufficiently refractory to resist the intense heat and it must be so solid that the matte or copper floating on it can find no crevice down which it can work. It is generally held that it cannot be made of brickwork; that no matter how tight its joints be nor how cunningly it be dovetailed together, the matte works down through the joints and gradually lifts the brick bottom up. It is therefore universally made monolithically of more or less pure quartz sand alone.

In Swansea an only moderately refractory sand is (or was formerly) used for making the bottoms, having the following composition:[1]

Silica	86.0
Lime	5.7
Magnesia	0.8
Alumina	1.6
Peroxide of iron	1.2
Carbonic acid, with trace of water	4.5
	99.8

At Bergen Point moderately refractory sand from Perkiomen is used with great success. From these instances it might be supposed that some degree of fusibility was necessary or desirable in bottom sand; but in the Copiapó Valley in Chili, and elsewhere, the bottoms made of pure crushed glassy quartz stand three years and more. Thus excellent bottoms may be made from materials of widely varying properties; but each material requires different treatment. Highly skilled workmen brought to New York Harbor from Britain could not make durable bottoms with the Perkiomen sand, which was used in a neighboring establishment with perfect success; and the sand they were accustomed to using had to be imported from Britain for them.

In making these pure quartz bottoms in Chili about one-third of the quartz required for the bottom is charged at a time, each portion being brought up to a bright red throughout with repeated stirring before the next portion is introduced. When all has been heated to a fair red heat the bottom is patted into shape (it is slightly dish-shaped, and from 2 to 3 inches lower at the tap hole to allow the furnace to drain) with a heavy flat dish-shaped tool. The doors are then closed, and the heat is raised to its utmost intensity; indeed a higher heat is needed in making the bottom than in the regular smelting work. Moderately fusible loam sand is piled against the exteriors of all the doors to keep the heat in, and the temperature is considered high enough when the sand on

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[1] Le Play, op. cit., p. 72.

the outside of the 3-inch thick tile doors has fused. This generally takes six to eight hours hard firing.

The doors are now opened, and the quartz (which is now quite plastic and viscid, being sintered for, say, one-half inch in depth) is beaten into shape in the tap hole. As soon as the bottom has cooled to a dull cherry red, a layer of three-fourths of an inch of loam sand is thrown on it, with the double object of preventing the charge from indenting it and of sintering the upper surface a little more completely. A light charge of fusible ores and scrap matte is now introduced and melted. The matte is mostly absorbed by the bottom. The charges are now rapidly increased in weight till the normal charge is reached. In making the bottoms of roasting and refining furnaces the same course is followed, but the seasoning, as it is termed, is done with slag and scrap copper instead of ore and matte. The matte or copper is at first rapidly and later slowly absorbed by the bottom, which is finally converted into a solid monolith of matte or copper mixed porphyritically with quartz. While the bottom is new care is taken not to indent it in charging, nor to allow the heat to fall below a cherry red, as below this temperature the bottom is liable to crack if currents of cold air strike it. If it does crack it must be quarried out and the whole operation begun again.

Size and shape of hearth.—Since all parts of the hearth must be readily accessible from the working doors for rabbling and fettling, its corners are rounded off and filled in. The hearth must not be so wide that the furnaceman's tools cannot reach conveniently at least to its middle, nor so long that the flame cannot reach with full power to its farthest end. It may be that the greatest length attainable with direct firing has been reached, and experience thus far seems to indicate that with the present length of hearth a width greater than that now used is not desirable. To bring the temperature at the end farthest from the grate up to that of the remainder of the furnace the hearth is narrowed there and the roof lowered; indeed, it is a general rule in reverberatory furnaces that the temperature is locally raised by narrowing the furnace and lowering the roof.

The flue is generally made narrow and short, and rises to the chimney with a steep inclination. The short and strongly inclined flue certainly favors strength of draft by diminishing the angle which the products of combustion have to turn on entering the chimney and by diminishing the friction of their passage through the flue. Narrowness of flue tends to check the draft, but it has the compensating advantage of giving the products of combustion a high velocity in the flue itself, so that it is kept hot for a long distance from the furnace and the radiation of heat from the laboratory into the flue is very small; but these short narrow flues, with an independent chimney for each furnace, are liable to cause a heavy loss in flue-dust.

CHIMNEYS.

Draft.—In the draft of chimneys we may distinguish between intensity and volume. Volume of draft is due mainly to area of cross-section, intensity to the height of the chimney and to the difference between the temperature of its contents and that of the external air. Of course these two factors influence each other. The friction in narrow chimneys checks the intensity, and intensity of draft, by increasing the speed of the gases, increases the volume passing a given cross-section in the unit of time. The quantity of air which a chimney is able to draw through the interstices between the fuel on the grate, which is, of course, proportional to the amount of fuel which can be burnt per minute by the draft of the chimney, is, neglecting friction—

1. Proportional to the sum of the areas of the interstices between the lumps of fuel (and hence proportional to the grate area), until this sum becomes as great as the area of the chimney itself or of the smallest aperture in the flues leading to it.
2. Directly proportional to the square root of the height of the chimney.
3. Nearly directly proportional to the temperature of the gases in the chimney. The weight of gas passed through the chimney, however, reaches its maximum when the hot gases in the chimney are at 604° F. (273° C). When the gases become hotter than this they indeed travel more swiftly through the chimney, but they are so greatly expanded that the actual weight of the gases passed through the chimney per minute diminishes. Hence for the maximum draft the hot gases in the chimney should be nearly, but not quite, hot enough to melt lead.

According to Gruner, the velocity of the hot gases in the chimney should not be over 10 to 16 feet per second.

If we base the area of the internal horizontal cross-section of the chimney on the weight of coal which is to be burnt in each hour by its draft collectively on all the grates which it serves, then it appears that we should provide for chimneys in general not less than 1 square foot of this area for each 120 pounds of coal burnt per hour, and that in some cases we should provide as much as 1 square foot of this area for each 75 pounds of coal burnt per hour. For copper-smelting furnaces the weight of coal to be burnt per each square foot of this area should not greatly exceed 75 pounds per hour, and in certain extreme cases (as, for instance, where the chimney is liable to become partly choked with accretions) as much as 1 square foot of this area is provided for each 50 pounds of fuel to be burnt per hour.

Further considerations on the dimensions of chimneys required for proper draft will be found in the immediately following paragraphs:

Separate chimneys.—Each furnace is usually provided with an independent chimney, built as close as possible to the skimming door; indeed, this arrangement is often, though erroneously, considered absolutely necessary. The chimneys have in general no damper, the draft through the furnace being regulated by the condition of the clinker in the grate, and being practically wholly suspended when the furnace is being charged or skimmed, by opening the skimming door, which allows a great volume of cold air to pass direct to the flue without entering the furnace farther than the velvetry, and to practically fully satisfy the draft of the flue. But for this checking of the draft during charging, a considerable loss in flue-dust would then occur; since at the moment of charging, the fine particles of ore while falling to the hearth would be much more liable to be carried off by the draft were it not thus checked than they are after they have come to rest on the hearth.

Chimneys serving each one furnace are usually from 50 to 60 feet high, and their inside area in a large number of cases which have come to the writer's notice varies from about one-fourth to one-half the total grate area. Their height is influenced by the contour of the immediately surrounding country, the "lay of the land," much higher chimneys being required when surrounded by hills or by very high buildings. The chimney should project several feet above the highest points of all buildings in its immediate vicinity.

Central chimneys, common to many furnaces, however, are desirable: First, when it is considered necessary to carry the products of combustion to a great height on account of the proximity of hills or very high buildings, or to avoid nuisance and injury to the neighborhood, the cost of a single very high chimney, of cross section large enough for ten furnaces, being not much greater than that of one of equal height but of internal diameter only large enough for one furnace, since stability requires large diameter; second, when very rich or argentiferous or auriferous ores or pulverized mattes are smelted, since here the value of a given weight of flue-dust is greatly increased, and long flues and settling chambers become desirable, their friction and cooling effect on the products of combustion making the employment of a quite high chimney indispensable for the sharp draft required. The experience at Bergen Point and elsewhere proves that the use of a common chimney for copper smelting reverberatories is not only wholly practicable but highly important as regards flue-dust.

For common central chimneys flues should be provided of great height: First, that their walls may afford large settling surface for flue-dust; second, that their large area of cross-section may cause the products of combustion to travel slowly and thus induce them to deposit their flue-dust; third, that much flue-dust may accumulate in them before reducing their effective cross-section to such a point as to unduly check the draft, so that it may only be necessary to clean them at long intervals; fourth, to allow abundant standing room when cleaning them out, diminishing the cost of that operation. Where the flues are bent, curves of long radius should be used, and the flues should be brought almost parallel before opening into each other, to avoid interference of cross currents, which seriously checks the draft; and where the separate flues discharge into the chimney they should be curved upwards strongly, and partition walls should be carried up inside the chimney to separate the currents of gas entering from the several flues till they become approximately parallel.

Copper-smelting reverberatories require very intense draft. Owing to the often very great length of the flues connecting the furnaces with common central chimneys, the gases reach these chimneys much cooler than in the case of separate chimneys built close to each furnace, thus greatly diminishing the intensity of the draft. To compensate for this it is necessary to give common central chimneys much greater height than is required for separate chimneys serving only one furnace each. The writer recommends a height of from 100 to 140 feet above the grate level for common central chimneys. According to Gruner little is in general to be gained by making chimneys more than 150 feet high.

In designing common-central chimneys we should bear in mind that the doors of some of the furnaces are liable to stand open, allowing great volumes of cold air to pass directly to the chimney and thus greatly tax its capacity, just when other furnaces need the sharpest draught. On the other hand a greater volume of gas can pass through a square foot of area in wide common central than in narrow single-furnace chimneys, other things being equal, since in the former the friction is less, the ratio of wall surface to area of cross-section being smaller. Furthermore, in a common central chimney the gases are cooler, owing to their passage through long flues, than in chimneys serving single furnaces, and hence are less bulky and require (other things being equal) smaller area of cross-section. All things considered, it is recommended that common central chimneys should have an area of cross-section somewhat larger than in the case of single-furnace chimneys, and that it should certainly not be less than one-fourth, but rather one-third of the total grate area of the smelting furnaces served. Kerl[1] recommends that the ratio of the least area of cross-section of chimneys in general to the *effective* grate area (*i. e.*, the openings between grate bars) should be from 1:1 to 1:2.

Shape.—Square chimneys are cheaper than round ones per thousand of bricks contained in them, since they are laid more easily, it being easier to plumb the work from the corners and to keep these corners square than to build the work truly circular. Hence a square section is properly preferred for low chimneys. Round chimneys, however, not only require less brickwork per square foot of internal cross-section for equal thicknesses of walls (the ratio of periphery to area being smaller in the circle than in the square), but the circular shape

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[1] Kerl, Grundriss der allgemeinen Hüttenkunde, 1879, p. 303.

is considered more effective as regards draft than the square, since in the corners of the square the increased friction affects the draft.

Stability.—The effective pressure of the wind, tending to overturn the chimney, is very much less on a round chimney than on a square one whose width is equal to the diameter of the round chimney, being, according to Rankine, only half as great. Hence stability requires that square chimneys shall have much thicker walls than are necessary for round chimneys of the same height and area of horizontal cross-section. These advantages make round high chimneys much cheaper than square high ones of equal intensity and volume of draft.

External buttresses on chimneys are not to be recommended. They, indeed, place material far from the neutral axis of the chimney, thus tending to increase its stability; but they increase the surface offered to the wind, and thus increase the wind stresses so much that stability is generally more cheaply obtained by using round chimneys of larger diameter.

The following rules should be observed to insure the stability of round chimneys:

(1) The outside diameter at the base should not be less than one-tenth of the height of the chimney.

(2) The weight of the chimney, according to Rankine, should be such that a pressure of about 55 pounds per square foot of a plane surface directly facing the wind, or 27.5 pounds per square foot of the plane projection of a cylindrical surface, shall not cause the center of pressure at any bed joint to deviate from the axis of the chimney more than one-fourth of the outside diameter at that joint.

Iron vs. brick chimneys.[1]—Iron chimneys are very much more costly than brick ones and their draft is less powerful, other things being equal, since their walls are much thinner and the gases become consequently much cooler during their passage than they do in brick chimneys. Iron chimneys, however, are built much more quickly, require less care in drying, occupy less room and are much lighter, often a matter of importance in case of bad foundations.

Linings.—To prevent the cracking of large brick chimneys they should have a lining of fire-brick 4½ inches thick separated from the main walls by an air space. The lining should not be bonded to the main walls, but should be at liberty to rise and fall freely as it expands and contracts with changes of temperature. A few bricks, however, should be allowed to project from the lining here and there, so as to touch the outer wall, and thus increase its stability.

Batter.—Brick chimneys are generally of nearly the same size inside at top and bottom, being battered on the outside to correspond with the diminution of thickness of their walls in height. This diminution is generally effected by offsets of 4 inches each on the inside of the chim-

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[1] Transactions of the American Institute of Mining Engineers, IV, pp. 108, 109.

ney, to avoid cutting bricks, the thickness of the walls being everywhere a whole number of brick-widths. The batter should correspond with the rules for stability which are given above.

GAS FURNACES.

Gas furnaces, both regenerative (Siemens), as at Stora-Kopparberg, and non-regenerative, have been employed for copper smelting and refining, but in many if not most cases (as at Mansfeld, Ansonia, and elsewhere) they have been abandoned, as it is said that they increase the cost of repairs by an amount greater than that saved by their economy of fuel. In the case of regenerative furnaces the repairs are very heavy, as the fine ore carried over into the regenerators while charging and the fine copper carried over during the boiling periods in "roasting-smelting" and refining furnaces rapidly corrode the regenerators. The present verdict against them is certainly surprising, and may yet be reversed. A regenerative gas furnace built by the writer for ore smelting in Caldera, Chili, effected a saving of about 30 per cent. in the cost of fuel. From a mathematical consideration of the subject the writer believes that while the Siemens system of reversing regeneration may for the very high temperatures required in steel smelting effect a considerably higher economy of fuel than systems of continuous regeneration, like those of Ponsard and Swindell, yet for the considerably lower temperatures required for copper smelting reverberatories these latter systems are likely to prove, under equally favorable conditions, more economical of fuel than the Siemens system.

TABLE VI.—*Examples of*

	Swansea, 1848.			
	Smelting roasted ore.	Smelting roasted coarse metal for white metal.	Smelting roasted coarse metal for blue metal.	Roasting-smelting raw blue metal for white metal.
	1.	2.	3.	4.
Dimensions of furnace:				
Length of hearth	13′			14′ 1½″
Width of hearth:				
At flue	2′ 3½″			2′ 2″
At middle	8′ 11″			10′
At bridge	7′ 4¾″			6′ 11″
Length of grate	4′ 8″			4′ 6″
Width of grate	4′ 3½″			4′ 4″
Depth of grate below top of bridge	3′ 11″			3′ 11″
Height of roof:				
Above bridge	1′ 7¾″			1′ 7¾″
Above hearth at flue	1′ 4″			1′ 4″
Height of bridge above hearth	1′ 4½″			1′ 4″
Height of chimney	46′ 7½″			46′ 7½″
Size of chimney inside	2′ by 2′ 4″			2′ by 2′ 4″
Size of flue inside	1′ by 1′			
Ratio of grate area:				
To cross section of chimney	4.3 : 1			4.2:1
To hearth area	1 : 6			1:5 (approx.)
Work done:				
Length of operation	$4^h 20^m$	$5^h 55^m$	$5^h 55^m$	$11^h 49^m$
Weight of charge in 2,000-lb. tons	1. 43 *a*	1. 76 *a*	2. 2 *b*	2. 2 *c*
Composition of charge:				
Sulphur	14. *b*	10. 8 *a*	16. 1	23. 3
Iron	22. 2 *b*	22. 8 *a*	32. 6	18. 5
Copper	11. 3 *b*	32. *a*	32. 6	57. 2
Composition of product:				
Sulphur	29. 5	20. 5	23. 3	20. 1
Iron	34. 2	6. 2	18. 5	2. 2
Copper	33. 7	73. 2	57. 2	77. 5
Composition of slag:				
Protoxide of iron	28. 5	56.	54. 4	51. 6
Copper	0. 5	4. 5	2. 8	6. 3
Alumina	2. 9	1. 0	0. 8	1. 0
Lime	2. 0	1. 0	1. 2	2. 0
Silica	60. 5	33. 0	36. 0	34. 7
Magnesia				
Outlay of labor, fuel, etc.:				
Kind of fuel	32 pts. bit. culm 68 pts. ant. culm	26 pts. bit. culm 74 pts. ant. culm	26 pts. bit. culm 74 pts. ant. culm	23 pts. bit. culm 77 pts. ant. culm
Weight of fuel per ton burden treated:				
For regular work, tons	0. 624	0. 654	0. 475	0. 712
For extra work, tons	0. 153	0. 300	0. 231	0. 108
Total, tons	0. 777	0. 954	0. 706	0. 820
Days of labor per ton burden treated:				
Furnacemen	0. 35	0. 32	0. 230	0. 22
Helpers	0. 24	0. 17	0. 131	0. 51
Total	0. 59	0. 49	0. 361	0. 76
Bricks consumed, tons. Per ton burden treated.	0. 01 *b*	0. 012 *b*	0. 009	0. 006
Fire-sand, 2,000 lb. tons	0. 002	0. 062	0. 045	0. 136
Fire-clay, 2,000 lb. tons.	0. 003	0. 012	0. 002	0. 007
Authority	L. P	L. P	L. P	L. P

a **Including slag and fluxes.**
b **Excluding slag and fluxes.**
c **No flux used.**

P. N = Private notes. (For columns relating to Caldera, direct from the books of the establishment.)
O = E. D. Peters, jr., private communication.
L. P = Le Play, Description des Procédés Métallurgiques, etc.

reverberatory smelting, &c.

Swansea, 1848.		Caldera, Chili, 1877.					Bergen Point, N. J., roasting-smelting white metal and black copper for blister, 1885.
Roasting-smelting white metal for regule.	Roasting-smelting white metal, &c., for blister.	Ore-smelting.	Smelting for blue metal.	Roasting-smelting for Chili-bars.	Chili ore-smelting.	Butte ore-smelting.	
5.	6.	7.	8.	9.	10.	11.	12.
		16′ 6″	16′ 6″	16′ 6″	14′ 6″	14′ 6″	15′ 4″
					2′ 2″	1′ 10″	
		11′ 6″	11′ 6″	11′ 6″	9′	11′ 2″	9′ 2″
		5′ 6″	5′ 6″	5, 6″	6′ 6″	6′ 2″	
		5′ 8″	5′ 8″	5′ 8″	5′ 6″	4′ 6″	3′ 9″
		5′ 3″	5′ 3″	5′ 3″	3′ 2″	4′	5′ 9″
		1′ 5″	1′ 5″	1′ 5″	1′ 8″	2′ 4″	3′ 4″
		1′ 4″	1′ 4″	1′ 4″	1′ 8″	2′ 2″	1′ 5″
		1′ 2″	1′ 2″	1′ 2″	1′ 1″	1′ 4″	
		1′ 2″	1′ 2″	1′ 2″	1′	1′ 4″	
						50′	
		3′ by 3′	3′ by 3′	3′ by 3′		3′ 2″ by 3′ 2″	
		3.2 : 1	3.2 : 1	3.2 : 1		1.8 : 1	
		1 : 5.2	1 : 5.2	1 : 5.2	1 : 5.2	1 : 4.5	
3h 48m	24h	7.5h	7.04h	24h		5h	24h
1.63	4.08	3.52	3.53 *b*	6.3 *c*		3.3	8 to 30h
20.1	18.					11	
2.2	4.8					24	
77.5	75.4	13.69 *d*	29.6 *e*	61.83 *f*		21. *c*	
18.5	0.8			Chili bars		25	
0.2	0.9					10	
81.1	98.3	48.74	66.5			65	
52.0	49.0					41.0	
11.0	20.	0.83	1.78			1.2	
1.0						4.0	
1.0						2.0	
34.0	26.2					48.0	
bit. and ant. culm.	30 bit. culm. 70 ant. culm	Bit. coal	Bit. coal	Bit. coal		Bit. coal	Bit. coal
0.324	0.534						0.493
0.048	0.060						
0.372	0.594	0.52 *b*	0.475	0.748		0.5 *b*	
0.16	0.15	0.18	0.166	0.32			0.28
0.30	0.22	0.36	0.334	0.64			0.28
0.46	0.37	0.54	0.500	0.96		0.31	0.56
0.002	0.005						
0.047	0.010						
0.006	0.008						
L. P	L. P	P. N	P. N	P. N	P. N	O	P. N

***d*, *e*, *f* The charge in these three cases was as follows:**

d.			*e.*			*f.*		
Material.		Per cent. of charge.	Material.		Per cent. of charge.	Material.		Per cent. of charge.
Kind.	Per cent. of copper.		Kind.	Per cent. of copper.		Kind.	Per cent. of copper.	
Sulphureted ores.	**14.11**	**46.89**	**Matte**	**48.75**	**40.22**	**Matte**	**48.4**	**4.87**
Oxidized ore	**13.33**	**53.11**	**Sulphureted ores.**	**14.46**	**1.62**	**Matte**	**66.6**	**86.38**
			Oxidized ores	**16.98**	**58.16**	**Oxidized ores**	**25.6**	**8.75**
		100.00						**100.00**
					100.00			

TABLE VI.—*Examples of*

	Boston and Colorado, ore smelting.	Mansfeld, smelting burnt matte.	Oker, smelting burnt matte.	Freiberg, smelting burnt matte.
	13.	14.	15.	16.
Dimensions of furnace:				
Length of hearth	16′ 1″	13′ 11″	13′ 1½″	12′ 1″
Width of hearth:				
At flue		1′ 6″		
At middle	10′	9′ 10″	8′ 10½″	8′ 10½″
At bridge		3′ 9″		
Length of grate	5′ 2″	4′ 0″	3′ 11″	3′ 8½″
Width of grate	2′ 7″	4′ 0″	3′ 3″	3′ 8½″
Depth of grate below top of bridge			2′	2′ 2″
Height of roof:				
Above bridge			1′ 6″	
Above hearth at flue			1′ 3½″	
Height of bridge above hearth				
Height of chimney	52′ 6″	111′	43′	187′
Size of chimney inside		2′ 9″ by 2′ 9″	2′ 4″ by 2′ 4″	
Size of flue inside		10″ by 10″		
Ratio of grate area:				
To cross section of chimney				
To hearth area	1 : 5.7	1 : 3.4 A		
Work done;				
Length of operation		6h	9 to 10h	5 to 6h
Weight of charge in 2,000-lb. tons		3 to 3.5	2	2.1
Composition of charge:				
Sulphur				
Iron				
Copper		40 to 45		
Composition of product:				
Sulphur				
Iron				
Copper		75		
Composition of slag:				
Protoxide of iron				
Copper				1
Alumina				
Lime				
Silica				
Magnesia				
Outlay of labor, fuel, etc:				
Kind of fuel				
Weight of fuel per ton burden treated:				
For regular work				
For extra work				
Total				0.52
Days' labor per ton burden treated:				
Furnacemen				
Helpers				
Total				
Bricks consumed, tons — Per ton burden treated.				
Fire-sand, 2,000-lb tons — Per ton burden treated.				
Fire-clay, 2,000-lb. tons — Per ton burden treated.				
Authority	K., p. 210	K., p. 180	K., p. 181	K., p. 181

a Including slag and fluxes.

reverberatory smelting, &c.—Continued.

Smelting and refining native copper.				Cwm Avon, South Wales (1884 ?).				
Hancock, Mich.	Pittsburgh, Park Bros.	Pittsburgh, Park Bros.	Pittsburgh, C. G. Hussey & Co.	Ore-smelting.	Smelting for red metal.	Smelting for blue metal.	Smelting for white metal.	Roasting-smelting for blister copper.
17.	18.	19.	20.	21.	22.	23.	24.	25.
...	12′ 10″ ...	11′ 3″ ...	10′ 0″ ...	16′ ...	16′ ...	16′ ...	16′ ...	16′ ...
...	...	...	3′ 0″ ...	...	...	...	...	...
...	9′ 9″ ...	7′ ...	7′ ...	12′ ...	12′ ...	12′ ...	12′ ...	12′ ...
...	6′ 9″ ...	3′ 10″ ...	4′ ...	...	...	...	...	...
...	6′ 9″ ...	4′ ...	4′ ...	4′ ...	4′ ...	4′ ...	4′ ...	4′ ...
...	4′ 9″ ...	3′ 1½″ ...	4′ ...	4′ ...	4′ ...	4′ ...	4′ ...	4′ ...
...	2′ 3″ ...	2′ 2″ ...	2′ 10″ ...	3′ 6″ ...	3′ 6″ ...	3′ 6″ ...	3′ 6″ ...	3′ 6″ ...
...	1′ 3″? ...	0′ 9″? ...	1′ 8″? ...	...	...	...	...	...
...	...	...	...	3′ 6″ ...	3′ 6″ ...	3′ 6″ ...	3′ 6″ ...	3′ 6″ ...
...	1′ 8″ ...	1′ 10″ ...	2′ 6″ ...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	10″ by 1′ 8″	10″ by 1′ 8″	1′ 6″ by 1′ 6″	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	1 : 3.63 ...	1 : 5.5 ...	1 : 3.25 ...	...	...	...	...	...
24^h ...	...	...	...	4^h ...	...	12^h ...	12^h ...	24^h ...
9 ...	10 ...	5 ...	5 ...	...	2. *a* ...	2. *a* ...	...	5 ...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
80 ...	...	...	...	...	20 ...	35 ...	50 to 60 .	75 to 85 .
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	20 ...	35 ...	50 to 60 ...	75 to 85	98 to 99 .
...	...	...	...	27. 5 ...	...	...	...	...
10 ...	...	...	...	0. 44 ...	2. 0 ...	...	...	...
...	...	...	...	6. 58 ...	...	...	...	...
...	...	...	...	4. 0 ...	...	...	...	...
...	...	...	...	57. 22 ...	...	...	...	...
...	...	...	...	4. 16 ...	...	...	...	...
Bit. coal.	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
0, 89 ...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
...	...	...	...	...	...	...	...	...
P. N ...	E ...	E ...	E ...	L ...	L ...	L ...	L ...	L ...

K = Kerl, Grundriss der Metallhüttenkunde, 1881, I.
P. N = Private notes.
E = T. Egleston, Transactions of the American Institute of Mining Engineers, IX, p. 688.
L = A. M. Levy, Revue Universelle, Vol. 16, No. 2.

SMELTING OPERATION.

The roasted ore, mixed with fluxes designed to form with the gangue and with the iron oxides produced in the preceding roasting a desirable slag, is introduced through a hole in the center of the roof, ordinarily closed with a tile, from a hopper which is sometimes fixed and sometimes on wheels running on an elevated track. Two workmen, one at the working door and one at the skimming door, now spread the charge as rapidly as possible. The doors are next closed and tightly luted, and the fire is urged. Three and a half to four and a half hours now elapse before the charge is again touched, the fire all the time being kept at its utmost intensity. About half an hour after closing the doors the upper surfaces of the more fusible portions of the charge have begun to melt; they filter across the still unmelted portions below, gradually reacting on them, and, collecting on the bottom, they leave fresh portions exposed to the flames. Soon black bubbles appear, from the formation of sulphurous acid by the action of the atmosphere of the furnace and of the metallic oxides on the undecomposed sulphides present. The ebullition increases in violence and then again decreases, till after three and a half to four and a half hours from the time of charging it has nearly ceased. The charge is now all melted except a few refractory lumps which adhere to the bottom and which are protected by the molten bath from the flames. To detach and expose them to the full heat of the furnace, the skimming door is now opened and the hearth diligently scraped with long rabbles, extending almost to the fire-bridge. This should detach all pieces adhering to the hearth, which should feel smooth and soapy under the rabble. If it feels rough and gritty, the door should be again closed and the bottom again rabbled after firing hard for half an hour or an hour. The appearance of black bubbles in any part of the hearth is an indication that fusion is still incomplete there, and we cannot in general completely detach the ore from the bottom of the furnace in any portion of the hearth where the fusion is still so incomplete that the black bubbling is still strong there. For this reason it is in general unwise to attempt rabbling the bottom until the black bubbles have almost entirely disappeared from every portion of the hearth, since a second rabbling would still be needed to detach the last portions of ore from the bottom and since the additional rabbling thus introduced would waste fuel and time as well as labor. Time and fuel would be wasted, because opening the door to rabble the furnace cools it very rapidly, first, by almost completely checking the draft, since the cold air rushing in at the skimming door almost completely satisfies the capacity of the flue, and second, because much heat radiates from the furnace through the open doorway. This cooling down is a much more serious thing than might at first be supposed, since in these copper-ore smelting furnaces the highest temperature we can possibly obtain is but slightly above the melting point of the ore, and if the temperature be thus temporarily lowered, much time and consequently much fuel will be consumed in recovering it.

Two cases, however, occasionally arise in which it is best to rabble the charge while bubbling continues locally in certain portions of the hearth, or possibly even over the whole of it.

In the first place, if, owing to careless charging, the charge has been placed in too great quantity in one part of the hearth, or if one part of the furnace works considerably colder than the rest, the ore will remain incompletely melted there, and local bubbling will continue after the nearly complete disappearance of bubbles from the rest of the hearth has indicated the general completion of the fusion. Now, if in such a case as this we deferred rabbling till this local bubbling had entirely ceased there would be great danger that in other portions of the furnace, where the charge had already been completely melted, the bottom would become so hot that the matte, which is very thin and penetrating, would work down into it, corrode it, and perhaps force up large lumps of it. If the working of the furnace proceeds normally, and the fusion of the charge is completed at about the same time in all the different portions of the hearth, there is little danger of the hearth becoming so hot as to be penetrated in this way by the matte to any injurious extent; for on introducing the cold charge into the furnace the bottom is greatly cooled, and the charge, when melted, is removed from the furnace shortly after its last cold portions have been scraped off the bottom, a new charge then being introduced which again cools the bottom before it has had a chance to get overheated.

In the second place, two rabblings may be desirable even when the ore is properly spread, if the furnace, and more especially if the bottom, has been unduly cooled. In this case the portions of the charge in contact with the bottom become heated only very slowly, and time and fuel may be saved by rabbling the bottom as soon as the upper portions of the charge have become thoroughly fused, scraping off as much ore as we can from the bottom, and then, an hour or so later, scraping off the remainder.

If, as nearly always happens, all adhering matter is removed at the first rabbling, the skimming-door is closed and the charge fired for thirty minutes to melt all yet unfused portions and to allow the matte to settle. The door is then opened again and the slag skimmed with long rabbles into the middle of a series of deep molds formed in slightly moistened sand on the floor immediately in front of the skimming-door. Between these molds are sand partitions reaching nearly as high as the sides of the molds, and over these partitions the slag flows from each mold, as soon as it is filled, to its neighbor. This arrangement allows most of the buttons of matte mechanically drawn out with the slag to settle in the central block of slag, called the plate slag, and few buttons pass beyond the two molds immediately adjoining it. Considerable skill is required to avoid drawing shots of matte out with the slag. The rabble must be dipped into it only very slightly and very gently, so as to avoid forming waves. Often the furnace is slightly cooled just before skimming, so as to stiffen the slag and make it easier to skim. In the hearths of some ore-smelting furnaces there is a basin at the tap-hole large enough to contain all the matte, which accumulates there as fast as it is released by the fusion of the lumps of ore. In this case the slag in the neighborhood of the skimming-door, where the skimming-rabble lays hold of it, rests directly on the bottom of the furnace, and the danger of drawing matte out with it is greatly diminished.

As soon as the slag is all skimmed the tap-hole is opened and the matte tapped into sand or iron molds, or, in case it is to be roasted in reverberatories or Gerstenhöfers, it may be run in a thin stream into a deep basin of cold water to granulate it. The basin should be deep and the water supply copious to prevent explosions. Generally the matte is tapped only every second or, indeed, every third charge, the second charge of ore being dropped into the furnace immediately after skimming the slag.

If the matte is tapped only once for every two charges the slag of the first charge is not skimmed very clean, and, in skimming the second charge, when the greater part of the slag has been removed, the working door is opened to cool and stiffen the remaining slag, so that it may be skimmed more completely. If the slag is slightly stiff the rabble takes hold of it firmly and drags it out; if it be too fluid the rabble slips through it. The difference is like that between skimming slightly viscid cream from milk and skimming petroleum from water. In skimming very liquid slags it is moreover very difficult to avoid propagating waves, which tend to mix up the matte and slag and thus to increase the mechanical loss of matte. The rabbling, skimming, and tapping must not be unduly delayed for fear of corroding the bottom, as already explained. After tapping the matte the fettling is repaired, if necessary, with a mixture of sand and clay, say of one part of fat clay to three parts by measure of pure quartz sand. The inwalls, for some 4 or 5 inches above their contact with the hearth, should be protected by a thick layer of this fettling to prevent them from being corroded by the slag. Every time the matte is tapped the fettling must be carefully examined and new fettling rammed firmly in where it has become corroded.

A new charge is now introduced and matters proceed as before. Much care must be paid to spreading the ore so that the fusion will be completed at about the same time throughout the furnace, and it must nowhere protrude so high as to interfere with the flame.

Should the hearth become too high in any part the ore is left very thin there in charging, so that it may melt quickly, and so that the accretions may be melted off while the fusion of the ore is being completed in other portions of the furnace. If this does not suffice, the accretion may be removed by leaving it quite bare when charging the ore and then placing some cast iron on it, which will cut it down while the charge is melting.

The slag is removed from its molds while hot, as it becomes extremely brittle on cooling. It is indeed preferable to remove all the blocks except the plate slag to the dump, while still liquid inside, and, by breaking off a corner of each block, to allow the still liquid contents to run out, leaving a shell, the bottom of which contains the greater portion of the total copper contents of the block, and is resmelted if rich enough.[1]

In viscid slags the top also often contains enough copper to pay for resmelting. The plate slag is usually so rich that the whole of it has to be resmelted.

Time occupied in smelting one charge of ore in reverberatories.

	Chili (private notes).	Swansea. (Le Play).
Spreading charge and luting doors	0^{h} 20^{m}	0^{h} 11^{m}
Melting down	4 30	3 38
Rabbling	5	5
Completing fusion	30	16
Skimming slag	17	10
Tapping	10	
Fettling	8	
Total	6 0	4 20
Weight of charge (tons)	4	1.3

The details of the charges and products, consumption of labor, fuel, supplies, etc., in many different instances will be found in Table VI, columns 1, 7, 10, 11, 13, and 21.

The composition of the charge, as determining that of the resulting matte and slag, should be governed by the following considerations.

THE MATTE.

The proportions between (1) the unoxidized, (2) the oxidized copper, and (3) the sulphur, more especially that existing as sulphide of iron, should be such that the resulting matte shall be neither so lean in copper as to render its subsequent bringing forward too costly nor so rich as to cause undue loss of copper in the slag.

THE SLAG.

In general the proportion between the silica, alumina, oxide of iron, lime, etc., in the charge should be such that the resulting slag shall be (1) specifically light, so as to separate readily from the matte; (2) at least moderately fusible, so as to economize fuel and time in smelting it; (3) not excessively fluid when melted, so as to be readily skimmed, and (4) not so excessively viscid that the matte will not separate from it. The following are some examples of composition:

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[1] Vivian, Copper Smelting, its History and Processes, p. 14.

Table VII.—*Composition of reverberatory ore furnace slags.* (See also Table VI.)

	Ore furnace slag from Kaafiord.[a]	From Dillenburg.[a]	From Bendorfer Hütte.[a]	From Swansea.[b]		
	1.	2.	3.	4.		
Silica:						
Combined				30.0		
Uncombined				30.50		
Total	52.00	47.54	45.41	60.5	Silicate	65.40
Alumina	8.90	3.90	1.48	2.9	Fluoride of calcium	2.10
Protoxide of iron	17.64	28.98	36.65	28.5	Ferrous sulphide	0.55
Lime	16.22	15.59	15.88	2.0	Matte	1.45
Magnesia	5.45	3.26	0.67	0.6		
Oxide of manganese						
Oxide of zinc						
Miscellaneous				5.0		
Copper	(Cu_2O?) 0.40	(Cu_2O?) 0.73		Cu 0.5		
Total				100.00		100.00
Specific gravity				3.21		
Oxygen ratio of bases to silica.	1:1.83	1:1.78	1:1.76	1:3.6		

	From Montana.[c]	From Argo, Colo.	Parrot, Butte, Montana.	Cwm Avon, Wales.[d]
	5.	6.	7.	8.
Silica	48.0	39.7	48.00	57.22
Alumina			4.00	6.58
Protoxide of iron		31.2	41.00	27.50
Lime		2.54	2.00	4.00
Magnesia			1.5	4.17
Oxide of manganese	30.0	4.00		
Oxide of zinc	12.5	12.00		
Miscellaneous		PbO 3.66	S 0.8	S 0.12
Copper		0.30	1.2	0.44
Total				
Specific gravity		3.65	3.35	
Oxygen ratio of basis to silica		1:1.88		

[a] Kerl, Grundriss der Metallhüttenkunde, 1881, I, p. 211.
[b] Le Play, Procédés Métallurgiques Employées dans le Pays de Galles, etc, p. 216.
[c] Transactions of the American Institute of Mining Engineers, XI, p. 59.
[d] Levy, Revue Universelle, Vol. XVI, No. 2, p. 286.

Of these the Swansea slag, which is even more acid than a trisilicate, is described by Le Play[1] as a sort of pudding, formed of fragments of quartz and silicious rocks, almost in contact with each other, suspended in a matrix, consisting of a silicate more or less compact. This porphyritic appearance is common in reverberatory ore furnace slags, the undissolved grains of quartz accumulating at the tops of the blocks of slag, the grains of matte toward the bottom. While these suspended quartz particles increase the loss of copper, first, by making the slag so

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[1] Le Play, p. 204.

pasty that it tends to retain shots of matte, and, second, by retaining small portions of copper in the interior of the individual unmelted grains themselves, it is quite possible to obtain economical results with slags containing even more unmelted silica than the Swansea slag above. Indeed, it is rarely advantageous to add fluxes to any great extent to even an extremely silicious ore for reverberatory smelting. To reduce the proportion of silica in this Swansea slag so low that it could all readily melt would require a great addition of flux, entailing expense for the flux itself, diminishing the quantity of ore which each furnace charge can contain, thereby increasing the outlay of labor and fuel per ton of ore, and, by augmenting the quantity of slag, tending to increase the loss from mechanically suspended matte probably much more that it would be lessened by the increased fluidity of the slag. Here lies the main advantage of the reverberatory over the cupola method, in that mixtures which are practically infusible, taken as a whole, may yet be economically treated in the reverberatory by simply performing this partial fusion, portions of the different components combining together in some fusible ratio, the excess remaining unfused and floating in the fused portion as a sort of pasty pudding, which can be pulled out of the furnace in skimming. The completeness with which the copper can be recovered from highly silicious mixtures by this semi-fusion is well illustrated by the above Swansea slag, which only contains on an average, year in and year out, 0.5 per cent. copper, while actually containing 30 per cent. of uncombined silica, part or even the whole of which may have remained completely unfused throughout the whole smelting operation. And from the fact that most of the small amount of copper contained in the slag is found not in the upper porphyritic portion, but in the lower completely fused layers,[1] we may infer that the amount of undissolved quartz might be considerably increased without causing a more than proportionate loss of copper.

Of course, the amount of flux, if any, which it is profitable to add in any particular instance is a question of dollars and cents, which can often only be answered by calculations of profit based on actual tests on a large scale of the results of fluxing. But it is evident that excellent results, metallurgically considered, may be obtained in reverberatories with mixtures which could hardly be made to run at all in cupolas.

Basic mixtures may also be treated in the reverberatory. While it is likely that the heavy and very liquid sub-silicate of iron slags which can, though with difficulty, be treated in cupolas would corrode the sides and hearths so rapidly and carry off so much matte mechanically that their treatment in reverberatories could not be satisfactory, yet it is probable

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[1] In the mean of 18 samples from the top, 18 from the middle, and 18 from the bottom of slag blocks from six different fusions Le Play found (op. cit., p. 206) in the top 0.6 per cent. copper, in the middle 0.3 per cent., and in the bottom 1.3 per cent., or on an average 0.67 per cent.

that the viscid and refractory aluminous silicates and the comparatively basic silicates of lime can be treated in reverberatories with fairly good results, metallurgically considered, when so infusible that they would not run at all in cupolas.

Classification.—The more important reverberatory ore furnace slags may be classified as follows:

Name of slag.	Oxygen ratio of bases to silica.
Sub-silicate	2:1
Singulo-silicate	1:1
Sesqui-silicate	1:1.5
Bi-silicate	1:2
Tri-silicate	1:3

Fusibility.—1. Let us consider first the simple silicates. The order of fusibility among silicates with a single base appears to stand about in the following order: Iron singulo- (most fusible), iron sub-; manganous singulo-, manganous bi-, manganous tri-; iron bi-; lime tri-?, lime bi-?, lime singulo-?; iron tri-; magnesia bi-?, magnesia singulo-; alumina singulo-?; magnesia tri-?; alumina bi-?, alumina tri-? (least fusible).

In regard to several of the above the evidence is quite contradictory. The exact position of those followed by ? seems especially doubtful. It seems pretty clear, however, that silicates of the protoxides of iron and manganese reach their maximum fusibility when the oxygen ratio of base to silica is approximately that of singulo-silicate; that of magnesia when the oxygen ratio is about that of bi-silicate; with lime it is not clear whether the most fusible composition lies nearest the relation of bi- or that of tri-silicate; and with alumina whether it is nearest to that of singulo- or to that of bi-silicate. Experience appears to bear out the inference that the larger the proportion of protoxide of iron and of manganese to the lime and magnesia, the more basic should the slag be to secure the greatest fusibility; and the more acid the slag the larger should be the proportion of lime to the protoxide of iron and of manganese.

2. Multiple silicates may next be considered. We can greatly increase the fusibility of the simple silicates of any one of the alkaline earths (baryta, lime, and magnesia), or of alumina, by adding to it the silicate of any other earth. Thus the simple silicates of alumina are much less fusible than those of lime; yet if we add silicate of alumina to silicate of lime in proper proportions we get a compound silicate more fusible than the simple silicate of lime is. Of course, if we add a fusible silicate, such as that of protoxide of iron or of manganese, we increase the fusibility of the silicate of lime (or of any other alkaline earth) to a very much greater extent.

The fusibility of the silicates of lime is increased more by adding to them a silicate of baryta than by adding a silicate of magnesia or of alumina, and it is increased more by adding a silicate of magnesia than by adding one of alumina.

According to Bodeman, the most fusible silicate of lime with alumina is the bi-silicate of the composition lime 30, alumina 14, and silica 56 per cent., which corresponds to the formula $4\ CaO, Al_2O_3, 7\ SiO_2$. It is made less fusible by adding either silica or lime to it, and much less fusible by adding alumina to it. It is of course made much more fusible by adding oxide of iron or of manganese to it, even without altering the oxygen ratio of the bases to the silica.

Among the compound singulo-silicates of lime with alumina, which are less fusible than the compound bi-silicates of these bases, those rich in the more fusible base lime (such as $12\ CaO, 2\ Al_2O_3, 9\ SiO_2$ idocrase) are more fusible than those richer in alumina.

The order of fusibility among the compound silicates of lime with magnesia is not so clear. Here too the bi-silicates seem to be the most fusible. Among them the bi-silicate whose composition is expressed by the formula $CaO, MgO, 2\ SiO_2$ (pyroxene), which contains one equivalent of lime to one of magnesia, appears to be more fusible than the other bi-silicates richer in the more fusible base lime. An increase in the proportion of magnesia in this compound (the oxygen ratio of the combined bases to the silica remaining unaltered) appears to diminish its fusibility more than an increase in the proportion of lime does.

The silicates in each of the following groups are placed in the order of their fusibility, as far as it can be made out from the altogether unsatisfactory data at present available. Where the relative fusibility of two or more silicates is altogether uncertain they are grouped together with a brace. Each such group taken collectively is to be understood as less fusible than the silicates above it, and more fusible than those below it in its own column:

TABLE VIII.—*Order of fusibility of certain silicates.*

Approximate order of fusibility.	Bases of silicate.		
	Lime and alumina.	Lime and magnesia.	Lime and protoxide of iron.
Most fusible	$4\ CaO, Al_2O_3, 7\ SiO_2$	$CaO, MgO, 2\ SiO_2$	
	$3\ CaO, Al_2O_3, 6\ SiO_2$	CaO, MgO, SiO_2	$CaO, 3\ FeO, 2\ SiO_2$.
	$12\ CaO, 2\ Al_2O_3, 9\ SiO_2$	$2\ CaO, MgO, 3\ SiO_2$	$2\ CaO, 4\ FeO, 3\ SiO_2$.
	$3\ CaO, Al_2O_3, 3\ SiO_2$	$CaO, 2\ MgO, 3\ SiO_2$	CaO, FeO, SiO_2.
	$6\ CaO, 4\ Al_2O_3, 9\ SiO_2$	$CaO, MgO, 3\ SiO_2$	$4\ CaO, 4\ FeO, 5\ SiO_2$.
	$3\ CaO, Al_2O_3, 9\ SiO_2$	$CaO, MgO, 4\ SiO_2$	$CaO, FeO, 2\ SiO_2$.
Least fusible	$6\ CaO, 2\ Al_2O_3, 3\ SiO_2$		

Of these compounds the silicates of lime with magnesia appear to be as a whole somewhat more fusible than those of lime with alumina, while those of lime with protoxide of iron are very much more fusible still. But the present state of our knowledge hardly warrants us in comparing the fusibility of the individual members of the lime-alumina group with those of the lime-magnesia group. Oxide of zinc appears in general to make the silicates less fusible.

Specific gravity.—The presence of an excessive amount of protoxide of iron or of manganese in the slag is to be avoided, since these bases make the slag very heavy, thus interfering with the separation of slag from matte by gravity.

The corrosiveness of slag is a point which requires much attention in the reverberatory smelting, since the bottom and more especially the sides of the furnace are exposed for a long time and at an exceedingly high temperature to the action of the slag.

With a given oxygen ratio between the bases and silica, slags whose principal bases are protoxide of iron and manganese are far more corrosive than those in which lime, magnesia, and alumina, or any one of them, predominates. Potash and soda also make slags highly corrosive. In general, for a given relation between the different bases, the more basic the slag the more corrosive it is.

Liquidity.—Basic slags whose chief bases are protoxide of iron or of manganese are so extremely liquid that they cannot be cleanly and satisfactorily skimmed, and are hence much dreaded in the reverberatory. In the shaft furnace their liquidity is rather an advantage and partly compensates for their corrosiveness, heaviness, and tendency to form salamanders. Oxide of zinc in general tends to stiffen the slag, hence a larger proportion of this base is permissible in reverberatory than in cupola work and in ferruginous and manganiferous than in earthy slags.

Scorification.—The loss of copper in the slag has been spoken of up to this point as if it were wholly mechanical. In general the copper tends to scorify, that is to actually chemically combine with the silica of the slag, more strongly when acid slags are produced than when the slag is basic, the affinities of the silica in basic slags being more completely satisfied than in acid ones. The tendency to scorification, other things being equal, appears to be much greater when lime, alumina, and magnesia preponderate among the bases than when protoxide of iron or of manganese does. In sulphureted and especially in pyritous ores this tendency is strongly held in check by the action of the sulphide of iron; but when the copper exists in the ore mainly or wholly in the oxidized condition, scorification becomes a serious matter, and the power of ferruginous and manganiferous slags to check it should weigh strongly in deciding what slag is on the whole the most profitable.

To sum up, while the most advantageous slag for the reverberatory is probably about a sesqui-silicate of lime and protoxide of iron, with rather more protoxide of iron than lime, ore mixtures may be successfully smelted which yield slags of a very great range of composition, from subsilicates to tri- and even quadri- silicates. Very acid slags, if rich in alumina, should contain also protoxide of iron or of manganese, since if they contained only lime or magnesia, together with their silica and alumina, they would be too infusible. Highly basic slags, on the one hand, require a certain amount of protoxide of iron or manganese to make them fusible; but, on the other hand, these oxides must not preponderate too strongly among the bases, as they make the slag corrosive, heavy, and hard to skim.

CHEMICAL REACTIONS.

In general.—The roasted ore contains silica, oxides and sulphides of iron and copper, and earthy bases (lime, magnesia, alumina), etc. The oxides of iron, reduced to protoxide by the sulphides and partially also by the flames, eventually combine with the silica, as do also the earthy bases.

The oxidized copper, some of which is probably taken up by the silica at first, is deoxidized, chiefly if not almost wholly, by the protosulphide of iron (Fe S), which plays a most important part in copper smelting. The reactions of these different substances on each other is promoted (1) by the gradual trickling and filtering of the uppermost and first fused portions over and through the lower and as yet only pasty portions; (2) by the incessant ebullition of the enormous volumes of sulphurous acid evolved (estimated by Le Play at four hundred times the volume of the charge itself, and stated by Vivian[1] to contain 13 per cent. of the sulphur originally contained in the ore), which automatically causes a most thorough stirring up and mixture of the different portions of the charge, constantly bringing fresh portions of the already liquified parts into contact with the still unfused surfaces and ever altering the surfaces of contact of the already melted matte and slag; (3) by the remarkable power possessed by protosulphide of iron of dissolving in the slag. Le Play[2] pointed out that the proportion of sulphur to copper in the slag was always much higher than in the matte (he found it 40 per cent. higher), whence it is inferred that in addition to the shots of matte mechanically suspended in the slag and wholly distinct from them there exists a considerable quantity of sulphide of iron. The absence of scorified copper in slags resulting from the fusion in a comparatively oxidizing atmosphere of mixtures containing at once large quantities of copper oxides and an immense excess of silica, whose powerful affinities for each other seem to offer conditions most favorable to scorification, is explained by the presence of protosulphide of iron in the slag itself, not in isolated buttons, which might not come in contact with the scorified copper, but apparently actually dissolved in it, and thus in the most favorable position for reducing by double decomposition what silicate of copper is formed during the fusion. $FeS + Cu_2SiO_3 = Cu_2S + FeSiO_3$. The fact which underlies this reaction, that iron has a vastly greater affinity for oxygen but only a slightly stronger affinity for sul-

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[1] Copper smelting, etc., p. 19.

[2] Le Play, Description des Procédés Métallurgiques, etc., p. 212.

phur than copper has, is the foundation on which are based all processes for smelting sulphureted copper ores.

Arsenic and antimony exist in the roasted ore partly as sulphides, arsenides, and antimonides, partly as arsenic and antimonic acids, free or combined with metallic oxides. The large quantity of sulphur and iron with which they are initially associated, and which have so much stronger affinity for oxygen than arsenic and antimony have, prevents that portion of these noxious metals which is present in an unoxidized condition in the roasted ore from being oxidized during the smelting to any considerable extent, so that this part of the arsenic and antimony passes directly into the matte. The arseniates and antimoniates, however, are decomposed by the silica, which tends to combine with their bases, setting arsenic and antimonic acids free. Were they isolated when set free, all the arsenic acid would be split up by the heat into free oxygen and arsenious acid which would volatilize, antimonic acid being reduced to tetroxide of antimony (Sb_2O_4), which is not volatile. But in the pasty mass in which the arseniates and antimoniates are split up by the silica, the presence of sulphur and iron in large quantities interferes with the simple action of the heat. While it is probable that a certain amount of arsenic is driven out as arsenious acid, most of the arsenic and probably nearly all the antimony are deoxidized and combine with the sulphur, part of the sulphides of arsenic and antimony thus formed probably volatilizing, the remainder and probably by far the largest part entering the matte.

The net result of the smelting on the arsenic and antimony contained in the roasted ore is to volatilize part of these metals, the remainder combining with the copper in the resulting matte.

3. ROASTING THE FIRST MATTE (COARSE METAL).

THE AIM.

The aim of this operation is to oxidize the iron, with a view to scorifying it in the following smelting operation, to a smaller extent to oxidize and thus volatilize the sulphur, and, at least in certain cases, to oxidize and volatilize the arsenic and antimony.

The extent to which the roasting is pushed depends (1) on whether the expulsion of arsenic and antimony receives special attention; (2) on whether oxidized ores are available for the subsequent smelting; (3) on whether rich sulphides nearly free from iron, arsenic, and antimony are to be had; (4) on the cost of labor and fuel.

These four considerations should determine whether it is best to push this roasting so far that the roasted matte shall reach the state of metallic copper in two operations (see p. 16), as in the "ordinary process"—(4) smelting for white metal and (5) roasting-smelting for blister copper, the Welsh "roasting"—or to limit it so that the metallic state shall only be reached after three operations, as in the "extra process"—(4*a*) smelting for blue metal, (4 *b*) roasting-smelting for regule and bottoms, and (5) roasting-smelting for blister copper.

1. If the expulsion of arsenic and antimony is not especially aimed at, as has generally been the case in this country, the roasting of the first matte may be pushed very far, so that on smelting this matte we may obtain a second matte which is very rich in copper, and which may therefore be brought forward to the condition of metallic copper with a small number of operations, and hence cheaply.

If, however, the expulsion of arsenic and antimony is especially aimed at (as is the case, 1st, when even fairly good copper is to be extracted from impure ores; 2d, when exceptionally pure copper is to be obtained from any but very exceptionally pure ores), enough sulphur and unoxidized iron must be left after smelting the roasted matte to allow three subsequent operations to be performed before reaching the stage of blister copper; for, the metallic state once reached, the expulsion of arsenic and antimony becomes far more difficult. Though an incomplete roasting doubtless expels less arsenic and antimony in itself than a thorough one does, yet, by leaving more sulphur and unoxidized iron in the roasted matte, it compels us to submit it to one more oxidizing operation than it would receive were the roasting thorough; and this additional operation increases the expulsion of these elements to an extent which far more than compensates for their being less fully expelled in the incomplete than in the thorough roasting.

2. Oxidized copper ores mixed with the roasted matte in the following smelting operation raise the grade of the resulting matte by oxidation and removal of part of the iron and sulphur; so that if such ores can be had at advantageous prices the roasting of the matte need not be so thorough, and hence not so costly, as would otherwise be desirable. Indeed, where oxidized ores are abundant, as in the desert of Atacama, in Chili, the roasting of the matte is generally wholly dispensed with if only fair copper (G. O. B.) is aimed at.

3. If rich sulphureted copper ores free from arsenic and antimony are available for the subsequent smelting, we may push our roasting further, and thus favor the expulsion of arsenic and antimony, and yet obtain on smelting a product containing enough iron and sulphur to enable us to submit it to the three operations of the extra process before reaching the metallic state. Indeed, if the original materials be very impure, or if the most complete possible expulsion of arsenic and antimony be desired, it may be advantageous to add enough pure sulphurets on smelting the thoroughly roasted matte to produce a new matte so rich in sulphur and iron that it may be submitted to an extra roasting and smelting in addition even to those of the "extra process."

4. If the prices of labor and fuel were high, that would incline us to limit the number of operations, and hence to push the roasting far, so that on smelting the roasted matte we would obtain a matte rich in copper and therefore cheaply "brought forward" to the metallic state.

In point of fact, variations in the four conditions just mentioned, combined with local traditions, prejudices, and whims, do endlessly modify the methods practiced. But the complexity and the great number of operations of the reverberatory process, so often complained of as senseless, really have at least a partial justification in the fact that the expulsion of arsenic and antimony, unlike that of sulphur and iron, is not favored by a small number of oxidations, each pushed very far, so much as by a greater number of less costly and less thorough oxidations and reductions; for while the fullest state of oxidation of sulphur and iron is not unfavorable to their expulsion (sulphates of iron and copper are readily decomposed with expulsion of their sulphur in reverberatory roasting and smelting), the complete oxidation of arsenic and antimony prevents their expulsion, since it is the partially not the completely oxidized compounds of these elements that are volatile. Furthermore, the total cost of a long series of operations may exceed but slightly that of a smaller number of more costly ones.

Matte roasting, unlike that of ore, is rarely limited by fear of loss of copper in the slag produced in the subsequent smelting, since this slag is almost always resmelted at any rate. This idea could not, of course, be pushed so far as to slag an important proportion of the total copper, as the cost of recovering a unit of copper from the slag is considerably greater than from the matte produced with it; since the slagged copper, after the slag containing it is smelted in the first fusion operation (2), has to undergo two operations (3 and either 4 or 4 *a*) before reaching the condition it would have been in had it passed originally into the matte instead of into the slag. But the roasting is generally limited by other considerations before proceeding so far as to slag an amount of copper so large that any considerable expense is caused by its extra treatment.

THE OPERATION.[1]

If in lumps the matte may be roasted in heaps, or, far better, in stalls or kilns; in either case repeated firings are needed. If in powder, reverberatories (either open or muffles, or Gerstenhöfer furnaces) may be employed. The use of revolving cylinders has not been successful, so far as the writer knows, but the difficulty found in roasting ore in these furnaces on account of its great plasticity would be greatly diminished in the case of mattes. Several attempts lately made in this country to roast mattes in Maletra's shelf-burners seem to have had indifferent results. Where the recovery of the sulphur of the matte as sulphuric acid is aimed at kilns[2] will probably be preferred for lump

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[1] Mineral Resources of the United States, 1883, p. 280.

[2] It is thought by many smelters in this country that matte cannot be successfully roasted in kilns. This is not correct; matte has been successfully roasted in kilns in several places in Europe, even for acid making. (Kerl, Grundriss der Metallhütten-kunde, pp. 174, 207.)

matte, muffles or better Gerstenhöfer furnaces for pulverized matte. Where the sulphur is allowed to escape, choice will generally lie between roasting lump matte in stalls and pulverizing it and treating in open reverberatories or Gerstenhöfers. At Mansfeld (see Table IX, below) 12 to 14 per cent. of sulphur from a matte containing 25 to 29 per cent. was recovered as sulphuric acid by the use of Gerstenhöfer furnaces, yet this course has been abandoned there, and the matte is now, or was lately, roasted in stalls in lump form, and this although the cost of pulverizing was greatly diminished by granulating the matte in water as it ran from the furnace. The experience at Bergen Point, however, seems to be that it is more economical to pulverize the matte with Blake or Sturtevant pulverizers, roast it in reverberatories, and grout it than to roast in stalls. Vivian, as late as December, 1880, speaks very favorably of the Gerstenhöfer, which I believe is still extensively used in Swansea for matte burning.

The roasting of matte is conducted in much the same way as that of ore, whatever be the mode adopted, except that a higher temperature can be used, and that much more fuel is required, since little available heat is developed by the combustion of the comparatively small amount of sulphur and iron in the matte.

The employment of coal or coke dust for the expulsion of arsenic and antimony in matte roasting as explained in considering ore roasting, is especially recommended.

The most marked difference between the roasting of ores and mattes is that in the latter no sulphur can be volatilized as such (unless pyritous ores be added). This deprives us of a potent means of expelling arsenic and antimony as sulphides. The economic features of the roasting of mattes as performed in stalls and heaps in several places are given in the accompanying table (Table IX), and several examples of matte roasting in open reverberatories may be found in Table IV.

Table IX.—*Matte roasting in stalls, open piles, and Gerstenhöfer furnaces.*

	Pt. Shirley, Mass., coarse metal, in stalls.	Agordo, coarse metal, in stalls.	Imaginary white metal, in stalls.	Ore Knob, N. C., coarse metal, in piles.	Mansfeld, matte, Gerstenhöfer.	Swansea, coarse metal, 1880, Gerstenhöfer.
	1.	2.	3.	4.	5.	6.
Inside dimensions			5′ by 6′ by 4′8″	14′ by 14′ by 5′		
With or without grate	With	Without	With			
With or without front wall	With	Without				
Weight of matte per charge, tons of 2,000 pounds.	6. 6	11	5	32		
Number of burnings	2	5	3	6		
Days occupied for total number of burnings.	5		12	30		
Output per furnace per twenty-four hours, tons.	1. 5 A		0. 5		10 to 15	

Table IX.—*Matte roasting in stalls, open piles, and Gerstenhöfer furnaces*—Continued.

	Pt. Shirley, Mass., coarse metal, in stalls.	Agordo, coarse metal, in stalls.	Imaginary, white metal, in stalls.	Ore Knob, N. C., coarse metal, in piles.	Mansfeld, matte, Gerstenhöfer.	Swansea, coarse metal, 1880, Gerstenhöfer.
	1.	2.	3.	4.	5.	6.
Composition of material treated:						
Sulphur	32.5	....	....	22.08	25 to 29	23
Iron	31.0	....	....	47.00	17 to 24?	......
Copper	35.5	25	72?	23.41	35 to 51?	33
Sulphur in roasted matte or ore.	....	....	....	3.38	....	12
Product smelted in	Cupola	....	Cupola	Cupola	Reverberatory.	......
Yields matte carrying:						
Iron	....	8 to 12	....	....	....	......
Copper	[2]65	[3]62	....	[3]55	65 to 66	......
Product of smelting consists of:						
Proportion of black copper.	Very large	71.5	Large	54	....	......
Proportion of matte	Very small	28.5	Small	46	....	......
Fuel, kind and quantity used per ton matte roasted.	Wood, 0.08 tons.	Wood, 0.195 tons; charcoal, 0.085 tons.	Wood, 0.08 cords.	Wood, 0.23 cords; charcoal, 15 bushels.	....	......
Labor, days per ton roasted	0.22	0.28	....	0.56	....	......
Authority	R., p. 285	R., p. 344	P.	E.	K.	V.

A. = Approximately.
E. = T. Egleston, Transactions of the American Institute of Mining Engineers, X, p. 32.
P. = E. D. Peters, jr., in Mineral Resources of the United States, 1882, p. 293.
R. = Rivot, Principes Généraux du Traitement des Minerais Métalliques, 1871, I.
K. = Kerl, Grundriss der Metallhüttenkunde, 1881, p. 175.
V. = Vivian, Copper Smelting, etc., p. 21.

4. SMELTING FOR WHITE METAL.

THE AIM.

In smelting the roasted coarse metal (the first matte) for the rich white metal of the "ordinary" process rather than for the comparatively lean blue metal of the "extra" process, we do not so much aim at the expulsion of arsenic and antimony as at bringing our copper forward to the metallic state rapidly and cheaply. To effect this most advantageously we endeavor to completely scorify the iron, to remove all sulphur in excess of that needed to combine with the copper present as disulphide of copper (Cu_2S), and at the same time to reduce all oxidized copper chiefly by means of the sulphide of iron undecomposed in the preceding operation. Practically this end cannot be accurately attained, and in fact we are compelled to leave some 4 to 8 per cent. of iron in the resulting matte, while the slag contains from 3 to 5 per cent. of copper.

We aim at a product approximating the composition of disulphide of copper with, say, 71 to 75 per cent. copper. If it were much leaner in copper it would be difficult to bring it forward to the condition of blister copper in the succeeding operation. It should not be much richer than

75 per cent. copper, since any excess of copper above that required to combine with the available sulphur present (part of the sulphur is appropriated by the iron and other metals present) to form disulphide of copper (Cu_2S) would precipitate out in the metallic state, carrying with it most of the arsenic, antimony, and gold, creating thus an additional product whose treatment complicates the process, and a considerable part of which works down into the sand hearth, causing a heavy locking up of very impure copper. That part of the metallic copper thus precipitated which is not absorbed by the hearth will, on tapping the charge, accumulate in the bottoms of the molds nearest the furnace, through which the matte passes on its way to those farther off, the molds communicating with each other by slight depressions in the tops of the sand partitions which separate them. It is hence called *copper bottoms.* If it be desired to thus eliminate the arsenic, antimony, and gold in bottoms it is in general better to do so after performing the additional fusion of the extra process, since this additional fusion itself expels a large amount of arsenic and antimony, so that both the resulting matte and the bottoms accompanying it are freer from these noxious elements. It is sometimes, however, considered desirable to separate bottoms in the first fusion of the roasted "coarse metal," eliminating arsenic and antimony at less cost, if less completely.

The silica for the scorification of the oxides of iron formed in roasting the coarse metal is furnished by the sand taken up mechanically by the coarse metal from the molds into which it is tapped in operation (2); by the fettling and bottom of the furnace; and very advantageously by the addition of silicious ores nearly free from iron sulphides, preferably containing the copper in an oxidized state, or of ore-furnace slags.

From the following examples it will be seen that the resulting slag is essentially a silicate of protoxide of iron, far more basic and ferruginous than the ore-furnace slag, and hence so heavy as to necessarily retain shots of matte. While this does little harm, as the slag generally carries so much scorified copper that it has to be resmelted at any rate, it should not, on the one hand, be so ferruginous as to corrode the furnace too rapidly, yet it should be sufficiently ferruginous to subsequently act as a desirable flux for the usually acid ore-furnace charge, with which it is generally resmelted.

	Welsh white metal slag.[a]	Kaafjord white metal slag.[b]
Silica	33.8	31.0
Protoxide of iron	56.0	56.0
Dioxide of copper	0.9	
Alumina	1.5	6.9
Magnesia	0.3	0.6
Lime	1.4	3.5
Miscellaneous oxides	2.1	0.3
Matte	4.0	1.67
Total copper	3.7	

[a] Le Play op. cit., p. 254.
[b] Kerl, Grundriss der Metallhüttenkunde, I, p. 214.

The furnace in which this operation is performed, called a "metal" furnace, is almost identical with that employed in ore smelting. There are generally, however, two air ports, horizontal passages, about 7 inches square and nearly parallel with the longer axis of the furnace, extending from the back of the furnace to the laboratory, and thus admitting a large volume of cold air into the laboratory beneath the flames. This cold and therefore heavy air passes over the melted or melting charge underneath the flames, so that when these air ports are open the charge is exposed to a strongly oxidizing atmosphere, the flame heating it by radiation but not coming in contact with it, and, therefore, exerting no reducing action on it. It appears better to the writer to heat the air thus admitted by passing it through the walls or bottom or roof of the furnace, since it thus intercepts and returns to the furnace heat which would otherwise be lost, and since by using hot air instead of cold a greater quantity of air can be admitted into the laboratory without undesirably cooling the furnace, and more strongly oxidizing conditions can thus be obtained.

The metal furnaces are not provided with wells for granulating the matte. The principal dimensions of some metal furnaces are given in Table VI.

THE OPERATION.[1]

The operation (see second column of Table VI) is performed very much as the ore smelting is. If part of the mixture is in powder and part in lumps, the lumps should be charged first and the fines placed on top of them. This arrangement places in the hottest position the fines, which are, from their structure, the least pervious to the heat, and the lumps, the easiest to melt, are most protected from the heat. As the fine matte melts it can readily trickle down past the lumps to the bottom, reacting on them, absorbing their silica in passing, and leaving them protruding and exposed to the heat. If the fines were at the bottom they would lie there unacted on, since when the lumps charged above were melted they would simply lie as a liquid layer above the fines, protecting them from the heat. Again, by placing the lumps below, when the upper portion of the charge melts and falls to the bottom it will arrange itself in the interstices between the lumps, in level layers, matte below and slag above, the hearth being thus protected by the matte from the corrosive action of the basic slag. But if fines were charged direct on the hearth the iron oxides in which they are so rich would lie for a long while in actual contact with the hearth under conditions most favorable for corroding it, satisfying their affinity for silica by dissolving it from the hearth instead of by combining with the silica contained in the charge.

The amount of sulphur given off in this operation is so much less than in ore smelting that there is much less ebullition. About five

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[1] Le Play, op. cit., pp. 243 and 254.

hours after charging, the bath is rabbled to remove any matter adhering to the bottom. After another half hour the slag is skimmed, the matte tapped, the fettling repaired, and a new charge introduced, much the same as in ore smelting. The whole operation generally occupies about six hours, though the length of this as well as of the other reverberatory smelting operations varies considerably.

CHEMICAL REACTIONS.

At the end of some three hours after charging the furnace we may distinguish three fairly well defined horizontal layers: below, the matte; next, a well fused slag; and above all, a layer of semi-fused slag, containing the original elements of the charge, quartz, metallic oxides, and sulphides, exposed to the most intense heat and gradually melting and passing to the lower layers. But these layers when first formed differ greatly from the final products, the matte at first being much leaner and the slag much richer in copper than at the end of the operation. Le Play found the following proportions of copper in the matte and slag at different stages:

Hours after charging.	Copper in matte.	Copper in slag.
	Per cent.	*Per cent.*
Three hours ten minutes....	54	9
Four hours	52	8
Five hours.....................	65	5
Five hours fifty minutes	74	3.5

As the fusion takes place in a strongly oxidizing atmosphere we should expect that the matte would grow richer in copper, owing to the expulsion of its sulphur as sulphurous acid and the scorification of its iron, and that the slag would at the same time grow richer in copper by the oxidation and scorification of that metal. But in point of fact the slag grows leaner in copper as the operation progresses; that is to say in the latter part of this strongly oxidizing fusion there is an actual deoxidation of the copper scorified in the beginning of the operation. It is evident that this reduction of copper from the slag is effected mainly by the sulphide of iron present, as explained in considering the ore smelting. That this reduction is effected mainly by sulphide of iron and not by sulphide of copper is indicated by the fact that so little ebullition occurs during the operation. If the oxide of copper were reduced by sulphide of copper we should have, first, a lively ebullition of sulphurous acid, and, second, the formation of a considerable amount of metallic copper, $2\ Cu_2OSiO_2 + Cu_2S = 6\ Cu + 2\ SiO_2 + SO_2$.

But as little ebullition takes place and as metallic copper is not separated, we must conclude that the reduction takes place thus: $FeS + Cu_2OSiO_2 = FeOSiO_2 + Cu_2S$.

The reaction of the matte and slag is promoted by the constant renewal of their surfaces of contact after fusion; first, by the subsidence of the richer and heavier matte from the plane of contact of matte and slag as fast as it becomes enriched, portions still comparatively lean in copper taking its place from below; second, by the sinking, from the uppermost and still incompletely fused pasty layer of slag, of drops of matte as they are released by progress of the fusion, not only reacting on the slag as they pass through it, but by their passage stirring it up. The mutual reaction of the matte and slag is further promoted by the great extent of the surface of their contact, due to the fact that they are spread out in two very thin layers extending across the whole area of the hearth.

4 *a*. SMELTING ROASTED COARSE METAL FOR BLUE METAL.

Since in smelting the roasted coarse metal for blue instead of for white metal we seek to produce a matte with less copper and consequently more sulphur and iron than white metal has, the oxygen in the metallic oxides in the charge must be in smaller proportion to the sulphide of iron present than in the operation last discussed, so that the sulphur and iron may not be oxidized to so great an extent as they there are. This should incline us to add sulphureted rather than oxidized fluxes. Since the extra process of which this operation is a part aims especially at the expulsion of arsenic and antimony, particularly pure ores are generally selected to furnish the silica needed for fluxing the oxides of iron, and the addition of the necessarily impure slags from the succeeding operations is avoided.

THE FURNACES (SEE TABLE VI).

For the same reason special furnaces are used, which, though identical in construction with those employed for the other fusions, avoid contaminating the products with impure materials left in the furnace from previous charges. Indeed, it is a general rule in the reverberatory process to avoid, as far as possible, performing any of the fusion operations of the extra process in furnaces in which impure materials are treated. The impurities absorbed by the hearth are liable to come up and contaminate the pure materials when least expected, even weeks and months after their absorption. So, also, special furnaces should, if possible, be reserved for each of the fusion operations, of both the "ordinary" and the "extra" process, lest low-grade material absorbed by the hearth should mix with the high-grade product resulting from a more advanced operation performed in a furnace in which a less advanced operation had previously been carried on. So, again, when the extra process is used, not to obtain exceptionally pure copper from very pure materials, but for producing perhaps merely a barely merchantable copper from impure materials, special furnaces should be employed, lest the impurities given off in great quantities and absorbed by the hearth in treating impure materials should later on contaminate the products of pure materials.

This great differentiation in the functions of the various furnaces renders the reverberatory process more applicable to very large establishments, where the many classes of furnaces, each reserved for a special work, and each with its special workmen highly expert in their special operation, may be kept fully occupied.

THE OPERATION, ETC.

With the exceptions stated above, the details of the operation, its aims and chemical reactions, are the same as in operation (4) smelting for white metal, already described, to which the reader is referred, p. 58. (See, also, the third and eighth columns of Table VI.)

4 *b*. ROASTING-SMELTING BLUE METAL FOR REGULE AND BOTTOMS.

THE AIM.

The aim of this operation is to scorify the small amount of iron which remains in the blue metal as completely as possible and to bring the percentage of sulphur slightly below that required to form disulphide of copper (Cu_2S) with the copper present; this causes the precipitation of metallic copper (bottoms) which has already been referred to.

The reason of this precipitation is that we cannot have a sulphide of copper, at least in the molten state, containing less sulphur than disulphide of copper has. Consequently, if, as in this operation, we expose molten disulphide of copper to the air and oxidize a portion of its sulphur, a corresponding quantity of copper at once separates in the metallic state, so that the remaining matte is still of the composition of disulphide of copper. The copper thus separated sinks to the bottom of the furnace and thus separates from the matte by gravity. The important feature of this is that nearly all the arsenic, antimony, and gold (on account of their having a weaker affinity for sulphur than copper has), will separate out from the matte with this first portion of copper which is thus separated; so that this affords us a ready way of removing arsenic, antimony, and gold from the bulk of the matte and concentrating these metals in a small quantity of metallic copper which can be specially treated.

The oxidation of the iron and sulphur, and incidentally of the copper is effected by very slowly melting down the large blocks of blue metal in a strongly oxidizing atmosphere.

THE OPERATION.

The operation (see Table VI, column 4) is performed in furnaces similar to the ore-smelting furnaces, but provided with air ports. As the material is much denser far heavier charges can be made than in ore smelting. The blue metal, cast in very large cakes and with much sand adhering to it, is piled on the hearth, nearly filling the laboratory and reaching almost to the roof. Care is taken to leave large interstices through which the flames and air can draw, both among the cakes of matte and along the sides and roof of the laboratory. The air ports being left wide open, the temperature rises very gradually and the matte melts very slowly, the current of air oxidizing the drops of matte as they trickle down. This is kept up till the cakes have been melted down to a pasty mass, when the air ports are closed, increasing the draft, and the temperature is raised till the fusion is complete. If the oxides of the charge have been in proper proportion to the sulphides, and if the operation has been properly timed, a small amount of metallic copper will now have been formed. The workman learns whether this has occurred or not by dexterously exposing the bottom of the furnace by suddenly pushing aside the bath of molten matte with his rabble, when, if any metallic copper has been formed, it may be recognized as it lies on the bottom of the furnace by its beautiful sea-green color and metallic luster. If none is found, the oxidizing action is prolonged by keeping the doors open and allowing the air to strike on the molten matte, its surface being kept free from slag, the oxidation of the sulphur which is thus effected eventually causing the precipitation of metallic copper. As soon as this has occurred the charge is tapped out of the furnace into a row of sand molds. The metallic copper which has been formed will be found in the bottoms of the moulds nearest the furnace; hence the name "bottoms." Above it, and filling the rest of the molds will be found the very rich matte known as regule.

CHEMICAL REACTIONS.

During the slow fusion arsenic and antimony appear to be partly driven off as arsenious acid and trioxide of antimony (As_2O_3 and Sb_2O_3); the gentle oxidizing action to which each particle is exposed as, by the gradual melting away of the outer portions, it approaches the surface of the lump which contains it, favors the formation of these imperfectly oxidized compounds, the tendency in any one layer to form them rather than the non-volatile completely oxidized combinations (the arseniates and antimoniates) being probably at its strongest just before that layer becomes fully exposed to the strong oxidizing action of the atmosphere of the furnace and while that action is still weakened by the interposition of a thin film of partly oxidized matte.

The surfaces of these cakes of matte, exposed as they are to the strongly oxidizing atmosphere of the furnace at this high temperature, apparently contain a considerable amount of oxide of iron and of copper. Now as the matte melts drop by drop, and as the drops trickle down one by one over these oxidized surfaces, each drop is exposed to the oxidizing action not only of the atmosphere of the furnace, but also of the pasty oxides over which it trickles; and these conditions, this intimate contact of drops of molten matte with semi-fused oxides, appear to effect the oxidation of the sulphur and iron of these mattes, containing as they do but little of these elements, much more energetically than the ordinary methods of roasting do, such as roasting in heaps, kilns, or reverberatory furnaces without fusion; and yet there can be no doubt that the oxidation of the sulphur and iron of mattes and ores rich in sulphur and iron, but lean in copper, is performed much more economically by the ordinary methods of roasting without fusion than by this roasting-smelting of the Welsh process. Just where the line should be drawn between mattes which are most economically roasted without fusion and those for which a roasting-smelting is most desirable it is hard to say, but it probably lies somewhere about blue metal containing say 57 per cent. of copper. At Bergen Point it is considered advisable in treating blue metal to roast part of it, which is then charged with large blocks of raw blue metal and subjected to a brief roasting fusion. At Swansea, I believe that nothing as rich as blue metal is roasted. At the time of the last information I have from Baltimore no roasting proper whatever was done, mattes even much leaner in copper than blue metal being oxidized partly by the Welsh roasting-smelting and partly by the use of the scanty supply of oxidized ores. The same course was formerly followed at Phœnixville, where the later introduction of reverberatory roasting furnaces effected a great saving, and few metallurgists will probably to-day advise relying solely on the Welsh roasting smelting where compounds containing much below 50 per cent. of copper are to be treated, especially if the ratio of iron to sulphur in them be large.

That portion of the copper which is oxidized during the roasting-smelting is again reduced for the most part when the mass unites on the hearth, partly by the sulphide of iron which has escaped oxidation, with formation of disulphide of copper and oxide or silicate of iron, but also in part by the disulphide of copper with the evolution of sulphurous acid and formation of metallic copper. The enrichment and purification of the matte is thus effected: First, by the direct oxidation of its impurities during the fusion; second, by the oxidizing action of the oxides and silicates of copper formed in the fusion and later reduced at the expense of the iron and sulphur; and third, by the elimination of the foreign metals and of arsenic and antimony in bottoms.

The products of the operation are (1) a slag which may be returned to the ore smelting, though it is generally so impure that it is better to treat it with impure materials for an inferior grade of copper. Though far more basic than the ore furnace slag, the sand adhering to the blue metal and the silica greedily absorbed from the hearth and fettling by the oxides of iron formed in the roasting generally make it rather more acid than a singulo-silicate; (2) a generally spongy matte called "regule," with more or less moss copper; (3) copper bottoms, generally very impure, and reserved till a sufficient amount has accumulated to make a special refining charge for copper of inferior quality. In treating auriferous materials most of the gold concentrates in these bottoms, which are specially treated for that metal.

Le Play gives the following analyses:

	Bottoms.	Regule.		Slag.
Copper	92.5	81.1	Silica	34.
Iron	Fe, Ni, Mn, 1.6	0 2	Alumina	1.
Sulphur	4.8	18.5	Lime	1.
Arsenic	.4	....	Oxide and sulphide of copper	12.
Tin	.2	....	Protoxide of iron	52.
	99.5	99.8		100.

5. ROASTING-SMELTING THE RAW WHITE METAL AND REGULE FOR BLISTERED COPPER.

THE AIM.

The aim of this operation is to expel as sulphurous acid the sulphur which up to this point has been retained to effect the reduction of the oxides and silicates of copper inevitably formed while oxidizing the iron. The conditions at the same time favor the expulsion of arsenic and antimony, and the scorification of the heavy metals (Sn, Ni, Co, Mn, Fe, etc.) present in small amount.

THE OPERATION.

The operation (see Table VI, columns 6, 9, and 25) is performed in furnaces similar to the smelting furnaces already described, and provided with air ports. The present tendency is to greatly increase the size of the "roasters," as these furnaces are called, and one has lately been constructed with a capacity of 35,000 to 50,000 pounds at each charge. Both labor and fuel are greatly economized by increasing the size of these furnaces. The labor, after the furnace is charged, consists almost wholly in overseeing the operation and managing the fire, the work of skimming the slag being short. As far as the supervision goes the furnacemen can nearly or quite as efficiently oversee 10 tons as an ounce. And since the exterior surface of the furnace is augmented by increments of its size in a much smaller ratio than its cubic contents are, the radiating surface and hence the loss of heat by radiation are smaller per unit of contents in large than in small furnaces. For very large furnaces several doors may be needed to enable the workmen to stow the large pigs of matte advantageously. These are charged by means of strong heavy peels, and piled almost to the roof, care being taken to leave interstices through which the flame and air can readily pass. On first charging, the air ports are closed for some ten minutes to raise the temperature of the furnace to the point at which the oxidation can take place. They are then opened and the temperature is gradually raised till the matte begins to melt and trickle down drop by drop, each particle as it melts and runs down being thus exposed in turn to the oxidizing action of the metallic oxides on the surfaces of the cakes of matte over which it runs and of the air which pours in through the air ports, the hot and light products of combustion passing in a thin layer close to the roof and heating the charge almost wholly by radiation. The first drops begin to run some thirty minutes after the furnace has been charged, and this gradual fusion is continued till, after from five to six hours, the whole has united on the hearth as a semi-liquid mass.

During this period the oxides of copper formed by the direct action of the air on the disulphide of copper react on adjoining particles of disulphide of copper with evolution of sulphurous acid and formation of metallic copper ($Cu_2S + 2\ Cu_2O = 6\ Cu + SO_2$), which, from its greater specific gravity, collects on the hearth beneath the remainder of the charge, and thus at once is itself protected from further oxidation and separates the abundantly formed oxides of copper from the hearth, whose corrosion is thus prevented, while the copper oxides are restrained from passing into the condition of silicates, from which the subsequent reduction of the copper would be very difficult. As in the roasting-smelting of blue metal the mildly oxidizing conditions presented to the different particles as, by the fusion and removal of the exterior portions, they in turn acquire a position immediately beneath the surfaces of their several cakes, should strongly favor the expulsion of arsenic and antimony as arsenious acid and trioxide of antimony, the oxidizing tendencies passing, by insensible steps, through every possible degree of strength as the thickness of the exterior shell of matte gradually decreases, till it finally vanishes, and we have the direct action of the current of air untempered. While a portion of the arsenic and antimony is expelled in this way, other portions become completely oxidized and form the non-volatile arseniates and antimoniates. During this oxidizing period a considerable portion of whatever zinc, tin, nickel, cobalt, and other heavy metals there may be present is oxidized, to be scorified and removed in the crasses in the following period. When this semi-fusion or sloughing off has progressed so far that but little of the still unmelted portion of the charge protrudes above the surface of the pasty mass gathered on the hearth, so that the oxidizing effect of the current of air is greatly lessened, the first period is brought to a close, the air ports are closed, the temperature is raised, and the whole charge thoroughly melted. During this (the second) period the reaction of the disulphide of copper, Cu_2S, on the previously formed copper oxides continues, with reduction of both sulphide and oxide to metallic copper and expulsion of sulphurous acid. The arsenic and the antimony of the previously formed arseniates and antimoniates are apparently also deoxidized by the disulphide of copper as well as by the escaping sulphurous acid, portions of them being completely deoxidized and uniting with the copper and the remaining sulphur, other portions being probably only partially deoxidized and brought to the condition of arsenious acid and trioxide of antimony, which are volatilized. The iron, which has been nearly all oxidized, rises to the surface as fusion progresses, and when all has become liquefied the iron crasses, very rich in copper and quite impure, are skimmed off. If the charge contains much iron its scorification may be promoted by charging a small quantity of very pure silicious oxidized ores at the end of the first period (say 1,200 pounds of ore of 27 per cent. copper to a charge of 5½ tons of matte). The third period now begins, in which the charge is gradually cooled down under the strong oxidizing action of currents of air pouring in through the air ports, which are again opened. The upper surface soon becomes so viscid that the escape of the sulphurous acid from beneath throws it up into little craters, thus enormously increasing the amount of surface exposed to the air and affording passages through which the air can descend to oxidize the sulphur and copper beneath. This is kept up till the whole mass is quite hard, except perhaps a small portion next to the bridge, unless the careful examination of the quantity of matte remaining undecomposed shows the furnaceman earlier that the oxidation has progressed far enough.

When the mass has become so hard that the reactions between dinoxide and disulphide of copper no longer take place, the third period is terminated and the temperature is again raised, the oxidizing action being continued by leaving the air ports open except for short intervals when they are closed to increase the draft of the fire. The furnaceman must now attentively watch the progress of operations and close the ports when the oxidation and removal of the sulphur have progressed so far that all the copper will have passed to the metallic state by the time the fusion has been completed. When all is liquefied the crasses are skimmed and the copper is tapped out into large sand molds. If the operations have been properly timed its surface will be covered with large smooth blisters; hence the name blistered copper.

In Chili, where the metallic copper produced is in general intended for the European market, and where consequently great care is bestowed on the appearance of the bars, a smooth surface being above all things desired, a small quantity of 50 per cent. matte, say 400 pounds to a charge of 5½ tons, is charged a short while before tapping. After most of the copper has run from the furnace this matte begins to run out with it, and, being much lighter, forms a thin coating on the top of each mold, protecting the copper in setting from the action of the air, which but for this would tend to cause little craters or pimples from the evolution of sulphurous acid. When the bars thus cast are quite cold this matte is chipped off with great care. The aim is to have the bars very smooth, from the belief that if they are rough a considerable amount of dirt may be caught on their rough surfaces, and that when the bars are sampled on their arrival in Europe by their purchasers an undue proportion of dirt will get into the samples.

For the same reason great care is bestowed on the preparation of the sand molds, the equivalent of six hours' labor being occupied in preparing them. Before tapping they are heated by burning wood on them.

The details of this roasting-smelting process differ greatly in different places. In some places the second period (the complete fusion after the slow oxidizing semi-fusion and the skimming of the crasses) is wholly omitted, the cooling down and setting commencing as soon as the first period of semi-fusion is complete. This was formerly the practice at Swansea. In others, as at Bergen Point and elsewhere in this country (and formerly at Phœnixville), the setting is wholly dispensed with, the charge being slowly melted down and then exposed, as it lies fully melted, to the action of the air admitted through the air ports and the side door, the crasses being frequently skimmed, until the whole is converted into metallic copper. As long as any sulphur remains the melted mass bubbles with a peculiar hissing noise from the evolution of sulphurous acid. When this ebullition ceases it is well to flap the metal for a short while to form a little dinoxide of copper, which dissolves in the metallic copper and completes the expulsion of the sulphur.

The following are examples of the time occupied in the several periods of this roasting-smelting process:

Chilian and Swansea practice.	Time.
Chilian practice (*private notes*).	
1. Slow semi-fusion:	
Charge 5 6 tons white metal of 70 per cent. copper:	
Charging finished at	0^{h} 0^{m}
Ports closed till	0 30
Roast with open ports till	5 23
2. Fusion completed at higher temperature:	
Charge 1,200 pounds carbonates of 27 per cent. copper:	
Close ports at	5 23
Fire hard to melt till	11 0
Skim slag from 11 hours till	12 0
3. Gradual setting:	
Open ports, cool down, and solidify from 12 hours till	19 0
4. Remelt and tap:	
Temperature raised till	22 47
Skim crasses at	22 47
Tap at	23 10
Charging:	
After fettling commence charging at	23 35
Finish charging at	24 0
Swansea practice (*after Le Play*).	
1. Slow semi-fusion:	
Charge finished at	0 0
Ports closed till	0 10
Roast with open ports till	5 30
2. Gradual setting:	
Open ports and cool down from 5 hrs. and 30 min. till	11 0
3. Remelt and tap:	
Slow semi-fusion from 11 hours till	17 0
Fusion completed from 17 hours till tap at	23 0
Charging:	
Commence charging at	23 0
Finish charging at	24 0

The products of this operation are, first, crude metallic copper, from which nearly all the impurities have been eliminated and which becomes merchantable after one more operation (refining); second, slags and crasses of varying degrees of acidity, according to the amount of iron contained in the matte treated, to the care which has been taken to prevent the scorification of the copper, and especially to the extent to which the oxidation has been carried in the final slow fusion of the charge after setting it. If mattes of high grade and consequently very free from iron, such as regule and pimpled metal, have been treated, the slags will be essentially silicates of copper. As they are practically equivalent to rich oxidized copper ores, usually so hard to obtain, it is very convenient to use them in operation (4) smelting for white metal, as their use enables us to obtain a product of a given grade in that operation, with a less thorough preceding roasting. As, however, they are generally very impure, it is often preferred to treat them separately for copper of an inferior grade[1], rather than to return to the regular process impurities which have been eliminated at such cost, and which, if thus returned, tend to accumulate, and have at any rate to be separated from the copper again and again.

The following are examples of the composition of the slag and copper produced by this roasting-smelting for blister copper:

TABLE X.

	Welsh "roaster" slag.[a]	"Roaster" slags from Kaafiord.[b]		Welsh blistered copper.	Blistered copper, from Kaafiord.[b]
Silica	47.5	36.0	Copper	....	99.2–99.4
Protoxide of iron	28.0	7.0	Iron	0.7–0.8	0.1– 0.2
Alumina	3.0	6.0	Nickel and cobalt	0.3–0.9	0.2– 0.3
Dinoxide of copper, Cu_2O	16.9	43.2	Zinc	....	0.0– 0.02
Lime	....	2.7	Tin	0.0–0.7	....
Magnesia	....	0.8	Arsenic	0.4–1.8	....
Nickel and cobalt oxides	0.9	4.9	Sulphur	0.1–6.9	0.1– 0.12
Oxide of tin	0.3	0.6			
Oxide of zinc	2.0	3.2			

[a] Le Play, op. cit., p. 318.

[b] Kerl, Grundriss der Metallhüttenkunde, I, p. 215.

CLASSIFICATION OF ORES AND SLAGS.

ORES.

Having considered the various operations of the reverberatory process, we can more intelligently consider the reasons for the classification of ores adopted and for assigning the several classes of ores and the slags produced in the several fusions to the operations in which they are respectively treated.

Class I.—In the ores of Class I, which is treated in the first roasting, may be included:

1. Impure sulphureted ores rich or poor in copper, because to eliminate their impurities as fully as possible they should undergo as many operations as possible. They should not be added to any of the later

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[1] Vivian (op. cit., p. 28) strongly recommends this course.

operations, because we must not contaminate with impurities the furnace products which have already had their impurities at least partly removed in the earlier operations.

2. Impure oxidized ores, rich or poor, if they contain impurities (arsenic-antimony) in a condition in which they may be eliminated as sulphides by being roasted with pyritous ores in the first operation.

3. Sulphureted ores, pure or impure, rich or poor, containing much sulphureted iron. Of course, they must be roasted before smelting, that their iron may be scorified. If very rich in sulphureted iron they will yield even after roasting much sulphide of iron on smelting, so that, if charged in the second instead of the first smelting operation, they would contaminate with sulphide of iron the product of the roasted matte charged with them, which has already undergone three operations for the express purpose of removing that compound.

Class II.—In Class II, treated in the ore smelting without previous roasting, may be included:

4. Impure oxidized ores, except those containing impurities (arsenic and antimony), which would be removed by roasting with pyrites. They are treated in the first rather than in a subsequent smelting, lest their impurities should contaminate products which have already been at least partly purified in the early operations, and that their impurities may be expelled as fully as possible by undergoing the full number of operations.

5. Low-grade oxidized ores, both pure and impure (except those coming under subclasses 2 and 7) should be passed first through the regular ore-smelting rather than through one of the later operations, since they yield a large proportion of slag, and since it is only in this operation that slags clean enough to be rejected can in general be produced. If smelted in a later fusion they would greatly and uselessly augment the quantity of slag to be resmelted.

Class III.—In Class III, which is treated in the second smelting (operation 4, smelting for white metal), may be included:

6. Rich, pure ores, both oxidized and sulphureted, if not containing much sulphureted iron. If oxidized their use enables us to obtain in this smelting a given product with less thoroughness, and hence less expense, in the immediately preceding calcination. If sulphureted, the proportion of sulphureted iron to copper is so small that they may practically be regarded as rich mattes mixed with gangue, from which they can be advantageously separated in this smelting, while if treated in the ore smelting (operation 2) we should contaminate their copper with sulphide of iron, which would have to be subsequently removed at great and useless expense.

7. Lean pure ores, oxidized or sulphureted, which do not contain any considerable quantity of sulphureted iron, but with a strongly silicious gangue, which forms an excellent flux for the oxide of iron whose scori-

fication and removal is the aim of the smelting for white metal. These ores, if of extreme purity and available in larger quantity than that required for operation (4) smelting for white metal, may be also advantageously used in (4 *a*) smelting for blue metal, and in (4 *b*) roasting-smelting for regule and bottoms; and even as in Chili, for operation (5) roasting-smelting for blister, if the matte treated in that operation contains iron. The preceding classification is summed up in the following table:

TABLE XI.—*Classification of ores.*

Purity.	Sulphureted or oxidized.	Richness in copper.	Miscellaneous.	Class.	Sub-class.	Operation in which treated.
Impure..	Sulphureted...	Rich or poor.	....	I.	1 and 3.	Ore roasting.
	Oxidized........	Rich or poor.	If impurities are removable by roasting with pyrites.	I.	2.	Ore roasting.
			If impurities are not removable by roasting with pyrites.	II.	4 and 5.	Ore smelting.
Pure......	Sulphureted...	Rich...	If containing much iron sulphide.	I.	3.	Ore roasting.
			If not containing much iron sulphide.	III.	6.	Smelting for white metal.
		Poor...	If containing much iron sulphide.	I.	3.	Ore roasting.
			If not containing much iron sulphide, but silicious.	III.	7.	Smelting for white metal.
	Oxidized.......	Rich...	....	III.	6.	
		Poor...	If silicious	III.	7.	
			If not silicious............	II.	5.	Ore smelting.

This classification does not, of course, aim to be complete; it is merely intended to illustrate the principles which guide one in assigning different varieties of ores to different operations.

SLAGS.

If the ore-furnace mixture be acid those portions of the resulting acid slags which have to be resmelted on account of containing shots of matte form a desirable flux for the iron oxides scorified in the second fusion (smelting for white and for blue metal). These acid ore-furnace slags are generally so free from injurious impurities (arsenic and antimony, etc.) that they will not contaminate the advanced products of the second fusion.

The more basic slags of the later fusions are generally more impure than the ore furnace slags. Owing to the opportunity the iron oxides have for absorbing silica from the fettling these slags are rarely basic enough to be of great advantage in the ore smelting for fluxing the silica of the ore; and owing to their impurity it is generally desirable to treat them separately for an inferior grade of copper, purified perhaps by the separation of bottoms, rather than to return them with their impurities to the regular course of operations. "Roaster" slag, however, from the roasting-smelting for blister copper, being very rich indeed in silicate of copper may, if not too impure, be advantageously used in the second smelting operation (4), its oxidized copper being very efficient in scorifying the sulphide of iron which we aim to eliminate in this operation.

The resmelting of slags, formerly carried out in reverberatories, is now almost universally done in cupolas, except in so far as they are used as fluxes in the various reverberatory operations.

REVERBERATORY METHOD FOR OXIDIZED ORES AND NATIVE COPPER.

In the comparatively rare cases in which oxidized ores are treated in the reverberatory wholly without sulphurets, the reduction of the copper has to be effected by the addition of some reducing matter, such as coal. In treating native copper we have to choose between (1) reducing in the operation itself the oxides of copper, which are necessarily largely formed during the inevitably oxidizing fusion, with coal or some similar reducing substance, and (2) subsequently recovering the scorified copper oxides from the slags by a separate reducing fusion or by treatment with iron bars (Rivot and Phillips).

REDUCING-SMELTING WITH CARBON.

The operation, whether for oxidized ores or for native copper, is performed in very much the same way as in smelting sulphureted ores, the fluxes needed to form with the gangue a desirable slag, together with the amount of coal or charcoal needed for the reduction, being mixed with the ore or native copper before charging. The furnaces employed are very similar to the ore-smelting reverberatories already described, and should have a deep fire-box, so that the atmosphere of the furnace shall be as little oxidizing as possible. As a reducing agent, bituminous coal is much better than charcoal, since the hydrocarbon gases copiously evolved by the former efficiently stir up the melting mass, constantly exposing fresh portions of it to the reducing action of the coal, and themselves offering great reducing surface as they bubble through the overlying slag. When fusion commences, we have, besides the gangue, the oxides and silicates of iron and copper, together with considerable metallic iron and copper reduced by contact with the coal and by the passage of the reducing gases. As fusion progresses the oxides and silicates of copper are reduced as they trickle down over the coal, and by the already reduced iron. Even after fusion is complete, the coal swimming in the slag will reduce scorified copper. Since iron is rapidly reduced at temperatures far below that of incipient fusion, much of this metal is inevitably brought to the metallic state unless the amount of coal charged is so small that much copper escapes reduction.

When melted, the charge is rabbled, skimmed, and tapped, just as in the case of sulphureted ores. When, however, rich native copper concentrates are treated, the quantity of metallic copper produced in each fusion is so great that it may be advantageously refined, after skimming the slag, without removing it from the furnace in which it is smelted.

The products will be a slag and either refined or crude metallic copper. The slag can probably be rarely brought below a tenor of 1 per cent. copper without reducing so much iron as to make the refining very difficult.

SMELTING WITHOUT CARBON.[1]

This method is employed at Lake Superior for treating the mass copper and (native copper) concentrates, which are so rich in copper that the metal is refined in the smelting furnace itself after fusion is completed at the same operation in which it is smelted. The details of the fusion are the same as when carbon is used for reducing. The slags are rich in copper and are resmelted in cupolas.

COMPARISON OF THE TWO METHODS.

By smelting with reducing agents we obtain clean slags, but, in case of ferruginous materials, a copper alloyed with so much iron that its refining is costly. By smelting without reducing agents we obtain pure and easily refined copper, but a slag so rich that it has to be resmelted. Evidently the former course is best suited to lean, the latter to rich ores and native copper. With lean ores, yielding much slag and little copper, it is more important to obtain a slag clean enough to be thrown away, even if this implies obtaining an impure and difficultly refined copper, than to obtain pure and readily refined copper together with slags so rich in copper that they have to be resmelted. On the other hand the plan of smelting without reducing agents is especially suited to rich ores, since with them the advantage of obtaining the large yield of copper in a condition in which it is easily refined greatly outweighs the disadvantage of having to resmelt the small quantity of slag produced.

SHAFT FURNACE PROCESS FOR SULPHURETED ORES.

GENERAL PLAN.

The general plan of operations, resembling that of the reverberatory process in its main features, is as follows:

1. Roasting the ore.
2. Smelting the roasted ore in cupolas, obtaining a clean slag, which is thrown away, and a matte *A*.
3. Roasting the matte *A*.

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[1] See columns 17, 18, 19, and 20 of Table VI.

4. Smelting the roasted matte *A* in smaller cupolas for black copper, obtaining —

(*B*) A slag, usually returned to the ore smelting, but sometimes clean enough to be thrown away.

(*C*) A rich matte, which is roasted and then resmelted in (4) again (either by itself or with the matte *A*).

(*D*) Black copper.

5. Refining the black copper in reverberatories.

In many if not most cases, the second matte *C*, after being roasted, is resmelted together with the first matte *A*, so that the products obtained from *C* blend with those of *A*. In other cases the second matte *C* is smelted by itself, when we in general obtain still a third matte *D* with black copper.

On resmelting the roasted matte *D* we may again obtain both black copper and another matte *E;* on reroasting and resmelting *E*, we may obtain again more black copper and a matte *F*, and so on, the proportion of matte to black copper obtained depending on the richness of the matte and the thoroughness of the roasting. Thus different portions of the copper contained in one and the same original charge of ore will reach the condition of ingot copper by means of greatly differing numbers of operations. Some, precipitating as black copper when the matte *A* is smelted, will undergo in all but the five operations above given; others will remain with the constantly decreasing quantity of matte till this has been reroasted and resmelted even five or six times, bringing the total number of operations up to thirteen, fifteen, or even more. The quality of the ingot copper produced from these different portions of one initial charge of ore will vary much. The arsenic, antimony, gold, and, to a smaller extent, silver will concentrate in the portions of black copper first produced, while that produced last, from matte which has been repeatedly roasted and smelted, will be much purer, first, because of the elimination of impurities in the first portions of black copper, and secondly, from their further expulsion in the repeated roastings and smeltings. By limiting the thoroughness of the several roastings, and by the addition of pure unroasted sulphureted ores or furnace products, we increase the proportion of matte to black copper which will be produced in the several smeltings and hence the proportion of pure to impure ingot copper finally produced, the average number of operations which the copper, taken as a whole, undergoes, and hence the total quantity of arsenic, antimony, etc., expelled, and at the same time the expense of the process.

By increasing the thoroughness of the roastings and by the addition of oxidized ores and products we lower the average number of operations, the total expense of the process, the purity of the copper taken as a whole, and the proportion of pure to impure copper obtained. Though some black copper is generally produced in the first smelting of the roasted matte *A*, by limiting the thoroughness of the first two roastings, by the addition of pyritous ores, etc., we may defer producing black copper till the third or even the fourth smelting, and thereby increase the total expulsion of arsenic and antimony. Since those portions of these elements which are precipitated out with the black copper can be less perfectly removed than if they had remained in matte, the greater the number of operations before the precipitation of the first black copper the purer will it be. It is much easier to expel arsenic and antimony from a matte than from metallic copper.

The black copper produced in the several different operations may be refined separately for ingot copper of varying grades of purity, or, as is generally the case in this country, it may be mixed together. The same is true of the different grades of the rich matte produced with the black copper.

The black copper frequently contains so much iron that it is desirable to divide the refining into two distinct operations.

1. ORE ROASTING.

The roasting is performed exactly as in the reverberatory process, except that, where the expulsion of arsenic and antimony is not especially aimed at and we simply strive to reach the condition of ingot copper as rapidly as possible, the roasting may be pushed to an extent which, in reverberatory smelting, would entail serious loss of copper both mechanically and from scorification, since in cupola smelting the tendency to scorification is very much less and the mechanical loss in the slag considerably less than in the reverberatory.

If the ore is roasted in reverberatory furnaces and if the retention of sulphur is aimed at (to favor the expulsion of arsenic and antimony), we should avoid raising the temperature very high towards the end of the roasting, as a high temperature decomposes the sulphates of iron and copper with the expulsion of their sulphuric acid; while if undecomposed in the roasting, they would in the following smelting be largely reduced to sulphides. If the retention of sulphur is not desired, our aim being simply to reach the condition of ingot copper as rapidly as possible, the temperature should be raised at the end of the roasting to decompose the sulphates. This is in shaft-furnace treatment a more important matter than in the reverberatory, since in the latter, with its comparatively violently oxidizing conditions, the reduction of sulphates to sulphides is hardly to be looked for.

2. ORE SMELTING.

IN GENERAL.

The roasted ore, mixed with appropriate fluxes, charged in more or less intimate contact with the fuel and traversed during the whole of its gradual descent by the inevitably rather strongly reducing gases, undergoes a reducing action from the time it enters the furnace till it is completely melted before the tuyeres. And even after melting, the reducing action is very strong, since the melted slag trickles down over the solid fuel, which reaches down to the bottom of the furnace. In addition to these powerful reducing factors, which we do not have in the reverberatory, we have the deoxidizing tendencies of the reverberatory as well, the melted slag being traversed by the matte, which gradually falls through it, energetically deoxidizing any scorified copper; and we have the same suspension and solution in the slag of that powerful reducing agent, sulphide of iron. Not only does the shaft furnace thus offer much more energetic deoxidizing tendencies than the reverberatory, but its sulphidizing tendencies are also much stronger than those of the reverberatory. The contact of the ore with the solid fuel and the reducing gases in the shaft furnace protects the sulphur from the oxides of the charge and from the free oxygen of the atmosphere of the furnace, and thus preserves it for union with the metals. It moreover reduces the sulphates of copper, iron, lime, and baryta to sulphides, the sulphides of baryta and lime playing a rôle similar to that of sulphide of iron, though weaker, in deoxidizing and sulphidizing scorified copper. The strength of the reducing and sulphidizing tendencies should be such that no copper should be scorified, and yet that no iron should be precipitated in the metallic state, enough sulphur being present to absorb any iron which is deoxidized. Where the expulsion of arsenic and antimony is not aimed at, the reducing action should be simply strong enough to prevent the scorification of copper, the deoxidation and sulphidizing of iron being avoided as far as possible, so that we may obtain a matte as rich as possible in copper.

In zinc-bearing ores the zinc, reduced to the metallic state in the lower part of the furnace and volatilized, is again oxidized higher up by the carbonic and sulphurous acids, attaching itself to the walls in hard crusts called cadmies.

Many examples of furnace construction and output, fuel and labor requirements, etc., in ore smelting are given in Table XII. The figures in columns 6, 10, 11, 14, 15, and 16 fairly represent the best American practice.

TABLE XII.--*Examples*

	Mansfeld, pile-burnt schists.	Oker.	Panulcillo, Chili, very fusible ores.	Oker, burnt matte.	Hancock, Mich., slags, 1881.	Bisbee, Ariz., oxidized ores, 1885.	Laurel Hill, N. Y., kiln-burnt pyrites, 1884.
	1.	2.	3.	4.	5.	6.	7.
Dimensions, &c., of furnace:							
Shape of furnace at bottom.	Round	Round	Round		Oval	Round	Round
Crucible	Internal	(*a*)			Internal	Internal	External.
Water jacketed or not	Not				Partly	Yes	Yes
Height from bottom:							
(1) To slag notch	2′ 9″				1′ 10″ (?)	1′ 10″	0″
(2) To tuyeres	3′ 4″				2′ 6″	2′ 6″	2′
(3) To throat	26′ 3″				10′ 4″	9′ 0″	10′
(4) To top of charge							
Inside dimensions:							
(1) At bottom	6′ 2″ diam.				7′ by 4′ 9″		4′ diam
(2) At tuyeres						3′ 6″ diam.	4′ 4″ diam
(3) At throat	7′ 2½″ diam					4′ 6″ diam.	5′ 6″ diam
Area of cross-section at tuyeres, per tuyere, in square feet.	4′ 9″					1. 17′	1. 47′
Tuyeres, size	2″					4″ and 5″	2″
Tuyeres, number	6	4	8		(*b*)	6	10
Blast pressure, ounces per square inch.	32. 4	7. 72			11. 2	8	18
Blast temperature	300° Cent.		Cold		Cold	Cold	Cold
Total burden smelted per twenty-four hours, 2,000 pound tons.	132 to 165.	11 to 11. 6.	50. 7	8. 26	25 (?) *c*	50 *d*	90 (?) *e*
Outlay of labor and fuel:							
Ratio of burden to fuel	6	5	7. 5 to 9	3. 5	3. 5 (?)	6. 6	7. 5
Kind of fuel	Coke				Anth	Coke	Gas coke.
Days of labor per 2,000 pounds burden smelted.					0. 407 (?)	0. 40 *k*	0. 2 (?)
Men employed at furnace per twenty-four hours.						20. *k*	18
Composition of slag:							
Silica		25			37	28	31. 12
Alumina					10	27	1. 60
Protoxide of iron		46			24	29	51. 84
Lime					27	9	2. 96
Magnesia							ZnO 3. 45
Copper	0. 6 to 0. 7	0. 3 to 0. 5	0. 6 to 0. 8		0. 5	1. 32	0. 55
Specific gravity					3. 25	3. 8	
Composition of product:							
Per cent. copper	36				Say 98	Say 96	40 to 50
Per cent. iron							29
Per cent. sulphur							24
Authority	K., p. 168.	K., p. 168.	K., p. 168.	K., p. 177.	P. N	W	H

a Partly internal, partly external (Sumpfofen).
b An open slit extending completely around the furnace.
c Runs only 12 hours at a time.
d Including slag and fluxes.
e This figure, given by the superintendent, seems almost incredible.
f Excluding slag and fluxes.
g Including fluxes but not slag.
h 54.3 bushels per ton of burden.
i Connellsville coke.
j Snowshoe coke.
k Includes engineer, blacksmith, etc.
l This slag is highly ferruginous, and usually a singulo-silicate or even somewhat more basic.
m This slag, while highly ferruginous, contains usually a considerable amount of lime, and is more basic than a singulo-silicate.
n For every 54 parts of black copper, 46 parts of matte are produced, containing 55 per cent. copper.
o Charge, 66 per cent. roasted and 17 per cent. raw matte (each containing 40 to 50 per cent. copper), with 17 per cent. slag; product, two-thirds black copper, one-third white metal.

of cupola smelling.

Ore Knob, N. C., pile-burnt ore, 1881.	Ore Knob, N. C., pile-burnt matte, 1881.	South Strafford, Vt., pile-burnt pyrrhotite, 1884.	Arizona (locality withheld), oxidized ores, 1884.	Oker, 1877.	Accessa, 1879.	Milan, N. H., 1884, kiln-burnt pyrite.	Bergen Point, N. J., kiln burnt cupreous pyrite, 1885.	Bergen Point, N. J., roasted and grouted matte yielding black copper, 1885.
8.	9.	10.	11.	12.	13.	14.	15.	16.
Square	Square	Round	Round	Round	Round	Round	Rectangular.	Rectangular.
Internal	Internal	Internal	Internal			External	External	External.
Not	Not	Yes	Yes			Yes	Not	Not.
		9″	1′ 2″			0″	0′ 0″	0′ 0″.
1′ 0″		1′ 9″	2′ 1″	2′ 2½″	2′ 0½″	1′ and 1′ 6″	0′ 9″	0′ 9″.
		9′ 6″	10′ 4″	11′ 3″	10′ 11″	9′	5′ 6″	5′ 6″.
		7′ 0″	10′ 4″			7′	4′ 6″	4′ 6″.
3′ 10″ sq	3′ 10″ sq	3′ 8″ diam.		2′ 11″ diam	4′ 3″ diam.	3′ 6″ diam	4′ by 11′	4′ by 11′.
2′ 4″ sq	2′ 4″ sq	3′ 9″ diam.	3′ 1″ diam.			3′ 8″ diam.	4′ by 11′	4′ by 11′.
		4′ 0″ diam.	4′ 6″ diam	3′ 11″ diam	4′ 11″ diam		6′ by 13′	6′ by 13′.
2.71′	2.71′ (?)	1.84′	1.2′	1.64′	2.27′		5.5′	5.5′.
2¼″	3″ (?)	2¾″	3″			2½″ and 2″	6″	6″.
2	2 (?)	6	6	4	6	10	8	8.
	6 (?)	7.25	8.8	7.72		16 to 24	12	12.
Cold	Cold	Cold	Cold			Cold	Cold	Cold.
13 *f*	13.5 *f*	44.5 *d*	37 *g*			30 to 45	120	90 *d*.
2.0 *f h*	2 *f h*	9.3 *d*	7	5		7.06	7	5.5.
Charcoal	Charcoal	Coke *i*	Coke			Gas coke	Coke *j*	Coke *j*.
0.69	0.59	0.29 *d k*	0.56 *g*			0.214	0.20	0.267.
		13	21 *k*			8	24	24.
27.56		32.25	29					
4.29			12					
52.49		46.79	35				(*l*)	(*m*)
9.59			13.6					
MgO 1.11			2.7					
0.57		0.42	2.0			0.2 to 0.5	0.34 to 0.88	0.75 to 2.66
		3.54	1					
23.41	94.24 } *n*	{ 22 to 33	Say 96			29	40 to 50	(*o*)
47.00	3.38							
22.08	0.74							
E	E	G	J. H	C	C	B	P. N	P. N.

B. = F. L. Bartlett, superintendent.
C. = Capacci, Revue Universelle, 1881, X, p. 315.
E. = Egleston, Transactions of the American Institute of Mining Engineers, X, p. 34.
G. = W. Glenn, superintendent.
H. = J. B. Herreschoff, superintendent.
J. H. = S. James, jr., and Carl Henrich, superintendent.
K. = Keil, Grundriss der Metallhüttenkunde.
P. N. = Private notes.
W. = Ben Williams, superintendent.

NOTE.—The information in columns 15 and 16, received while this bulletin is in press, represents the very latest practice at the Orford Company's works, one of the most progressive in the country. In some points it does not agree accurately with certain remarks on shaft furnace practice made elsewhere in this paper, which was written more than a year earlier.

THE DEOXIDIZING EFFECT.

The deoxidizing effect of shaft-furnace smelting depends, first, on the length of exposure to the deoxidizing action of the furnace; second, on the strength of that deoxidizing action. The former is directly proportional to the cubic capacity of the furnace and inversely to the rapidity of smelting. The latter is increased by increasing—

1. The average temperature of the ore, since deoxidation takes place much faster at high than at low temperatures.
2. The proportion of fuel to burden.
3. The porosity of the fuel, since porous fuel reduces carbonic acid to carbonic oxide faster than dense fuel does.
4. The porosity of the ore, porous ore being more readily penetrated by the carbonic oxide.
5. The intimacy of mixture of fuel and burden.
6. The basicity of the mixture, since the more completely the silica is saturated with bases the less resistance does its affinity offer to the reduction of such weak bases as the oxides of copper and iron.
7. The infusibility of the mixture. This increases both the strength and the length of the deoxidizing action; the former because a larger proportion of the deoxidizing agent, the fuel, is required to raise a comparatively infusible mixture to its higher melting point; the latter because a longer time also is required to raise it to that point, so that the smelting proceeds more slowly.
8. The preponderance of oxide of iron among the bases, since this base, readily reduced at very low temperatures, seems to carry oxygen from the copper oxides to the carbon and carbonic oxide.

The deoxidizing action is weakened by increasing the quantity of blast, whether by increasing its pressure or the number or size of the tuyeres within certain limits; this hastens the rate of smelting, and if carried beyond a certain point increases the proportion of air to fuel entering the furnace per minute, and hence the ratio of the free oxygen and carbonic acid to the carbon and carbonic oxide to which the ore is exposed.

The deoxidizing action is probably also indirectly weakened by raising the temperature of the blast,[1] which, by increasing the supply of sensible heat, enables us to smelt the ore with a smaller proportion of the deoxidizing agent, the fuel. Of course if the proportion of fuel is not diminished, heating the blast may merely strengthen the deoxidizing action by raising the average temperature. Thus in certain cases it is said to have increased the tendency to form salamanders by the reduction of iron.

Intimacy of mixture of ore and fuel is promoted by distributing ore and fuel in uniform layers over the surface of the charge. It is diminished by charging the ore in one part of the furnace and the fuel in

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[1] Kerl, Grundriss der allgemeinen Hüttenkunde, 1879, p. 239.

another, so that they descend as two somewhat separate columns. This is frequently effected by charging the fuel in the middle of the furnace and the burden against the sides. It weakens the deoxidizing action, directly by diminishing the extent of contact between fuel and ore and indirectly by diminishing that between the ore and the reducing gases, since the latter pass in greater proportion through the comparatively open column of fuel than through the more compact column of ore.

The deoxidizing tendencies are stronger with internal and partly internal crucibles than when the crucible is wholly external, since in the former the slag, as long as it remains in the furnace, is in intimate contact with the column of fuel, which extends through it and rests on the bottom of the furnace.

9. If we have two furnaces of equal cubic capacities, so that the ore remains for the same length of time in each of them, one of them being narrow and high, and the other broad and low, it is probable that the latter will on the whole exert a more powerful deoxidizing influence than the former, because it will probably be necessary to use a larger proportion of the deoxidizing agent, the fuel, in the latter, in order to attain a given temperature in the hearth. This is because in the low broad furnace the gases have a shorter distance to travel from the tuyeres to the top of the charge and consequently arrive there hotter. Escaping from the top of the furnace hotter they carry away with them more sensible heat, and consequently a larger proportion of fuel has to be burnt to generate the temperature required in the crucible of the furnace. Moreover, as the gases in this case reach the top of the furnace so hot, there is a greater tendency for the carbonic acid which they contain to be reduced to carbonic oxide[1] by the surrounding fuel than in the comparatively cool-mouthed higher furnace. This reduction of carbonic acid to carbonic oxide of course absorbs a great amount of heat; to compensate for this we have to use more fuel in order to attain the temperature needed in the crucible of the furnace, and an increase in the proportion of fuel means an increase in the strength of the deoxidizing action.

It is possible that in the low, broad furnace the average temperature of the ore may be higher than in the narrow high furnace, for two reasons: First, as the gases escape hotter from the top of the low furnace they may more quickly heat the descending ore to the temperature at which deoxidation commences; secondly, in the low broad furnace it may well be that there is a larger region which is intensely heated by direct radiation from the region of active combustion at the tuyeres than in the high narrow furnace. This, too, would tend to raise the average temperature of the ore, and of course the higher the average temperature of the ore the greater will be the average strength of the reducing action. This is, however, a mere conjecture.

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[1] Transactions of the American Institute of Mining Engineers, III, p. 247.

Both these considerations point to the low furnace as a more powerful deoxidizer than the high one of equal cubic capacity, supposing the ore and fuel to be mixed with equal intimacy in both. But we can more completely separate the ore and fuel in the low furnace than in the high one, by charging them in separate columns, and we may thus offset the stronger deoxidizing tendencies of the low furnace largely and perhaps completely. Hence, while the low broad furnace is probably a stronger deoxidizer than the high narrow one of the same cubic capacity when burden and fuel are charged in horizontal layers, the reverse may be true when fuel and ore are charged in separate columns.

10. Sulphides in general, and pyrites especially, play a most important double part in the smelting operation, warding off the troubles which are caused both by too strong and by too weak deoxidizing action.

If the deoxidizing action be too weak, the tendency is, of course, for oxidized copper to become actually scorified; this is checked by the powerful affinity of the sulphur of the sulphides for copper.

If, on the other hand, the deoxidizing action be too strong, the tendency is for iron to pass to the metallic state and form salamanders; this, too, is checked by the powerful affinity of the sulphur for iron, the iron combining with the sulphur to form a matte instead of separating out in the metallic state.

The large amount of heat sometimes developed by the combustion of the sulphur and iron of the pyrites at the tuyeres in certain cases enables us to diminish the quantity of the fuel employed and thus to weaken the deoxidizing action.

Copper actually scorified may be deoxidized thus:

$$FeS + Cu_2O, SiO_2 = FeO, SiO_2 + Cu_2S$$
$$Cu_2S + 2\ Cu_2O, SiO_2 = 6\ Cu + 2\ SiO_2 + SO_2$$

ARSENIC AND ANTIMONY.

Arsenic and antimony may be expelled—

(1) By volatilizing as arsenious acid and trioxide of antimony (As_2O_3 and Sb_2O_3).

(2) By volatilizing as sulphides.

(3) By concentrating them in a speise.[1]

(4) By producing a small quantity of black copper, in which the greater part of these elements will be found concentrated.

They probably exist in the calcined ore largely, if not mainly, as arseniates and antimoniates. In the shaft furnace these compounds are in part reduced to arsenious acid and trioxide of antimony, which volatilize, and in part completely deoxidized, entering the matte, or the metallic copper if any is formed. Arsenic and antimony are expelled more fully by a gentle and prolonged than by a powerful and brief reducing action. To volatilize and expel them as completely as possible,

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[1] Kerl, Grundriss der Metallhüttenkunde, 1881, p. 134.

the length of their exposure to the deoxidizing conditions should be increased, either by enlarging the furnace or by running it slowly; while, to weaken the net deoxidizing action, both so that the arsenic and antimony may be reduced only to the volatile condition of arsenious acid and trioxide of antimony and to prevent the increased length of the deoxidizing exposure from reducing an excessive amount of iron with the formation of salamanders, the deoxidizing tendencies should be weakened by the means already set forth, such as (1) using the minimum of fuel; (2) using dense compact fuel; (3) charging ore and fuel in separate columns; (4) running on slags low in iron and not too basic; (5) adding pyritous ores, which also favor the expulsion of arsenic and antimony as sulphides.

By an extremely thorough roasting of ores rather rich in copper and comparatively lean in sulphur, followed by a strongly reducing fusion, we may produce with the matte a small quantity of black copper which will contain most of the arsenic and antimony; this product may be so exceedingly impure that it should be separately treated for an inferior grade of copper.

In ore smelting the shaft furnace appears to the writer to favor the expulsion of arsenic and antimony more than the reverberatory furnace does. That portion of these metals which exists in the roasted ore as sulpharsenide or sulphantimonide of the other metals, is probably expelled to but a very slight extent in either furnace. That portion which exists in the condition of arseniate and of antimoniate, however, may probably be expelled to a considerable extent in the shaft furnace, where these compounds are reduced by the solid fuel and the reducing gases, with the formation of the volatile arsenious acid and trioxide of antimony at temperatures below the melting point of the charge, and the actual escape of these volatile compounds may therefore be looked for. In the reverberatory smelting, however, the arseniates and antimoniates are reduced, not by the fuel, but by being decomposed with liberation of arsenic and antimonic acids, by the silica of the charge, and by then being deoxidized by the sulphides of the charge. But as these reactions will only take place after fusion has set in, and as the deoxidation is effected by actual contact with the melted sulphides, for whose sulphur both arsenic and antimony have a strong affinity and which are present in so great excess, there appears to be comparatively little chance for the expulsion of arsenious acid and trioxide of antimony, the conditions powerfully favoring the complete deoxidation of the arsenic and antimony and their combination with the matte.

VOLUME OF FURNACE.

The greater the cubic capacity of the furnace the more completely does the descending column of burden and fuel abstract from the escaping gases their heat, and consequently the more economical of fuel is the furnace. At the same time the output of the furnace increases with its cubic capacity. We must not, however, make the cubic capacity of the furnace so great that the ore will remain in it so long that an excessive deoxidation of iron will occur with formation of salamanders or of undesirably ferruginous matte. Thus the desire to economize fuel and to increase our output impels us to enlarge our furnaces; fear of deoxidizing iron restrains us.

Could we increase the output of the furnace proportionally to its increase of volume, so as not to lengthen the sojourn of the ore in the furnace, we might perhaps increase its volume indefinitely without undue deoxidation of iron, but beyond a certain point it appears to be impracticable to cause the output to keep pace with increased volume, so that the fear of deoxidizing iron actually limits the size of the furnace. Thus for ordinary ores it is generally believed that furnaces should not be over 13 feet high nor over 4 feet 6 inches wide at the tuyeres. (See dimensions of cupola furnaces in Table No. 12.)

Anything which lessens the tendency to form salamanders through the reduction of iron to the metallic state enables us to enlarge our furnaces with corresponding economy of fuel. Among other things which diminish this tendency may be mentioned fusibility of mixture, large proportion of sulphur to iron oxide, charging fuel and burden in separate columns, and more especially freedom from iron oxides. Thus at Mansfeld, where the ores are nearly free from iron, furnaces even 22 feet 7 inches high are successfully used.

In copper smelting comparatively high narrow furnaces, somewhat widened at the top, are preferred to lower wide furnaces for the following reasons: In a high narrow furnace the fuel is better economized, and, as already stated, there is probably less tendency to reduce iron than in low broad furnaces of the same cubic capacity; so that as far as the formation of salamanders is concerned we can give a narrow furnace larger volume than we can a broad furnace. We thus get better economy of fuel in narrow high furnaces, both because we can give them greater volume without reducing iron, and because even for equal volumes they economize fuel better than low broad furnaces. The furnace may, however, be advantageously widened toward the top for the following reasons among others: A wide throat gives us a larger area over which we can scatter fine ore, and thus lessens the troubles due to its sifting down to the tuyeres. The gases escape with lower velocity from the top of the charge and thus carry off less fine ore and fuel. With zinciferous ores there is less danger of the throat becoming choked with cadmies[1] (Gruner[2] considered that cadmies deposit less easily in narrow than in wide throated furnaces, owing to the more rapid passage of the gases; but even admitting this, a small deposit of cadmie will cause more inconvenience in blocking up a narrow

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[1] Kerl, Grundriss der allgemeinen Metallhüttenkunde, 1879, p. 247.

[2] Gruner, Traité de Métallurgie, I, p. 250.

throat than a large one will in a wide throat). The wide throat gives us a chance to separate burden and fuel by charging them in columns.

RASCHETTE FURNACES.

As has been already stated, it has not been found practicable to increase the output of shaft furnaces proportionally to their increase of size, so that if, by enlarging the width or height of a furnace, we carry the cubic capacity beyond a certain point, the sojourn of the ore in the furnace becomes so long that an excessive deoxidation of iron occurs, with the formation of salamanders or of undesirably ferruginous matte.

By keeping the height and width of the furnace small, however, and increasing its cubic capacity by making it oval or oblong in horizontal cross-section, as in the Raschette furnace, the increase of output so nearly keeps pace with the increased cubic capacity that practically the tendency to reduce iron does not appear to be materially increased. Furnaces of this shape, with tuyeres in both of the long walls and in the ends, have been very successfully used for both lead and copper smelting in Europe and in this country. Those at Bergen Point, N. J., are 4 by 11 feet inside at the tuyeres, and smelt much greater quantities of burden than any other furnaces in the country. Their use is more general here than in Europe, where the round Pilz furnace is preferred. While the Raschette furnaces offer a certain advantage in saving labor (owing to the greater output per furnace), and in saving fuel, owing to the smaller proportion of heat radiated from their walls (the ratio of whose external surface to the volume of the furnace is much smaller than in a round furnace of a diameter equal to the width of the Raschette furnace), their main advantage is considered at Bergen Point to lie in the fact that they are less susceptible to serious derangements from irregular working. Where fine ore forms a large proportion of the burden it is liable to run down to the tuyeres irregularly and sometimes in large quantities. This in a round furnace may cause very serious derangement or even necessitate drawing the furnace out. In the Raschette furnace, however, should one end become deranged the furnace may be quickly brought into good order by carefully keeping the other end free from obstructions, and charging a little extra fuel, so as to raise the temperature there, and thus melt out the obstructions in the deranged end. Owing to this a much greater proportion of fine ore can be smelted in Raschette than in round furnaces. At Bergen Point as much as 75 to 85 per cent. of the total burden is sometimes fine ore, and frequently *the whole burden* consists of grouted fines.

The shape and size of the vertical transverse section of these furnaces are governed by the same considerations as in the case of round or square furnaces.

DISPOSITION OF CRUCIBLE.

With regard to the disposition of the crucible, we may divide shaft furnaces into those with—

1. Internal crucible (Tiegelöfen).
2. Crucible partly internal and partly external (Sumpföfen).
3. External crucible, if any (Spuröfen).

1. *Internal crucibles.*—With these the separation of matte from slag takes place inside the furnace, where a very high temperature prevails, and where, consequently, the liquidity of the slag is at its highest. The matte gradually accumulates and is tapped at intervals; the slag floating above it is removed continuously or at very short intervals. As the slag remains for some time within the crucible, where it is penetrated by the column of fuel, and by the matte, which falls into it as fast as it melts before the tuyeres, the conditions are very favorable for the reduction of any scorified copper, and for that of iron with formation of salamanders. At the same time the matte or black copper and slag are kept so hot that they separate better than when the separation takes place outside the furnace. On the other hand, accretions in the crucible are liable to form, since every time it is tapped fine pieces of coke fall down into the space thus left empty, tending to choke it up. Moreover, accretions which form can only be removed with difficulty, even though a temporarily closed aperture be left, which can be opened for removing them. Should they become excessive the furnace must be drawn out.

Hence these furnaces are especially desirable where the production of black copper, or of a viscid refractory slag requires that the separation of metal and slag shall take place at a high temperature, and they are to be avoided when the formation of accretions, such as iron salamanders, is to be expected (*e. g.*, in case of ferruginous ores). Thus furnaces of this class, such as are made by Fraser & Chalmers, and by Rankin & Brayton, are admirably adapted to most of the silicious oxidized ores of Arizona and New Mexico, which yield not only black copper but frequently very infusible slags, both of which would tend to chill if we attempted to separate them in external crucibles. On the other hand, for the ferruginous ores of the Eastern coast, with their very fusible slags and strong tendency to form salamanders, these furnaces are, in the writer's opinion, much less desirable than those with wholly external crucibles.

The slag is generally tapped from these furnaces intermittently, since if it flows continuously it not only cuts away the opening through which it escapes, but it is very difficult to adjust the size of the slag notch to that of the stream of slag closely enough to prevent the gas from blowing out of the furnace along with the slag. This may, however, be prevented by providing a water-cooled bronze cinder notch inclined upward, so that the column of slag flowing through it seals it and prevents the blast from blowing through it. When the slag flows in a continuous stream its velocity is much less than when it flows intermittently, and it consequently flows more quietly from the crucible, and thus deposits more fully any mechanically suspended shots of metal.

These furnaces with internal crucibles may advantageously be provided with external settling basins as well, to catch any shots of metal mechanically carried out with the slag, and they should be so arranged that, in case accretions form on the bottom, they can be worked temporarily as external-crucible furnaces, the melted matte and slag being allowed to run across the bottom and out of the furnace as fast as they melt, so as to cut down the accretions.

2. *Crucibles partly internal, partly external.*—In these furnaces the internal crucible is continued outside of the front wall of the furnace as a sort of fixed forehearth. This arrangement permits the easy introduction of tools to remove accretions in the lower part of the furnace, and pasty slag may be pulled out of the furnace. Here, as in furnaces with internal crucibles, the matte is tapped intermittently, the slag running constantly or at short intervals. These furnaces, now little used for copper smelting, are chiefly applicable where the slag is necessarily so refractory that the shots of metal would not settle out of it if it were cooled by running into an external crucible, and where at the same time there is a strong tendency to form accretions, which are very difficult to cope with in furnaces with wholly internal crucibles.

3. *External crucibles (fixed or moveable forehearths, steep boxes, settling basins, etc.).*—The bottom of the furnace is strongly inclined, and all slag and metal run out of it as soon as melted into an external basin in which their separation by gravity occurs. Except when black copper or a very refractory slag is produced this arrangement is the most convenient, as the formation of accretions is reduced to a minimum by the constant flow of the molten matter over the bottom itself, the tendency being in general not so much to deposit salamanders as to cut the bottom down. This in turn is prevented by keeping the level at which the molten matte issues from the furnace constant by means of a water cooled block (generally of iron, but better of bronze, since iron is readily cut by the stream of matte) set in the masonry, over which the matte flows. Where a large quantity of fusible matte is produced both matte and slag may be made to flow continuously from the external settling basin, which, to this end, is divided into two compartments by a transverse diaphragm extending down nearly to its bottom.[1] The matte and slag enter the compartment nearest the furnace. The matte accumulates at the bottom till its upper surface reaches the lower edge of the diaphragm, thus sealing it and preventing the lighter slag from passing it. As the matte accumulates it will rise on the out-

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[1] The employment of this plan by the writer at Capleton in 1879 is the earliest of which he is aware.

side of the diaphragm so that we will have on its outside a column of matte alone, on its inside a column below of matte and above of slag. The slag may be drawn continuously from the upper portion of the inside column, the matte from the outside column, though of course at a little lower level than the slag.

As these settling basins are liable to become choked by chilling, especially with refractory slags, they should be made readily removable, so that the working of the furnace need be interrupted but a few minutes when it becomes necessary to provide it with a fresh basin.[1]

If the slag contains scorified copper (chemically combined) it may be advantageously deoxidized, precipitated, and collected by placing a large number of wrought-iron bars (old rails, etc.) across the settling basin, so that the slag must pass over and between them, the copper being precipitated by the iron exactly as in the wet way.

To mitigate the disadvantage under which these furnaces suffer (the cooling of the melted matter before its separation takes place), the settling basins are sometimes roofed over, as at Chañaral, Chili, and a large aperture is left in the breast of the furnace, so that a quantity of gas is blown from the furnace into the settling basin above the settling slag. By making these basins very wide and providing against undue cooling the speed of travel of the matte and slag may be made so low that a very complete separation takes place.

WATER-JACKETED VS. BRICK WALLS.

On account of the difficulty of protecting brick walls against the corrosive action of the slag, water-jacketed walls are coming constantly into greater use. Brick walls may last for many months (or even, as at Mansfield, for years), first, when the slag is nearly free from the protoxides of iron and of manganese, since it is these bases which corrode the brick-work; second, where skill is attained in "sticking up" the walls, *i. e.*, in coating them while the furnace is running with a refractory ferruginous coating (probably consisting mainly of magnetic oxide of iron). This is effected by charging against the walls fine and thoroughly burnt pyrites or other form of oxide of iron. It is carried to great perfection at Bergen Point, where, with highly ferruginous and extremely corrosive slags, campaigns of many months are made. Probably the enormous volumes of cold air blown in at low pressure through very large tuyeres facilitate this, the blast, owing to its excess and low pressure, passing in large quantity along the walls and both cooling them and preventing the reduction of the magnetic oxide.

Brick walls, when thus protected, effect a certain saving in fuel. They may be thus kept so cool that the radiation from them will hardly exceed that from the outer wall of a water-jacket, while in the latter the circulating water abstracts a large amount of heat from the furnace,

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[1] See description by F. H. McDowell of a furnace built by J. B. Herreshoff, at Laurel Hill, N. Y., in Transactions of the American Institute of Mining Engineers, Vol. XIII.

which has to be met by burning more fuel. If the brick walls are allowed to get very thin, however, the loss of heat by transmission through them may become nearly equal to that carried off in the water of a water-jacket. The saving in fuel is offset to an extent which varies very greatly by the extra cost of repairs in brick furnaces, due to the corrosion of the brick-work by the slag.

The actual outlay for repairs and relining brick furnaces at Bergen Point, excluding repairs to the settling basins, is less than two bricks per ton of burden smelted, or, say, 5 cents. The extra fuel consumption due to abstraction of heat by the jacket-water is in two instances as follows:

Heat abstracted by jacket-water.

Furnace at which observations were made.	South Strafford, Vt.	Laurel Hill, N. Y.
Designation number in Table 12	10	7
Initial temperature of water	30° C.	15.5° C.
Final temperature	67° C.	77° C.
Gallons water per hour	3,228	2,000
Pounds of coke required per 24 hours to heat jacket-water, assuming that 25 per cent of calorific power of coke is utilized	1,322	1,328
Pounds coke for jacket-water under same assumption, per ton burden	30	14.7 (?)
Value of this coke per ton burden at $5 per ton coke	$0.075	0.039 (?)
Percentage of total coke consumption required for heating jacket-water	14	5.5

While these figures are not based on sufficiently extensive observations to allow us to generalize much from them, they certainly indicate that, even with the exceptional skill attained at Bergen Point, the cost of repairing the brick walls is nearly equal to the value of the coke consumed in heating the jacket-water at South Strafford, where the abstraction of heat in the jacket-water is not unusually low. But beyond the fuel consumption necessitated by the abstraction of heat in the jacket-water, water-jacketed furnaces in many cases require expensive repairs. In careless hands the jackets are liable to burn out from deposition of scale by the water, and they are apparently even in careful hands often eaten through by the matte.

While in central places, and extremely skillful and careful hands, the cost of repairing brick furnaces is probably considerably less than that of the combined cost of repairs and extra fuel consumption of water-jacketed furnaces, in remote places the reverse is probably true, as in them the cost of repairs weighs heavily against the brick furnace, because the freight charges on our two bricks consumed per ton of burden smelted are much heavier than those on the extra fuel needed by the water-jacketed furnace. Further, where the greatest care and skill are not exercised the consumption of bricks for repairing brick furnaces becomes vastly greater than two bricks per ton of burden; and, even in the most careful hands, where the slag is ferruginous, the burning out of the brickwork causes constant annoyance, frequent stoppages for repairs, and, by altering the internal shape of the furnace, serious derangement of its working.

All things considered, it is believed that, except under altogether unusual conditions, brick furnaces are destined to give way to water-jacketed ones. Three sets of conditions, however, suggest themselves, under which brick walls may be the most desirable: First, when the slag is nearly free from oxides of iron and of manganese, so that corrosion of the brick-work is not likely to occur; second, where, as in parts of Arizona, the water supply is too scanty to permit the use of water-jackets (here abundant blast should be provided and the mixtures should be as free as possible from iron); third, in extremely cold climates, where the freezing of the water connections may become a serious annoyance.

The lining of brick furnaces should be of the most basic bricks attainable. Bauxite, graphite, and the basic linings used for Bessemer converters have been suggested. The lining should be laid in headers, and it should not be bonded to the outer wall except above the zone of fusion, since this bonding implies the use of stretchers, which are liable to fall out and expose the adjoining courses when only a few inches of their width has been melted off, while headers are much more stable.

In furnaces with internal crucibles the water-jacket should not be carried down to the upper level of the bath of melted slag and matte, since there would be danger both of its chilling them and of the matte's corroding it. With impure water scale rapidly collects in the jacket and must be frequently removed to prevent the jacket from burning.

CAST VS. WROUGHT-IRON JACKETS.

Cast-iron jackets are much cheaper than wrought, and if injured the old casting is worth, if in the immediate neighborhood of a foundry, a considerable part of the cost of a new one. Cast-iron jackets are, however, more liable to accident, both from blows of the workmen's tools and from cracking owing to cooling strains in the castings, and they cannot be repaired as easily or as well as wrought-iron ones. Hence in the immediate neighborhood of a foundry cast jackets may be preferred; in remote places wrought-iron ones are more desirable.

To diminish the abstraction of heat by the jacket-water it should be hot when fed in and nearly boiling when drawn out; to utilize the heat which it abstracts it may be used to feed the boilers. If, as is generally the case, the jacket is closed at the top, the water connections must be so arranged that the water may be forced into the jacket under strong pressure, since if steam forms in the jacket considerable pressure is often required to force water into it.

Instead of the ordinary water-jacket, a simple iron or copper shell, cooled by sprays of water, has been successfully employed. This is a much cheaper construction, and any cracking or undue heating of the shell is more easily detected than in the inner walls of ordinary jackets. It requires, however, rather more vigilance to prevent overheating and probably a larger water supply than the double-walled jacket, and has met with little favor.

BLAST, TUYERES, ETC.

The width (and through it the output) of the furnace is in one sense limited by the penetrating power of the blast, which depends on its pressure, or, to speak more accurately, the velocity at which it enters the furnace, which, in turn, depends on its pressure, the size of the tuyeres, and the resistance offered to its passage by the solid materials in the furnace. It is evident that for a given volume of blast per tuyere the smaller the tuyere the greater will be the velocity at which the blast issues from it, and the farther consequently will its momentum carry it into the furnace. If, however, the diameter of the stream of air becomes very small it is more easily turned aside, so that to get the farthest penetration into the furnace for a given set of conditions our tuyeres must be so small as to give the blast high velocity, yet not so small that the stream of air is too easily diverted and that the friction in the tuyere becomes excessive. With the pressures ordinarily used the maximum penetration is probably attained with a tuyere diameter of from 2.75 to 3.5 inches. Very large tuyeres, with low velocity of blast and small penetration, may be desirable in the case of brick furnaces, since here we may wish a large quantity of air to ascend along the walls to prevent their corrosion.

The conditions will evidently be much more uniform across the furnace at the tuyere level if the blast is introduced at a large number of points than if it all enters through a single tuyere; the area of the horizontal cross-section of the furnace at the tuyere level, formerly in the old-fashioned single-tuyered furnaces as high as 5 square feet per tuyere, in modern poly-tuyered furnaces rarely exceeds 2.25 square feet, and in this country is generally from 1.2 to 2 square feet per tuyere. This spreading out of the blast is carried to its extreme limit in the Hancock cupola (No. 5, Table 12), where the blast enters through a slit extending completely around the furnace. We may surmise that the tendency at Hancock to form iron salamanders (which is said to be so strong that the furnace has to be drawn out every day, bringing the reported fuel consumption per ton of burden to nearly double that of any other well-managed American furnace within the writer's knowledge) may be due to the blast being so weakened by this extreme spreading out that it penetrates but a little way into the furnace, leaving a dead core in the center, a most potent cause of salamanders. There appears to be nothing in the composition of the charge at Hancock to cause precipitation of iron, unless it be the enormous proportion of fuel employed.

The employment of two rows of tuyeres, one above another, as in Ireland's foundry cupola and in the Milan copper furnace (No. 14, Table

XII), seems to have little to recommend it, since, instead of concentrating the heat, so that with the minimum consumption of fuel we may obtain the maximum temperature before the tuyeres, it scatters it.

With very refractory ores an upper row of tuyeres, ordinarily closed, but used in case accretions on the bottom of the furnace obstruct the lower tuyeres, might, however, be desirable.

The tuyeres should be so placed that the streams of air issuing from them will not interfere with each other. Thus in Raschette furnaces the tuyeres on the opposite sides should be placed staggering; in round furnaces they should not be diametrically opposite each other. They may each point a little to one side of the center of the tuyere horizon. The tuyeres often point downwards a little. This lowers the region of intense heat relatively to the tuyere level, enabling us to place the tuyeres a little farther above the bottom of the furnace without unduly lowering its temperature, and this in turn allows the accumulation of a greater quantity of molten matter in the furnace without interfering with the entrance of the blast. With internal-crucible furnaces this may be quite an advantage.

HOT BLAST.

Hot blast, heated in cast iron pipes in the upper part of the furnace and above the top of the charge, as well as in separate stoves, is successfully used in several places in Europe in smelting copper ores, and is said to have been lately introduced at an Arizona copper smelting establishment. It has been found to reduce the fuel consumption very considerably.

PRODUCTS.

1. *Matte.*—Since the slag is more perfectly fused in the shaft furnace than in the reverberatory, and since the tranquil settling in the crucible, whether internal or external, favors the complete separation from the slag of shots of matte more than does the skimming of the reverberatory, the matte may be richer in shaft furnace than in reverberatory smelting without causing an undue mechanical loss of copper in the slag. For this reason and because there is far less tendency in the shaft furnace to scorify copper, owing to its strong reducing action, we may aim at producing a much richer matte (by more thorough preliminary roasting, by adding oxidized ores or rich slags, etc.) than in reverberatory smelting. The writer can vouch for cases where, when producing very lean mattes with slags of favorable composition, the slag has contained only traces of copper.

TABLE XIII.—*Examples of composition of shaft-furnace matte.* (*See, also, Table XII.*)

	Mansfeld.	Oker.	Riechelsdorf.	Fahlun.	Stefanshütte from tetrahedrite, speise being formed with it.	Roraas.	Phönixhütte.	Oker.
	1.	2.	3.	4.	5.	6.	7.	8.
Copper	34.1-51.2	57.6-61.9	42.9-43.8	8.3-12.0	22.1	22.03	33.03	41.36
Iron	17.6-24.7	11.1-16.9	25.1-27.1	55.8-62.9	47.6	52.14	50.00	25.54
Zinc	2.4- 9.6	....	....	1.2- 2.9	....	....	....	4.82
Nickel and cobalt	1.0- 2.6	1.1- 2.1	0.6- 1.1	....	....	....	....	....
Lead...........	0.9- 2.5	0. - 1.4	0.9- 1.2	0.6- 3.9	....	....	....	3.87
Silver..........	0.1- 0.3	0.06	....	....	0.08	....	....	....
Manganese ...	....	....	0. -2.33	....	....	....	....	....
Antimony	....	....	....	....	3.5	....	1.04	....
Sulphur	25.1-29.8	23.9-24.3	26.6-28.3	24.6-27.2	25.8	25.15	25.49	21.76

The above is from Kerl, Grundriss der Metallhüttenkunde, 1881, I, p. 169.

2. *Slag.*—The composition of slags has been so fully discussed in considering reverberatory smelting that little remains to be added here. While the highly acid and aluminous slags which are permissible in the reverberatory could not be made to run in cupolas (and even those which can be smelted smelt so slowly and at so high a temperature that there is great danger of forming salamanders), we can in the cupola, especially if it be water-jacketed, run successfully on highly basic ferruginous and manganiferous slags which in the reverberatory would cause great annoyance and expense by their corrosiveness and by the difficulty of skimming them, on account of their fluidity, which in the cupola is rather an advantage. With these slags the use of external crucibles (Spuröfen) is desirable to prevent the formation of salamanders, and even then much care must be taken and the deoxidizing tendencies diminished as far as possible (by rapid running, excess of blast, charging fuel and burden in separate columns, low and narrow furnaces, etc.) to avoid forming them.

In zinciferous ores the sulphide of zinc remaining in the roasted ore and that formed in smelting by the deoxidation of oxide and sulphate of zinc in contact with the fuel and the metallic sulphides, tends strongly to enter the slag and to drag disulphide of copper with it; and this tendency appears to be stronger the richer the matte is in sulphur, perhaps because mattes rich in sulphur and consequently lean in copper are comparatively light and hence separate less readily from the slag.

The most desirable slag is probably between a singulo- and a bi-silicate, mainly of lime and iron oxide, the latter base predominating, and with a little alumina, but only a little. If too rich in iron oxide, the slag becomes so heavy that the matte does not separate readily. If there is not enough iron oxide the slag becomes refractory and a tendency to scorify copper arises. In smelting oxidized ores and native copper the presence of a considerable amount of oxide of iron is especially desirable to prevent scorification of copper. The oxygen ratio of base to silica may vary at least from 1 : 2.5, when lime predominates among the bases, to 1: 0.7, when oxide of iron predominates. The following are examples.

TABLE XIV.—*Composition of ore cupola slags.*[1] (*See, also, Table XII.*)

	1.	2.	3.	4.	5.	6.	7.	8.
Silica	47.91	34.67	36.39	50.00	45.20	54.13	57.43	56.5
Alumina	7.51	4.38	11.06	15.67	.....	10 53	7.83	6.0
Lime	1.11	3.53	4.63	20.29	6.28	19.41	23.40	6.3
Magnesia	0.35	....	0.54	4.37	2.40	1.79	0.87	14.3
Protoxide of iron	39.3	48.25	44.14	8.73	42.69	10.83	7.47	14.9
Peroxide of iron	1.22	....	....	Cu₂O 0.67	....	....	....	......
Pro oxide of manganese	0.38	2.00	....	....	....	....	....	......
Oxide of zinc	....	2.89	....	1.11	....	....	....	......
Oxide of lead	......	1.07	PbS 2.71	....	....	....	....	......
Zinc	0.49	....	Cu₂S 0.78	....	....	....	....	......
Copper	0.16	....	FeS 1.05	....	Cu 0.63	....	....	......
Oxide of copper	...	....	....	....	....	2.03	0.30	......
Sulphur	0.70	1.85	....	....	2.61	....	....	......
Oxygen ratio of base to acid	1:1.94	1:1.2	....	1:1.5	....	1:2.5	1:2.1	......

	9.	10.	11.	12.	13.	14.	15.	16.	17.
Silica	56.51	31.44	28.5–29.2	21.62	22.55	25.24	50.20	44.00	47.50
Alumina	9.42	7.86	8.9– 9.6	5.15	5.10	14.00	9.10	8.31	4.61
Lime	20.05	....	1.2– 3.0	2.57	2.95	4.86	19.72	6.17	13.72
Magnesia	6.40	4.46	5.1–11.6	....	1.33	ZnO 6.40	6.63	2.50	3.80
Protoxide of iron	6.21	55.21	51.0–52.8	65.62	38.31	45.96	29.18	35.43	26.68
Peroxide of iron	....	....	....	....	....	....	......	......	......
Protoxide of manganese	....	....	....	....	....	....	......	......	......
Oxide of zinc	....	....	....	....	....	....	......	......	......
Oxide of lead	....	....	....	....	....	....	......	......	......
Zinc	....	....	....	....	5.22	....	......	......	......
Copper	....	....	....	....	....	....	......	......	......
Oxide of copper	....	....	trace—0.4	2.26	....	Cu 0.3–0.5	......	0.52	0.16
Sulphur	....	....	....	2.08	....	....	......	......	......
Oxygen ratio of base to acid	....	1:1	....	1:0.7	....	....	......	......	......

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[1] Nos. 1 to 14 from Kerl, Grundriss der Metallhüttenkunde, 1881, p. 157; Nos. 15 to 17, inclusive, are made at Ahrn, Jochberg, and Brixlegg, respectively, all three being in the Tyrol. See Balling, Compendium der metallurgischen Chemie, 1882, p. 134.

Of these the 2d, 3d, and 5th are probably metallurgically the most desirable, while slags more acid than 7 and 9, or more basic than 12 and 13, will probably rarely be found advantageous.

The slags are generally so free from copper that they are rejected. Those portions carrying shots of matte may, if acid, be advantageously used to flux the ferruginous charge in the second smelting (operation 4). If basic, they are returned to the ore smelting.

DETAILS OF WORKING.

1. *Mechanical charging*, which effects an important saving in labor, has been introduced in Europe, but not yet in this country. The ordinary form of cup and cone charger is hardly applicable to copper smelting shaft furnaces, since it delivers the charge always in the same place, while in copper smelting it appears to be necessary to be able to place the charge in the middle or against the walls of the furnace at will, according to the working of the furnace. In many cases too, it is desir-

able to be able to charge the fuel in the middle of the furnace and the burden against the walls habitually.

This may be effected by using the double cup and cone, or by the common single cup and cone provided that the base of the cone is of the same diameter as the base of the cup, so that the cone can be lifted up above the cup. Then when the cone is lowered in the ordinary way the charge is thrown towards the walls of the furnace. But if the cone be raised above the cup the charge will slide directly from the surface of the cup towards the center of the furnace.

The practice of charging the fuel in the middle of the furnace, the burden against the periphery, is largely followed in this country, especially where there is danger of forming salamanders.

It is much to be regretted that the plan, widely adopted among American lead smelters with admirable results, of preparing the charge by systematically and uniformly mixing in horizontal layers great quantities of the ores and fluxes, whose composition has been previously accurately determined, in proportions designed to give a slag of predetermined composition, is not often followed among our copper smelters, who often charge flux and ore by the shovelful, or, if by weight, without precautions which insure uniformity of charge, and rarely with a view to producing slag of predetermined composition. Indeed, with a few notable exceptions, our copper smelting is in a much more crude condition than is the metallurgy of iron, of lead, and of the precious metals.

2. *Changing tuyeres.*—It is a very common practice, especially with refractory ores, to have an excessive number of tuyeres, not using them all at once, but changing them off. If a tuyere becomes dark and the slag appears to be chilling in front of it, it is closed for, say, thirty to sixty minutes. During this time the chilled slag will have melted, and when the tuyere is again opened it will be found bright. This plan gives admirable results.

3. *Filter charging.*—To prevent the fine ore from sintering around the lumps of coke, and thus hindering their combustion, Glenn[1] (who charges fuel and burden in uniform horizontal layers) recommends charging the lump ore on top of the coke, and then the fine ore on the lumps, the lump ore serving as a filter for the fines. Part at least of the resulting advantage may be due to its preventing the fine ore from trickling down to the tuyeres, and part of the apparent saving in fuel and increase in rate of smelting may arise from the fact that the escaping gases draw the fine ore out of the furnace to a greater extent on account of its being placed on the top of the charge, so that the quantity of ore actually smelted is less than that charged into the furnace.

4. *Fine ore.*—If a large proportion of the ore be fine, and especially if it be very valuable, from being rich in copper, or from carrying gold or

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[1] Engineering and Mining Journal, 1884, II, p. 34.

silver, the fine ore must be grouted, or we must provide large settling flues or dust chambers, to prevent excessive loss from the ore being mechanically dragged off by the gases which escape from the top of the furnace, as well as to prevent it from sifting down to the tuyeres irregularly and deranging the furnace, though this may sometimes be more economically prevented by charging lumps of slag along with the ore.

For grouting ores we may use either lime or clay, adopting whichever will form the most desirable slag with the other slag-making components of the charge. In grouting roasted matte or exceedingly ferruginous roasted ores clay has a great advantage over lime, since it lessens the quantity of silicious matter which is needed to flux the oxides of iron produced in the roasting, while the use of lime for grouting would necessitate increasing the quantity of silicious flux.

In rare cases tar or petroleum residues may be used for grouting. The grouting may be advantageously done in a pug-mill, the grout being dried by the waste heat of the smelting or roasting furnaces.

The grouted ore is sometimes bricked under strong pressure, and some fine ores may be bricked by very strong pressure without grouting.

3. ROASTING THE FIRST MATTE.

The aim and the details of working are substantially the same as in roasting the first matte (coarse metal) in the reverberatory smelting process, to which the reader is referred.

4. SMELTING THE ROASTED FIRST MATTE. (SEE TABLE XII.)

The conditions are much the same as when smelting very rich ferruginous ores. To make the weight of slag as small as possible, only enough silicious flux (oxidized ores, silicious slags, etc.) is added to form a slag of about a singulo-silicate or even a somewhat more basic composition with the iron oxides so largely produced in roasting the matte. If the roasting has been thorough, little matte will form, and our products will thus be mainly a ferruginous slag and impure black copper. On account of the excess of oxide of iron, the small amount of sulphur available for combining with any reduced iron, and the use of internal crucibles, the tendency to form salamanders by the reduction of iron is very great. To avoid this we must weaken the deoxidizing action as far as possible, especially by diminishing the ratio of the volume of the furnace to its cross-section at the tuyeres (by making it low and by carrying its walls up straight, or even drawing them together towards the throat), and by smelting very rapidly with excess of blast, so as to diminish the sojourn of the ore in the furnace; by adding only acid fluxes (basic ones such as limestone would weaken the hold of the silica on the protoxide of iron and facilitate the reduction of the latter); charging in columns; using the minimum of fuel, etc.

Internal-crucible furnaces are generally employed, on account of the tendency of the black copper to set in the comparatively cold external crucibles. They should have a large aperture just above the bottom, closed while running, but easily opened to remove accretions.

The reactions are much the same as in ore smelting, except that, as the proportion of sulphur is so much smaller, the deoxidation of oxidized copper takes place more by the action of carbon and carbonic oxide with formation of metallic copper and less by that of sulphur and sulphides with formation of matte.

Here we may seek the explanation of the fact that while in the reverberatory method we do not obtain metallic copper until the matte has reached the condition of regule (say 80 per cent. Cu), in the shaft furnace we obtain black copper together with matte carrying sometimes only 57 per cent. copper. Three causes suggest themselves:

(1) The deoxidizing conditions are so much stronger in the shaft furnace than in the reverberatory that in the former a much larger proportion of iron is deoxidized. As iron has a stronger affinity for sulphur than copper has it crowds the copper out of the matte, precipitating it in the metallic state ($Fe + Cu_2S = 2\,Cu + FeS$), and we get with the metallic copper a matte impoverished in copper by the substitution of iron for copper. It is to be observed that the sulphur of the matte produced along with metallic copper in the shaft furnace is really about as fully saturated with metals as in the reverberatory process, only in the shaft furnace this matte contains more iron and less copper than in the reverberatory.

(2) Owing to this same strong deoxidizing action in the cupola the copper oxides in the charge are brought directly to the metallic state by such reactions as $Cu_2O + CO = 2\,Cu + CO_2$; as we lack this deoxidizing action of carbon and carbonic oxide in the reverberatory, the oxides of copper are there chiefly deoxidized by sulphide of iron with the formation of sulphide of copper instead of metallic copper, $Cu_2O + FeS = Cu_2S + FeO$.

(3) In the reverberatory the matte has a stronger tendency to absorb any already precipitated metallic copper than in the shaft furnace, because the two substances lie for a longer time in contact, because their surfaces of contact, as they lie spread out each in a thin sheet, are much greater, and because the overlying slag rapidly carries oxygen from the comparatively strongly oxidizing atmosphere of the furnace to the iron of the matte, whose sulphur readily parts with iron to take in its place an equivalent of any previously precipitated metallic copper present, with which it is in such extended contact and for which it has nearly as strong an affinity as for iron.

This greater tendency to precipitate metallic copper in the shaft furnace impairs its efficiency in expelling arsenic and antimony, which are far more readily removed from matte than from metallic copper.

PRODUCTS.

1. *Black copper*, whose composition varies greatly. It is usually leaner than the blister copper of the reverberatory process, being contaminated with metallic iron so that its tenor in copper, rarely above 95 per cent., falls as low as 80 per cent. and occasionally even to 70 per cent.[1] Indeed in some places the product consists of a mixture of cupriferous pig-iron and ferruginous black copper. On this account it is in general much harder to refine than the blister copper of the reverberatory process.

2. *Matte*, with slightly less sulphur, much less copper, and much more iron than the regule of the reverberatory process from which metallic copper first precipitates. By limiting the thoroughness of the preceding roasting by adding pyritous ores, etc., mattes much richer in sulphur and leaner in copper may be produced and the precipitation of metallic copper or iron prevented.

TABLE XV.—*Composition of shaft furnace matte produced with black copper.*

	From Mansfeld.[a]	From Mansfeld.[a]	From Fahlun.[a]	From Riechelsdorf.[a]	From Phönixhütte.[a]	From Ore Knob, Egleston.[b]	From Swansea.[c]
	1.	2.	3.	4.	5.	6.	7.
Iron	15.8–16.3	4.6	17.2	13.7	16.3	20.66	0.2
Zinc		0.5	0.7				
Sulphur	20.2–24.4	21.4	24.5	22.5	22.3	23.44	18.5
Copper	57.2–61.2	72.5	57.8	61.3	60.0	54.93	81.1

[a] These from Kerl, Grundriss der Metallhüttenkunde, 1881, I, p. 184.
[b] Transactions of the American Institute of Mining Engineers, X, p. 44.
[c] Regule of reverberatory process, Swansea (Le Play).

While, as already pointed out, mattes rich in copper like this second matte are most advantageously oxidized by the roasting-smelting of the Welsh process, yet it often happens that for the cupola process they are roasted in heaps, stalls, or reverberatory roasting furnaces. In these, owing to the small quantity of sulphur the matte contains, a large amount of fuel is needed to roast it, and the oxidation of the sulphur and iron is exceedingly difficult. After being roasted it is again smelted exactly like the second matte, again yielding black copper, together with more or less rich matte.

3. *Slags.*—These are generally ferruginous and about singulo-silicates. The following are from Kerl, Grundriss der Metallhüttenkunde, 1881, p. 184:

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[1] Kerl, Grundriss der Metallhüttenkunde, 1881, p. 183.

Table XVI.—*Black copper shaft furnace slags.* (*See, also, Table XII.*)

	Mansfeld.	Stefanshütte.	Riechelsdorf.	Fahlun.	Röraas.	Phönixhütte.
Silica	31.6–38.1	31.7	31.7	21.0–32.8	30.9	25.36
Alumina	6.6– 7.3	....	2.8	0.0– 4.2	4.0	....
Protoxide of iron	45.1–52.4	62.3	47.8	64.8–69.1	66.3	70.13
Lime	3.4–11.6	0.4	8.1	0.0– 1.2	0.5	0.15
Magnesia	0.1– 1.5	0.1	3.9	1.5– 2.3	....	0.20
Dinoxide of copper, Cu_2O	0.6– 2.9	0.9	1.1	1.0– 1.6	0.6	....
Oxide of zinc	1.3– 5.5	....	....	0.0– 0.2	....	....
Sulphur	....	1.2	....	....	....	....

These slags often contain injurious impurities to such an extent that it is best to smelt them by themselves for an inferior grade of copper, though they are frequently used to flux silicious ores in the ore-smelting cupola. They are, indeed, often so free from copper that they may be thrown away.

SHAFT FURNACE PROCESS FOR OXIDIZED ORES AND NATIVE COPPER.

These are smelted directly and very much like thoroughly-roasted matte; as they are less ferruginous, the tendency to form salamanders is weaker, and the deoxidizing action may therefore be stronger; indeed, in the case of oxidized ores it is frequently necessary to make the reducing action very strong to prevent scorification of copper.

On account of the tendency of the product, black copper, to set, it is generally separated from the slag in internal crucibles.

COMPARISON OF THE REVERBERATORY AND THE SHAFT FURNACE METHODS.

1. *Composition of charge.*—With very silicious, aluminous, calcareous, or magnesian ores the reverberatory possesses a great advantage in being able to treat mixtures so refractory that they cannot be completely fused, but have to be dragged out of the furnace in skimming. These ores would require heavy fluxing in the shaft furnace where their complete fusion is imperative. With very ferruginous ores the shaft furnace, especially if water-jacketed, has the advantage, since these ferruginous slags strongly corrode the reverberatory, and its oxidizing atmosphere tends to form refractory high oxides of iron, which in the shaft furnace pass into the fusible silicates of the protoxide of iron. With mixtures suddenly varying between extreme limits the reverberatory has the advantage, since these variations do not derange it as they do the shaft furnace.

2. *Labor and fuel.*—While for simple smelting, whether of ore or of matte, the number of days of labor required per ton of burden is about the same for shaft furnaces and reverberatories, the latter demand much more highly skilled men, so that the actual cost for labor for the rever-

beratory is considerably higher and in remote places often very much higher, than for the shaft furnace.

The fuel consumption is moreover enormously greater in the reverberatory than in the shaft furnace. Vivian[1] estimates the average ratio of burden to fuel in the reverberatory at 2.2:1. In shaft furnaces, with fair coke, it should not fall below 5.5:1, even with refractory ores, and it rises even to 9:1 with well-managed furnaces and fusible ores (Nos. 3 and 10, in Table 12), or more than four times the average reverberatory duty. For the shaft furnace we must, however, use coke, charcoal, anthracite, or a decidedly dry bituminous coal, which *coheres* in burning, but does not *cake*. For the reverberatory we may use any form of fuel, from anthracite to spent tan and sawdust. The cupola or reverberatory will have the advantage as regards fuel consumption according to whether the price of coke, anthracite, or charcoal is less or more than, say, three times that of wood, bituminous coal, etc., per unit of calorific power. In the majority, and perhaps in the great majority of places the cupola has in this respect an advantage so enormous as to greatly outweigh all other considerations. This is especially true of works situated at a distance from coal-fields, the freight on coke by land being but little, if at all, greater than that of coal, and, of course, very much less than that of wood, per unit of calorific power. By sea freights still greatly favor coke per unit of actual smelting power (on account of its high efficiency in the cupola), though they are heavier on coke than on coal per unit of calorific power.

The use of gas furnaces would greatly diminish the disadvantage of the reverberatory as regards fuel consumption, but hitherto their saving in fuel has been outweighed by their increased cost for repairs.

3. *Arsenic and antimony*, it is generally stated, are much more completely expelled in the reverberatory than in the shaft-furnace process. While reasons have been given for believing that in the fusion of the roasted ore the cupola may be quite as effective in expelling these elements as the reverberatory is, if not more so, in smelting the roasted matte the reverberatory fusion favors their expulsion more than the cupola does. This is at least partly because in smelting the roasted matte the tendency of the shaft furnace to precipitate metallic copper is much stronger than that of the reverberatory. Most of the arsenic and antimony present concentrate in this metallic copper, from which their removal is far more difficult than if, as in the reverberatory, they had remained in a rich matte.

Reverberatory roasting, generally associated with reverberatory smelting, expels arsenic and antimony more completely than pile and kiln roasting, which are generally associated with cupola smelting. But these associations are by no means necessary, and, while the factor which generally causes the adoption of the reverberatory smelter, cheap

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[1] Copper Smelting, its History and Processes, p. 19.

bituminous coal, often at the same time favors reverberatory rather than pile roasting, yet reverberatory smelting is often preceded by pile roasting and cupola smelting by reverberatory roasting.

4. *Technical skill.*—The successful conduct of the reverberatory method depends, in a large degree, on the skill of the workmen and relatively less on the skill of the metallurgist in charge than does that of the shaft-furnace method. In the former, much depends on the proper care of the fire, skillful skimming, careful fettling, etc. In the latter, as in the iron blast furnace, success depends almost wholly on the skill of the metallurgist, the proper composition of the charge, ratio of fuel to burden, pressure of blast, size and number of tuyeres, etc.

5. *The first cost of construction* is in general somewhat greater per unit of daily capacity for reverberatory than for shaft-furnace plants. This is especially true in remote places on account of the much greater weight of the reverberatory plant, and the difference in favor of the shaft furnace is greater in large establishments, since the cost of the blowing apparatus does not increase in proportion to the capacity of the plant.

6. *Repairs* also are considerably heavier in the reverberatory than in the shaft-furnace method, at least in the case of water jacketed shaft furnaces. In remote places this tells strongly against the reverberatory.

7. *The loss of copper* in the slags is much less in the cupola than in the reverberatory, because in the former the powerful reducing action of the fuel hinders scorification of copper, and because the separation of matte and slag is more complete as the slag gently flows from above the matte in the cupola than when it is skimmed in the reverberatory. Even in smelting roasted matte for black copper in the cupola the slags are often clean enough to throw away, while in the case of the reverberatory such slags must almost always be resmelted.

8. *Fines.*—The objection to the cupola frequently urged, that great quantities of slags must be added to enable any considerable amount of fine ore to be smelted, is not sound. In the Raschette furnace as much as 80 per cent. of the total burden may be ungrouted fine ore without necessarily deranging the working of the furnace; by grouting and bricking them the proportion of fines can be raised still higher.

9. *Salamanders.*—The objection frequently made to cupolas on account of the formation of salamanders deserves little weight.[1] With proper furnaces, mixtures, and precautions to avoid excessive deoxidizing action they do not occur. There are cupola smelting works in this country which have been run on most ferruginous mixtures for years without the formation of a single serious salamander.

10. *Rapidity.*—The cupola process, thanks to its smaller number of operations, recovers the copper from the ore somewhat quicker than the reverberatory process.

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[1] Vivian, op. cit., p. 19.

11. *Size of establishment.*—The reverberatory process is so complex, and consists of so many different and delicate operations, at least when low-grade ores are treated, that the large staff of skilled men it requires can only be employed to the best advantage where the output is very large. It is especially hard to maintain in remote places. So, also, the large number of furnaces which it is desirable to have so as to reserve each furnace for a special class of work fits this method for large rather than small establishments. The simple cupola process, however, needing but few skilled men and few furnaces, is applicable to both large and small works.

12. *Bringing forward.*—For "bringing forward" rich mattes (blue and white metals, etc.) the reverberatory has special advantages. The small portions of sulphur and iron in these rich mattes are with difficulty oxidized by ordinary roastings and hardly at all in smelting in shaft furnaces, but they are very readily oxidized in the slow oxidizing fusions of the reverberatory, which thus, in addition to the fusion which may be effected equally well in the cupola, at the same time accomplishes work which in the cupola method has to be done by separate roastings. Hence the reverberatory may be advantageously used for bringing forward in places where its great fuel consumption would preclude its use for ore smelting.

RÉSUMÉ.

To sum up, for SMELTING ORE the *cupola* is especially advantageous—

I. With highly ferruginous ores.

II. Where the cost of anthracite coke or charcoal is not excessively greater than that of bituminous coal, wood, and other fuels fitted for the reverberatory only.

III. For oxidized ores.

IV. For low grade native copper.

V. Where, as in the case of lean ores, clean slags are a necessity.

The *reverberatory* is especially advantageous—

VI. With highly refractory silicious, aluminous, calcareous, or magnesian ores.

VII. Where the composition of the ore changes suddenly and greatly.

VIII. Where bituminous coal, wood, or other reverberatory fuel is very much cheaper than anthracite, coke, or charcoal.

IX. For smelting and immediately refining rich native copper.

X. Its disadvantage in yielding richer slags than the cupola weighs less heavily in case of rich ores.

For SMELTING FURNACE PRODUCTS of the several grades the following are the chief considerations which should influence our choice between the cupola and the reverberatory:

XI. The reverberatory expels arsenic and antimony from furnace products to a greater extent than the cupola does.

XII. Reverberatory smelting exerts an oxidizing action on the sulphur and iron of mattes which are rich in copper which the cupola does not exert at all, and which can be effected only with great difficulty in roasting operations strictly so called. This consideration and the preceding one apply with the greater force the richer the material to be treated is in copper.

XIII. Cheap coke, etc., favor the cupola; cheap bituminous coal, etc., the reverberatory.

XIV. The reverberatory bringing forward method, especially with low grade products, demands a large staff of skilled men, very difficult to maintain and relatively very costly in remote places, and advantageously occupied only in large establishments; while the simple cupola method, needing the minimum of skilled men, is applicable alike to large and small, remote and central establishments.

ADDENDUM.

MATTE ROASTING IN OPEN REVERBERATORY FURNACES AT THE WORKS OF THE ORFORD COPPER AND SULPHUR COMPANY, BERGEN POINT, N. J.

The matte treated in this operation is that produced in ore smelting (column 15 in table of smelting in cupola or shaft furnaces). The roasted matte is smelted, with the results given in column 16 of the same table. (See pages 78 and 79.)

The principal features of this roasting are as follows:

Dimensions of furnace:	
Length of hearth inside	60′
Width of hearth inside	12′
Length of grate	5′
Width of grate	3′
Depth of grate below top of bridge	2′
Height of roof above bridge	0′ 9″
Height of roof above hearth at bridge	2′ 6″
Ratio of hearth area to grate area	48:1
Square feet of hearth area per ton of matte roasted per 24 hours	60
Roasting operation:	
Thickness of charge on hearth	7″
Hours roasted	24
Tons roasted per 24 hours	12
Percentage of copper in matte treated	30 to 60
Percentage of sulphur in the roasted matte	About 5
Outlay of labor and fuel:	
Kind of fuel employed	Bituminous coal.
Fuel burnt per shift, tons	2
Tons fuel per ton matte roasted	0.33
Men employed at furnace per 24 hours	8
Days of labor per ton of matte roasted	0.67

All tons given in the above table are of 2,000 pounds. Sixty-seven parts of this roasted matte, mixed with 17 parts of the same matte unroasted and 16 parts of rich slags, are smelted in cupola furnaces. (See column 16, table of smelting in cupolas or shaft furnaces, pp. 78, 79.) The product obtained in this smelting operation consists of about 67 per cent. of black copper and 33 per cent. of rich matte, containing about 80 per cent. copper. The slag produced at the same time contains from 0.75 to 2.66 per cent. of copper, and is resmelted in cupola furnaces with a small quantity of raw cupreous pyrites, yielding a matte containing 35 per cent. copper and a new slag containing 0.6 per cent. copper, which is thrown away.

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