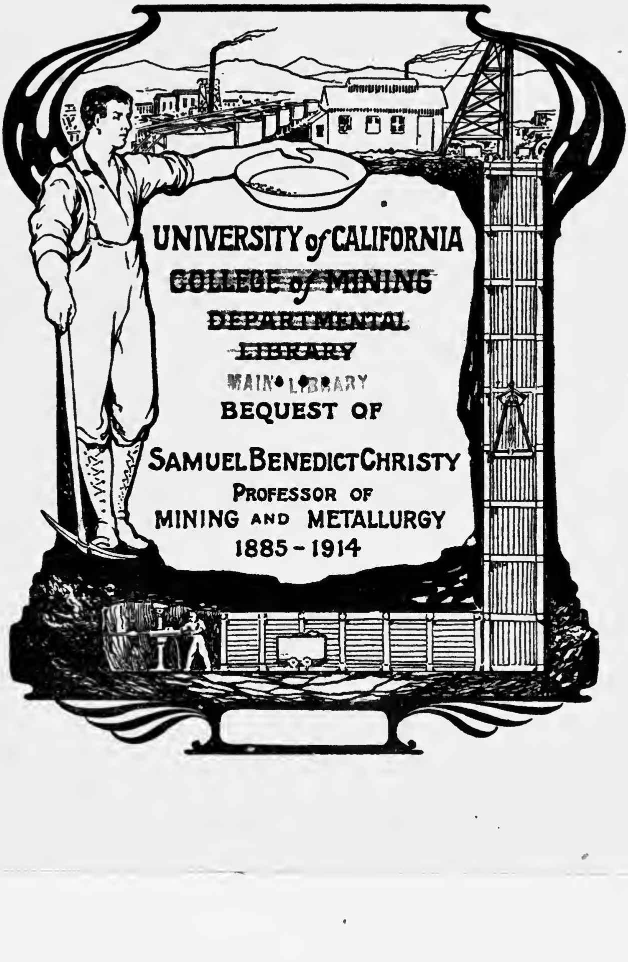


Saml. B. Chumley —

GEOLOGICAL SURVEY OF MISSOURI.

INDUSTRIAL REPORT

ON

LEAD, ZINC AND IRON,

TOGETHER WITH

NOTES ON SHANNON COUNTY

AND ITS COPPER DEPOSITS.

By CHARLES P. WILLIAMS, Ph. D.,
ACTING STATE GEOLOGIST.

JEFFERSON CITY.
REGAN & CARTER, STATE PRINTERS AND BINDERS.
1877.

Session Acts, Page 20.

That the sum of fifteen hundred dollars be and the same is hereby appropriated, out of any money in the Treasury not otherwise appropriated, chargeable to the Revenue Fund, for the printing of five thousand copies of the report of the Acting State Geologist.

REPORT.

To His Excellency, CHAS. H. HARDIN, *Governor of Missouri:*

SIR: I have the honor to forward herewith the report of the operations of the Geological Survey in the direction of an examination of the conditions surrounding certain of the prominent industries of Missouri.

The economical aspects presented in the production of two of the great staples of the State have been investigated with considerable detail by a study of the means, appliances, and the chemical problems connected with the manufacture of the metals, lead and zinc. Some attention has been given to the iron industry as well, but the general stagnation of this great business has manifested itself by such a decrease in the activity of the furnaces of this State, that facilities were wanting for the collection of sufficient data for anything approaching that comprehensive and thorough study which the importance of the subject demands. The production of charcoal iron is, however, presented with some degree of fullness.

In shaping the work intrusted to me by the Board of Curators of the State University, towards the attainment of the ends indicated, I have been influenced by the fact that while a large amount of the structural geology of the State, and the special geology of its mineral deposits had already been made public in the reports of Messrs. Swallow, Pumpelly, Broadhead and others, but little had been accumulated towards unfolding the history of the technology which the conversion of the raw materials from those mineral deposits into commercial products had created within the borders of Missouri. Further, the appropriation of five thousand dollars yearly was not adequate to keeping in the field a force sufficient for thorough and rapid work in those counties which had not before received special attention.

The material which could be collected under existing circumstances is not full enough for a geological history of those counties; however, as interesting copper occurrences have long been known to exist within the limits of Shannon county, some space has been devoted to a geological history of those deposits, this course being regarded as in accord with the design of this report.

Notes on the supposed gold deposits of Northern Missouri have also been given in the appendix.

I must acknowledge, in this connection, the very efficient assistance rendered in the laboratory by Mr. A. W. Hare. His zeal and industry have enabled me to make a much fuller presentation of the chemical illustrations of the work than could have been done otherwise.

Other aid, whenever rendered, is acknowledged in the report.

Some of the notes on the St. Louis industrial district collected, by Dr. Adolph Schmidt, have been used with his sanction. These notes, having been taken as early as 1872, have lost much of their value in the lapse of time and the progress and growth of the industries, and their reproduction would only extend this report without presenting the actual conditions of the manufactures to which they relate. Much of present value is contained in Dr. Schmidt's notes on the iron deposits, but as an account of these was foreign to this present work, these notes have not been reproduced.

All of which is respectfully submitted.

Your obedient servant,
CHAS. P. WILLIAMS,
Acting State Geologist.

Rolla, Mo., December 13, 1876.

LEAD INDUSTRY.

CHAPTER I.

MINERALOGY AND GENERAL METALLURGY OF LEAD.

The geological positions and modes of occurence of the ores of lead throughout Missouri have been described in such detail in preceding reports, that this report is restricted to a discussion of the economical considerations connected with the conversion of those ores into marketable lead. The facts that the lead bearing formations occupy a very considerable part of the area of South Missouri, and that those formations contain concentrations of plumbiferous minerals, of which many have been strikingly remunerative, are known much more generally than are the character of the ores, the features of their metallurgical treatment or the commercial value of the product as a factor in the material wealth and prosperity of the commonwealth.

The pressure of a large and rapidly increasing production from the mines has left little time for examination or discussion of those smelting methods, which, adopted in the early development of the lead industry, have been reproduced with but trifling modifications at each new mining centre. At the present time when lead production has become a permanent industry in the State, and Missouri brands of metal are in competition with refined leads from other domestic, as well as from foreign establishments, in the markets of the east, it may be wise to scrutinize the common metallurgical practices of the State more closely than has heretofore been done, in order that the fullest possible advantage may be gained from them.

Among the various methods of lead smelting and the wide range of variations which local conditions and experiences have added to them, there are but few which have any extended use in the State, and it is these alone which need be examined in detail to ascertain how far the practice, which is in general adapted to the ores to be treated, is worked to its fullest capabilities or, in other words, how nearly its utmost advantages are realized. To reach this knowledge, some account must be taken of the experiences and results elsewhere, and the comparisons which will be thus instituted will not only explain the local practice, but furnish suggestions which may be adopted with advantage leading to cleaner work and closer economy.

A consideration of the metallurgical processes involved in the production of metallic lead necesitates a knowledge of the composition of the ores and of those associated minerals which influence the furnace operations or affect the quality of the resulting metal. A description of the mineral species occurring in the lead deposits is therefore given, and their composition generally given from the results of chemical analyses. This has been followed by a short account of the various practices of lead smelting, including the relations of the foreign minerals to the operations and to the product and a fuller discussion of the chemical changes taking place in those methods which are in service in Missouri, together with descriptions of the apparatus in which these changes are brought about and illustrations furnished by analyses of the intermediate and final products.

GEOGRAPHICAL DISTRIBUTION OF LEAD ORES.

The classification of the lead regions of the State recognizes three chief areas of lead developments, denominated, respectively, the Southwestern, the Central and Southeastern districts, a distinction in the main geographical, though warranted to some extent from a scientific standpoint by geological features peculiar to each.

The Southwestern Lead Region—This includes the counties of Jasper and Newton, as those in which in the most active developments have been made, besides Greene, Dade, and portions of McDonald, Barry, Stone and Christian counties. The ores are found in the sub-carboniferous system—notably in certain upper members of the Keokuk or Archimedes limestone. The productive rocks have a thickness of about 125 feet, consisting, in a descending order, of (a) more or less broken and softened chert, accompanied by large masses of clay and sand, (20-75 feet), (b), a silicious limestone (occasionally wanting and never exceeding 20 feet thickness), and (c) a maximum of thirty feet of alternate layers of limestone and chert. This productive series rests upon upwards of 140 feet of unproductive limestone, distinguished as the bed-rock.*

The Central Region—The principal points of development of this region are found in the counties of Miller, Cole, Morgan and Moniteau, but the district includes in addition to these, Saline, Cooper, Pettis, Benton, St. Clair, Hickory, Camden, Osage and Maries. In parts of Moniteau and in Saline and Cooper counties, the deposits are found in the same horizon as in the Southwestern region, viz: The Archimedes limestone, and therefore, a description other than a geo-

---

* For full descriptions, consult Schmidt & Leonhard, in Broadhead's Report for 1873-1874, page 401 *et seq.*

graphical one would include these in the Southwestern rather than in the Central region.

In the other counties named, the workable deposits of lead minerals are found in members of the lower silurian system, the second magnesian limestone being the chief productive rock in the northern and western parts of the region, while the third magnesian limestone constitutes the horizon of the deposits of the southern and eastern.

The Southeastern Lead Region: The seats of the most active operations are found in the counties of Franklin, Jefferson, Washington, St. Francois, Madison and St. Genevieve. To this list may be added the counties of Crawford and Bollinger, within the limits of which lead deposits have been found and developed to some extent.

The productive geological formation in this area, is the third magnesian limestone, which, excluding the southwestern and portions of the central regions, is the great lead-bearing series of the State. It therefore has a wide extent, preserving, over the great area in which is found a remarkable uniformity of character, * and is nearly a true dolomite, containing the calcic and magnesic carbonates in the proportion of one to one. It has a large vertical range, having a thickness, where entire, of probably more than five hundred feet.

To the three areas above indicated, may be added a fourth, *The Southern*—including a large extent of country, underlain by the magnesian limestone series, and in which lead ores have been discovered, either *in situ* or as float mineral, but in which there has been little, if any, systematic exploration. Some little work or casual examination has resulted in the discovery of lead ores in the counties of Phelps, Texas, Wright, Douglas, Webster, Christian, Taney, Stone, Barry, Laclede and Dallas.

ORES AND ASSOCIATES.

In the classification adopted by metallurgists, sulphuretted ores are distinguished from the oxidized ores. The first class includes but one species of any practical importance in the production of lead, viz: galenite; the second class includes cerrusite, anglesite and pyromorphite, all of technical importance, but still subordinate to galenite.

Galenite (Von Kobell): This species, commonly called Galena, and, by the miners, "mineral," is emphatically the ore of lead. Chemically it is lead sulphide, containing, by theory,

1 atom of lead	207	or	86.61 per cent.
1 atom of sulphur	32	or	13.39
	——		——
1 molecule lead sulphide	239		100 00

---

* Swallow's *First Annual Report;* Schmidt and Leanhard, *loc. cit.;* J. R. Gage, *Broadhead's Report,* page 601 *et seq.*

Usually it is contaminated to a greater or less extent by various other metallic sulphides or by intermixture of certain earthy substances (gangues), either loosely associated with it, or so intimately disseminated through it as to be unrecogizable by any other than chemical means.

Galenite crystallizes in the isometric system, usually in cubes, or in cubes with planes of the octahedron or dodecahedron. The cleavage is ordinarily cubical and perfect; occasionally traces of the octahedral cleavage are observed. This latter is particularly noticed in the galenite from the Pequa Mine, Lancaster county, Pennsylvania. * Large cubical crystals are of common occurrence at many of the localities of this State, and specimens of this description, measuring upwards of three inches edge are not infrequent. It is also found tabular, and passing from coarsely crystalline through many grades to the finely granular, the last constituting the steel-grained ore. Its color is lead-gray; lustre, metallic; fracture generally even, more rarely subchonchoidal; hardness 2.5—2.75; specific gravity, 7.25—7.75—a cubic foot of the solid mineral, therefore, weighing from 435 to 484 pounds.

Though distinguished among the mines of this State, by the names of *mineral* or *blue* mineral, the galenite receives various other names, according to the variations in the forms and modes of occurrence. When it occurs as large or coarsely crystallized masses, it is generally called *block* or *chunk* mineral; smaller crystals constitute the *cog* mineral, while still smaller crystals are denominated *dice* mineral.

Among the foreign matters found in galenite, either in a state of mechanical admixture, or in various forms of combination, or as isomorphus replacements of either the lead or sulphur, are, silver, gold, antimony, arsenic, copper, cadmium, zinc, iron and nickel; more rarely, selenium, and, reputed in the galenite from the Dept. Charente, France, platinium. † Notable amounts of these matters influence the metallurgical treatment of the ore, and the quality of the product from it, while, in some cases the presence of those metals of much greater value renders the lead a subordinate product of the ore, or causes it to act as an adjunct to the extraction of those metals.

Silver is rarely, if ever, absent. ‡ It occurs occasionally in the galenite, in the free or native state; commonly it exists as argentic sulphide, isomorphus with and replacing more or less of the lead sulphide. This is believed to be the form of combination in cases in

---

* Genth's *Preliminary Report on Mineralogy of Pennsylvania*, p. 12.

† Dana's System of Mineralogy.

‡ The same claim is made for gold. See, on this point, Percy's *Metallurgy of Lead*, page 96, *et seq*.

which the silver does not exceed one per cent. of the galenite; in ores richer than this, the silver is regarded as occurring as some one or more of its ores mechanically mixed through the galenite. Malaguti and Durocher examined a large number of specimens of galenite and found the silver to range in amount between a mere trace, and upwards of 7 per cent.*

A somewhat popular belief is to the effect that the texture of the galenite is, in some measure, an indication of the relative amount of silver it contains, the finely granular varieties being regarded as relatively richer, and the coarsely granular or crystalized, as poorer. This view is not supported, either by chemical analyses or by experience, and is, therefore, not to be relied on. The coarsely granular galenite from some localities, *e. g.* Arizona (according to Küstel), is decidedly richer in silver than the finer varieties. The argentiferous character is decidedly more closely connected with the character of the deposit furnishing the galenite, and with the structure and lithological characters of the rocks enclosing the deposits, than with the texture of the ore itself; but even these fail as reliable indications, or hold good only for well established silver-bearing districts. Assays of the ores from a newly discovered or opened mining region are the only trustworthy guides in this direction.

The galenite from localities in this State are not argentiferous in any practical sense. This has been so well established by repeated assays of the ores, and has been so fully substantiated by analyses of the leads produced from them that, were it not for the fact that portions of the State are very frequently excited over reported silver discoveries within the limits of Missouri, it would not be necessary to make more than a passing allusion to this fact of the invariable absence of notable amounts of silver from our ores. Periodically reports appear of discoveries of rich silver ores, the search for which originates from some vague tradition connected with De Soto's expedition, or with the mythical silver mining Indian, and which generally have no more substantial basis than a few specks of calcite or pyrites in a limestone, or at the best, the discovery of a galenite of fine-grained texture. †

I have collected the following results of the estimation of silver in the galenas from various parts of the State, the results being given both in per centage and in the customary manner of ounces to

---

* Annales des Mines, 4 Ser. vol. xvii, 1855.

† These are facts within my own experience, for I can count upwards of fifty cases where limestone has been brought to me for silver ore. In a number of cases, yielding to the importunities, assays have been made, it is needless to say with what result, and without carrying conviction.

the ton of two thousand pounds of ore, and are accompanied, wherever it was known, by the per centage of pure galena, or lead sulphide in the ore:

	Locality.	Silver, Per Cent.	Silver Ounces, Per Ton.	Per Cent. Galenite in the Ore.
I.	Mine La Motte	.0027	0.86	....
II.	Mine La Motte	.0025	0.80	....
III.	Mine La Motte	.0012	0 38	....
IV.	Perry's Mines	.0009	0.29	"Well Crystalized."
V.	Potosi Mining Company	.0012	0.38	.. ....
VI.	Village Diggings, Granby	.0031	1.00	....
VII.	Holman Diggings, Granby	.0039	1.25	97.05
VIII.	Trent Diggings, Granby	.0039	1.25	96 86
IX.	East Point Diggings, Granby	.0052	1.75	....
X.	Temple Diggings, Joplin	.0039	1.25	97.24
XI.	Swindle Diggings, Joplin	.0021	0.75	97.79
XII.	Oronogo	.0031	1.00	96 34
XIII.	Jefferson county	.0114	3.75	91.84
XIV.	Franklin county	.0021	0.75	93.09
XV.	Phelps county	.0009	0 29	47.87
XVI.	Phelps county	.0038	1.25	93.36
XVII.	Maries county	.0039	1.25	94.00
XVIII.	Pulaski county	.0029	0.87	89.12
XIX.	Wright county	.0030	1.00	84.29
XX.	Laclede (?) county	.0030	1.00	87.00
XXI.	Crawford county	.0011	0.36	89 97
XXII.	Miller county	none.	none.	99.34
XXIII.	Morgan county	trace.	......	99.89
XXIV.	Morgan county	trace.	......	98.76
XXV.	Morgan county	none.	... ..	99.87
XXVI.	Madison county	.0124	4 00	"Pure."

Assays I to V are by Dr. Litton, (*Swallow's Report*, 1854 and 1855); VI to XII, Chauvenet (*Broadhead's Report*, 1873 and 1874); XIII to

XXI, by myself, or under my direction in this laboratory; XXII to XXVI by Chauvenet (*loc. cit.*)

The number of assays and the range of localities might have been greatly extended, but enough has been given to show the quality of the Missouri galenite, in respect to the amount of silver contained. The maximum amount thus far found, is in the specimen from Jack's diggings, Mine la Motte, (XXVI in the list), and gives a value of $5.16 per ton of similar ore. This value is entirely too small to cover expenses of desilverization, even if the separation of silver and lead could be fully made in practice. Reference to the various analyses of the pig lead from the furnaces using ores from the mines of Missouri will show that the amount of silver present in these ores must have been small and scarcely noteworthy.

Antimony is of very frequent occurrence in galenite, upwards of five per cent. being recorded as existing in the ores from some foreign localities. It probably exists as antimony tri-sulphide (*stibnite*) either alone or combined with other metallic sulphides, and disseminated through the galenite. In whatever form of combination, its presence is objectionable, for its separation by known mechanical means is impossible, and its presence in the furnace causes loss of lead (and silver) by volatilization, and impairs the quality of the metallic lead.

Careful analyses, made either by myself or under my direction, were undertaken to determine the existence of antimony in the Missouri galenites, and its amount, if found to exist. Numerous specimens of ores, pig leads and slags have been examined, usually operating on large amounts, and while the results show that antimony is almost invariably present, it always occurs in amounts too small to exert any marked influence either on the smelting processes or the resulting lead. Its existence in the galenites of Missouri is therefore a point more of scientific interest than of technical importance. (For results see table of results of analyses, galenites, pages 16 and 17, also analyses of pig leads from the various furnaces.)

Arsenic has been found in faint traces in some of the ores, and shows in small amounts in the metallic leads produced in the State. It operates in the same direction as antimony, but the remarks above concerning the latter apply equally to the arsenic.

Zinc as sulphide (*sphalerite* or *blende*, the *black jack* of the miners), occurs in association with galenite, either intimately mixed with and disseminated through it, or so coarsely associated with it that each mineral is easily distinguished by the eye. In cases of intimate mixture or dissemination, mechanical preparation will not effect anything approaching a thorough separation of the two materials, while in the other cases hand-sorting will relieve the galenite largely of its associated blende. Its action in the furnace, within certain limits, is not sensibly injurious beyond the increased loss of lead by volatilization, nor is its presence sensibly felt in the metallic lead, for the latter does not appear to be able to take up in the furnace but a very small per cent. of its weight of metallic zinc.

Cadmium, where found, is probably an associate of the sphalerite rather than of the galenite. In selected specimens of nearly pure galenite, it is found only in mere traces, and I have been able to find it in the metallic lead in but one case, and that of metal produced from the treatment of slags. (Granby slag lead, 0.00296 per cent. cadmium.)

Both modes of association of the blende and galenite are common in the lead deposits of the State, but the most frequent is that of loose association, which permits of easy separation of the two minerals. Large amounts of blende are thus produced as an incidental or subordinate product of the lead mines of the State. These matters will be more fully treated in the section of the report devoted to the zinc industry of the State, but the annexed analysis of blende may not be without interest in this connection.

Analyses I. was made under my direction by Mr. John D. Greason, the specimen showing a small amount of calamine and giving a separation of gelatinous silica, on treatment with acid. The silica is therefore assumed as existing as calamine, and from it the amount of the other constituents (zinc, oxide and water) were calculated. The specimen was from Granby, Newton county, and was of a yellowish-brown color. The association of blende and calamine is not uncommon, and is thus described by Schmidt (*loc. cit.,* page 391):

"In many places the blende seems to pass gradually into calamine, there being between the two minerals a layer of dark colored, fine-grained blende, intimately mixed with portions of columnar structure, and resembling calamine. This mixture passes into regular columnar or fibrous calamine, which itself is dark colored near the blende, and is lighter and more transparent the more remote it is from the unaltered blende. Single crystals or large crystalline aggregations of blende are often surrounded by a thick layer of calamine of botryoidal form and radiating columnar structure. The surface of the enclosed piece of blende is then generally corroded and tarnished. Blende, especially the coarsely crystalline variety, is often intimately mixed with galena. Mr. Broadhead observed, at Leadville, round globules of from one-sixteenth to one-eighth inch in diameter, grayish-brown color, and radiating fibrous structure, on crystals of galena. Mr. Chauvenet analyzed them, and found them to be pure sulphuret (sulphide) of zinc, without any iron or lead."

Analyses II and III are by Mr. Chauvenet (*loc. cit.*):

		I. Granby. [Greascn.]	II. Leadville. [Chauvenet.]	III. Joplin. [Chauvenet.]
	Zinc	61.934	64.87	65.92
	Iron	0.788	0.37	0.32
	Cadmium	trace.	0.623	0.509
	Copper	0.043	....	....
	Antimony	0.0018	....	....
	Sulphur	31.269	....	....
Calamine,	Silica	0.911	1.410	1.250
	Zinc oxide	2.461	....	....
	Water	0.231	....	....
Gangue,	Lime	1.361	....	....
	Magnesia	0.043	....	....
	Carbonic acid	1.148	....	....

Iron occurs probably as pyrite, which is not objectionable in the furnace charge in small amounts; or perhaps in some ores as mispickel or arsenical pyrites, which introduces arsenic into the lead.

Copper, in the form of chalcopyrite or copper pyrites is noticeable in the galenite from some of the localities of the southeastern district (St. Joe Mines, Mine la Motte). Smaller amounts of copper pyrites are found associated with the ores in almost all the deposits of the southwestern and central districts. Other forms of copper combinations or minerals doubtless occur in the galenite, for very few specimens but give small amounts of copper on chemical analysis, though no chalcopyrite or other copper-bearing mineral can be distinguished under the magnifying glass.

Nickel and Cobalt are found in notable amounts in association with the lead ores from the Mine la Motte property, and are a source of profit in the metallurgical treatment of those ores. Other localities in the southeastern district also yield nickel-bearing minerals as associates of the lead ores, and traces or small amounts of nickel are found in many specimens of apparently pure mineral from many localities in the southwestern, and, more rarely, from some in the central region. Most pig leads smelted in the State show small but still appreciable amounts of nickel on careful analysis.

The results of analyses of ores from various Missouri localities will exhibit the character and amounts of the foreign metals—excluding the silver—found in the galenite mined in this State. No estimations were made of the sulphur and lead, because, as the specimens were generally selected as being free from visible admixture of foreign sulphides, such estimations would have no practical significance. Effort was made to have the specimens analyzed as free as possible from earthy gangues, but wherever otherwise, note is made in the description of the locality:

FOREIGN METALS IN APPARENTLY PURE GALENITE.

	Per cent. Antimony.	Per cent. Copper.	Per cent. Iron.	Per cent. Zinc.	Per cent. other foreign metals.
Southwestern Region.					
I	trace.	trace.	.05867	.06782	......
II	.07429	.00478	.02169	.00938	Nickel.. .0660
III	.00551	.00239	.08602	1.35554	......
IV	.03323	.01038	.13020	2.02701	......
V	.02784	.03724	.08400	1.31648	......
VI	.00709	.01517	.15260	2.22539	......
VII	.02764	.01677	.03220	1.76558	......
VIII	trace.	none.	.12040	.14679	......
IX	......	......	.09240	.10601	......
X	.04912	none.	.10640	.07320	......
XI	......	......	.5880	61.96314	......
XII	......	......	.07040	.08771	......
XIII	.32194	.04232	.02870	.02011	Ni......... trace.
Central Region.					
XV	.04178	trace.	.02310	.00804	Ni...... .09755
XVI	.16545	.15141	.05040	1.26352	......
XVII	.00317	.00798	.07050	.00268	......
XVIII	.00475	.01197	.00980	.00602	......
XIX	.04754	none.	.02940	trace.	......
XX	trace.	none.	.02240	.00372	......
XXI	trace.	none.	.78400	.00019	......
XXII	.00392	.02151	.02513	.03945	......
Southeastern Region.					
XXV	.05069	4.6557	8.1304	.3255	Cd... trace.
XXVI	.00400	.0730	1.681	.2150	As... .002 Ni... .012 Co... .023
XXVII	.032	.097	1.583	.339	As... .003 Ni... .031 Co... .019
XXVIII	trace.	none.	2.529	.040	Ni... .214 Co .. trace.
XXIX	trace.	.080	1.953	none.	Ni... .168 Co... .357
XXX	trace.	.103	1.435	trace.	Ni... .213 Co... .042

Southeastern Region.

	Per cent. Antimony.	Per cent. Copper.	Per cent. Iron.	Per cent. Zinc.	Per cent. other foreign metals.
XXXI	.010	.009	.118	.034	As...... trace.
XXXII	none.	none.	.05180	.0436	Ni&Co.. traces.
XXXIII	.031	.041	.011	.054	As...... .026
XXXIV	.004	trace.	.055	.015	As...... .009
XXXV	trace.	.754	.494	.374	As...... trace.
XXXVI	none.	13.319	13.517	.020	Ni... .233 Co... trace.
XXXVII	none.	0.798	2.244	0.611	Ni&Co.0.137
XXXVIII	trace.	0.167	1.559	1.276	" 0.573
XXXIX	trace.	0.040	0.067	0.237	" 0.042
XL	trace.	0.091	0.823	0.670	" 0.078

No. I. Washed galenite from jig at Granby, Newton county; II, block mineral, Jasper county, Joplin; III, block mineral, Granby; IV, V and VI, washed mineral from jigs at Granby; VI, Holman's diggings, Newton county. Chauvenet's analysis of ore from these diggings shows: (*loc. cit.* page 388,) lead, 84.06; zinc, 0.94; iron, 0.16; antimony, none; silver, 0039 [=1¼ oz. per ton of 2000 lbs. ore]; silicious matters, 0.61 per cent. The lead corresponds to 97.05 per cent. galenite, and the zinc, to 1.41 per cent. sphalerite; VIII, Birch diggings, Joplin, Jasper county; IX, another specimen from same locality as VIII; X, XI, XII, from Joplin; [XI, was almost entirely sphalerite, but analyzed to ascertain if the blendes of a given locality were more antimonial than the galenites of the same locality]; XIII, McGee shaft, Holman diggings, Granby; XIV, Dade county.

XV, block mineral of remarkable purity, selected at the Eagle Furnace, Sec. 23, T. 43, R. 14 W., Cole county; XVI, Cole county, exact locality not known—collected by previous surveys; XVII, ore used at Murphy and McClurg's furnace; XVIII, Pioneer furnace ore; XIX, Star furnace ore; XX, Buffalo furnace; XXI, ore used at O'Brien's Scotch hearth; XXII., large crystals, from Richardson's Ozark shaft, Ozark township, Phelps county.

XXV, St. Joe Mines—with visible admixture of chalcopyrite, pyrite and blende—contained 31.032 per cent. lead; 5.826 per cent. silica, besides calcite, and showed distinct traces of nickel, but the amount was not estimated; XXVI, "sludge," from the washers at the St. Joe mines; XXVII, from washers at St. Joe mines—headings; XXVIII, from Bluff diggings, Mine la Motte; XXIX and XXX, washed ore from Bluff diggings, Mine la Motte, sludge and headings, respectively; XXXI, ore used at Valle furnaces, Jefferson county;

XXXII, Jefferson county—No. XIII of the silver assay results; XXXIII, Perry furnace, near Potosi, Washington county; XXXIV, Hopewell furnace, Washington county; XXXV, galenite, washed from Scotch hearth residues at Hopewell furnace for re-treatment; XXXVI., from "Seed-tick diggings," Mine la Motte tract, analyzed under my direction by W. C. Minger—sample showed chalcopyrite; XXXVII, ore from Mine Desloge, containing 19.675 per cent. lead by humid analysis; XXXVIII, XXXIX, XL, dressed ores from Desloge concentrating machines, containing respectively 63.61, 76.42 and 67.96 per cent. metallic lead, estimated by fire assay, each, mean of two.

OTHER SULPHURETED ORES.

The following species containing lead and sulphur are described by mineralogists:

ZINCKENITE (G. Rose,)

contains, by theory, sulphur, 22.1; antimony, 42.6; lead, 35.3; resembles stibnite and bournonite, but is distinguished from these by its greater hardness and higher specific gravity.

PLAGIONITE (G. Rose,)

has the following per centage composition: sulphur, 21.3; antimony, 38.2; lead, 40.5.

JAMESONITE (Haidinger,)

contains, theoretically, sulphur, 21.1; antimony, 32.2; lead, 43.7; iron, 3.0.

BOULANGERITE, (Thaulow.)

The percentage composition, as calculated from the formula, is as follows: sulphur, 18.2; antimony, 23.1; lead, 58.7.

GEOCRONITE (Svanberg.)

In composition it contains, sulphur, 16.5; antimony, 16.7; lead, 66.8.

BOURNONITE (Jameson,)

has the following composition: sulphur, 19.7; antimony, 25.0; lead, 42.4; copper, 12.9. None of the above species have been noticed in any of the lead deposits of Missouri. It is possible that some of the foreign metals (*e. g.*, antimony, copper, etc.,) enumerated in the tables of analyses of galenite may be so combined and disseminated throughout the lead ores. This, however, is purely hypothetical.

B.—OXIDIZED ORES.

Of the oxidized ores, three mineral species only are of significance to the metallurgist, and with reference to Missouri localities, but one requires more than a passing notice.

Pyromorphite (Hausmann): Occurs frequently in aggregations, either botryoidal, reniform or globular; sometimes granular and fibrous; when crystallized, the forms belong to the hexagonal system, being slightly modified six-sided prisms. It has a resinous luster; color, usually various shades of green, from a light yellowish to a deep olive. Hardness, 3.5—4; specific gravity, 6.5—7.12.

It consists of triplumbic phosphate and lead chloride, with 76.4 per cent. metallic lead by theory. Frequently the phosphoric acid is replaced, more or less, by arsenic acid, the lead oxide by lime, and the lead chloride by calcium fluoride.

Beautiful specimens were found at the Washington or Silver Hill mine, North Carolina, and at the Chester County and Wheatley mines, near Phœnixville, Pennsylvania. At the last-named large quantities were mined and smelted. The mineral is of very rare occurrence in the lead deposits of Missouri. When found it is usually as a coating either on galenite or cerussite. It has been noticed only in the southwestern district.

Anglesite (Beudant): Is also of rare occurrence, being found in the form of small, yellowish green crystals, with adamantine lust, occurring in cavities in crystals of galenite. It consists of sulphuric acid, 26.4; lead oxide, 73.6 per cent., and contains 68.3 per cent. metallic lead by theory.

Cerussite (Haidinger): The only oxidized lead ore metallurgically important in Missouri. It is lead carbonate, containing, by theory:

Lead oxide	83.5 per cent.
Carbonic acid	16.5 per cent.
	100.0
Metallic lead	77.5 per cent.

Its crystalline forms belong to the orthorhombic system; occurs also granular, massive and compact; color, grayish-white, yellow, brownish-black. grayish-black and colorless; lustre, adamantine, passing to vitreous and resinous; hardness, 3.35; specific gravity, 6.47 to 6.48.

Crystallized cerussite is found abundantly at many of the mines in the southwestern district, notably at Oronogo, Jasper county, where excellent specimens are found illustrating the change of galenite into cerussite.* The ores from the upper portions of the lead deposits of Dade county, show interlacing crystals of cerussite contained in brown iron ore or limonite.

A common form under which cerussite presents itself at many localities in all of the lead regions of the State, is as earthy, amorphous, compact masses of white, yellow or reddish color. This form is the "drybone" of the miners. Of still more frequent occurrence is the gray or ashy variety surrounding cubical crystals of galenite or filling cavities in them. This form is the "ash mineral" when the galenite predominates in the mixture, or the "wool mineral" when the carbonate is in excess.

The carbonate ores of the furnaces always contain some lead sulphide and are, in reality, mixtures of cerussite and galenite. The pure or even approximately pure cerussite is never found in quantities sufficiently great to warrant notable changes in the routine of treatment in the furnaces.

The following are analyses of the carbonates:

	I. Williams.	II. Chauvenet.
Lead carbonate	84.0770 per cent.	(Lead). 66.35 per cent.
Lead sulphide	6.2387 per cent.	
Zinc oxide	2.0914 per cent.	0.75 per cent.
Antimony oxide	.1095 per cent.	
Copper oxide	.0578 per cent.	
Ferric oxide	.4340 per cent.	1.51 per cent.
Insoluble matter	2.4904 per cent.	9.95 per cent
Carbonic acid	Not est.	Traces.
Lime, magnesia		
Per cent. lead	65.287	

No. I from Granby; No. II also from Granby, described by the analyst as "massive; of brownish-red color."

Cerussite in the form of stalactites has recently been found in notable quantities in a "dropped opening" or low cave, struck by a drift at the Douglass and Morgan diggings on Sec. 6, T. 25, R. 30 W., (Granby Company's property), Newton county. Mr. Thomas Richeson, the President of the Granby Company, kindly presented a specimimen, measuring nearly three inches in length and with a diameter at the base of half an inch. Some fragments were detached and were found to have a specific gravity (at 19° C.) of 6.099, determined by the flask

---

* See Schmidt (*loc. cit.* page 389) on this point.

method. By analysis, the following was ascertained to be the composition:

Lead carbonate	98.453 per cent.
Lime carbonate	0.259 per cent.
Magnesic carbonate	0.303 per cent.
Lead sulphate	0.107 per cent.
Ferric oxide	0.654 per cent.
Silicious matter (clay?)	0.205 per cent.
	99.981

The ferric oxide occurs as mere stains or patches on the exterior of the specimen.*

C.—ASSOCIATES OF LEAD ORES—GANGUES.

CALCITE (Haidinger.)

This mineral is almost the exclusive gangue of the ores in the southwestern region, and preponderates also in the deposits of the central and southeastern regions. Chemically it is lime carbonate. Its crystaline forms belong to the hexagonal system, the cleavage being eminently rhombohedral. Hardness 2.5—3; lustre, vitreous; color ranges through a great variety of shades, from the colorless varieties to white, gray, yellow, red, brown and black.

Its most common form in the lead deposits of the State is that of a coarsely crystalline mass, generally colorless, or nearly so, and translucent. It is known among the miners as "glass tiff." Specimens analyzed in the laboratory of the survey show the annexed composition.

	I. Jefferson County. Garvens. Mean of two analyses.	II. Newton County Williams
Calcic carbonate	98.680	97.962 per cent.
Magnesic carbonate	.945	1.853 " "
Ferrous carbonate	.047	253 " "
Insoluble matters	None	None
	99.672	100.068

DOLOMITE (Kirwan)

Is isomorphous with calcite and consists of lime and magnsia carbonates. In the southwestern deposits it is a very common gangue,

---

* For analysis of stalactites of cerussite from Brigham's diggings, Wisconsin, by J. D. Whitney, consult Dana, p. 701.

occurring either in densely crystalline masses, or loosely aggregated crystals, with curved faces, reddish brown on the surface, but colorless interiorily. The surface coloration is due to the change of ferrous carbonate (isomorphous with and more or less replacing calcic and magnesic carbonates in calcite and dolomite) into hydrated ferric oxide or limonite?) Chauvenet (*loc. cit. page* 396) finds two specimens of dolomite from Joplin to have a composition as under:

	I. Moon Diggings.	II. Murphy's Diggings.
Calcic carbonate	54.72	54.52
Magnesic carbonate	41.98	44.85
Ferrous carbonate	2.94	1.14
Insoluble matters	1.24	0.26
	100.88	100.75

Theoretically dolomite requires,

Calcic carbonate	54.35 per cent.
Magnesic carbonate	45.65 per cent.

So that both the specimens analyzed approach very closely the composition calculated from the formula.

Dolomite is one form of "soft tiff," so-called by the miners.

SIDERITE (Haidinger.)

Ferrous carbonate has not been noticed as a distinct species in any of the lead deposits. It is found replacing magnesic and calcic carbonates, as above, and occurs also dissolved (as acid ferrous carbonate) in mine waters.

BARITE (Karsten?)

Commonly called *heavy spar* or *barytes*, and by the lead miners of Missouri "bald tiff," or "soft tiff," is an important gangue in the deposits of the central and southeastern regions, subordinating in abundance in these, only calcite and dolomite. It has not been recognized in the deposits of the southwest, though I have found it in minute amount in a mine water from Oronogo, Jasper county.

By theory, barite contains: Baryta, 65.7 per cent.; sulphuric acid, 34.3 per cent. Two specimens of white, coarsely laminated variety, from Jefferson county, have been analyzed in this laboratory with the annexed results:

	I. W. C. Minger.	II. J. D. Greason.
Specific gravity at 4°C	4.3109	....
Baric sulphate	98.174	96.791
Strontic sulphate	0.500	1.436
Calcic sulphate	trace	0.325
Ferric oxide	0.176	0.191
Silica	0.898	1.200
Ignition, loss	0.454	0.148
	100.202	100.091

On Sec. 24, T. 36, R. 9, Phelps county, barite is found in a horizontal seam in mixture with galenite, and in a cave on the same section, rounded pebbles of the same mineral, coated with oxide of manganese, are found in a small stream.

Silica and Silicates.

The silicious minerals noticed in association with the lead ores, are silica itself, either as quartz, (rare,) quartzite, sand or chert (quite common); or silica in combination, as clay and calamine. The chert frequently contains galenite, in intercalated seams, filling small cracks, or as implanted crystals. A common form is that of a brecciated mass of fragments of chert, cemented by crystalline sphalerite, with small amounts of galenite intermixed. The chert passes, at times, into sand, white, or more or less colored by ferric oxide; sometimes slightly argillaceous, and occasionally impregnated with cerusite to a small amount. At other times it changes abruptly into a microcrystalline quartzite, sometimes of a light gray color, but generally of a dark brown or black color, and containing galenite and sphalerite.

Two chief varieties of clay are noticeable, and are locally distinguished as "tallow clays," and "red tough clays." The first is unctuous to the touch, slightly plastic, and on exposure to the air, cracks, and finally crumbles to a greasy powder. Crystals of galenite are not uncommon in this clay. The "red tough clay" has a much higher degree of plasticity. Its color is either yellow, brownish red or deep red.

Calamine.

Or hydrous zinc silicate occurs in large quantities, in association with the lead ores, and is of great value as a zinc ore. Its features and modes of occurrence will be described when treating of the zinc ores of the State. An analysis of a specimen from Granby, made by

Mr. Christian R. Winters, under my direction, gave the following results:

	I.	II.	Mean
Silicic acid	25.243	25.735	25.489
Zinc oxide	not est.	66.813	66.813
Ferric oxide	1.230	.999	1.114
Lime	trace.	trace.	trace.
Water	7.517	7.487	7.502
			100 918
Metallic zinc			53.62

D.—OTHER ASSOCIATES OF THE LEAD ORES.

Limonite, or the so-called brown hematite iron ore, is of general occurrence in the deposits, being a product of the alteration of pyrite or siderite. Pseudomorphs after pyrites are not uncommon.

Smithsonite (zinc carbonate) and metallurgically are of the most important ores of zinc, occurs very generally, but less abundantly than calamine. The lead deposits of Dade county yield large quantities. Two specimens from a shaft belonging to the Dade County Mining and Smelting Company, and taken at depths of twelve (I) and thirty-five feet (II) below the surface, gave in this laboratory:

	I. Pack.	II. Ohmann. Dumesnil-
Zincic carbonate	83.000	93.440
Ferrous carbonate	1.932	4.040
Calcic carbonate	0.803	1.525
Magnesic carbonate	0.861	0 454
Ferric oxide	3.046	0 800
Alumina	2.573	0.261
Silica	7.390	0.477
Water	0.165	......
	99.770	100.277
Metallic zinc	43.019	48.618

Azurite and Malachite (basic copper carbonates) are found in small amounts at many deposits, especially at those in Morgan, Miller and Cooper counties, resulting from the oxidation of chalcopyrite.

Goslarite (zinc sulphate) and Gypsum (hydrous lime sulphate) are found abundantly dissolved in the mine waters, and result from the oxidation of sphalerite and other sulphides, and from the action of free sulphuric acid thus produced, upon the lime carbonate.

Chalcanthite (copper sulphate) is also found in minute amounts in the same waters, and has a similar genesis.

Bitumen, either in the plastic or the liquid state, is found in the mines of Jasper county, in cavities in the limestone associated with sphalerite, galenite and calcite.

The chemical analysis of the water from the mines at Oronogo, Jasper county, is here introduced as throwing light on the genesis of the various ores and minerals found in the lead deposits of the State. The samples were collected on two consecutive days. The absence of lead sulphate is noticeable, none being detected, though careful search was made:

Analyses of Mine Water from Oronogo—Results in grains per gallon of 231 cubic inches.

	I. Williams.		II. Hare.	
Sodium chloride	0.16032	grains.	.18094	grains.
Sodium sulphate	0.49009	“	.48331	“
Potassium sulphate	0.31879	“	.33772	“
Calcic sulphate	27.98303	“	28 31665	“
Magnesic sulphate	2.74154	“	2.23635	“
Baric sulphate	0.08164	“	.07115	“
Zinc sulphate	13.14142	“	13.08099	“
Copper sulphate	0.01711	“	.01171	“
Alumina sulphate	0.58649	“	.69906	“
Calcic carbonate	4.58545	“	448954	“
Ferrous carbonate	2.29817	“	2.73376	“
Manganous carbonate	trace.		trace.	
Arsenious acid	0.11877	“	.15198	“
Antimonous acid	0.12232	“	.13022	“
Silicic acid	0.81645	“	.64033	“
Organic matters	4.14058	“	3.63904	“
Total solids	57.60315		57.18282	

The copper, arsenic and antimony were obtained by the treatment of four litres of water, and distinct traces of Cadmium were found in the same amount. The other constituents were estimated with the amounts ordinarily used in mineral water analysis, the method of Fresenius. [*Quantitative Chem. Anal.*, 4th English edition, page 506, *et seq.*,] being followed with such deviations as were found necessary from the presence of the extraordinary constituents.

GENERAL OUTLINE OF THE METALLURGY OF LEAD.

The extraction of lead from its ores is carried on in various forms of furnaces, referable, however, to three types, and giving rise to three chief methods, distinguished as

I—The Reverberatory, or air furnace methods.

II—The Blast furnace [shaft or cupola furnace] methods.

III—The Hearth methods.

In the second and third methods, the chemical changes in the materials to be treated are aided by a blast of air from auxillary blowing apparatus, while in the first the natural draught of the stack or chimney alone is relied on.

The Reverberatory and Hearth methods only find general employment in Missouri. The blast furnace methods proper being employed to a limited extent in the treatment of residues (or so called slags) from the other processes (Granby, Joplin and elsewhere) or for treatment of residues and nickeliferous ores for nickel mattes and lead, as at Mine la Motte.

I—*Reverberatory Treatment* is applied chiefly to high grade ores which have, as their associated gangues such minerals, the constituents of which do not readily unite with lead oxide to form fusible glasses or slags, but which tend rather on the contrary, to interfere with the fusion of the oxidized lead compounds formed in the smelting operations. Calcic carbonate and barite are therefore favorable gangues. The method is not usually applied to the treatment of ores with silicious gangues without the use of metallic iron as a desulphurant, and in such cases it is commonly supplanted by the blast cupola methods as the more economical mode of extracting the lead.

Two forms of reveberatory treatment are in use:

A—The method of reaction, as it is commonly called, but more properly, the method of *air reaction*, in which nearly all the lead is produced through the agency of atmospheric oxygen, a series of reactions between various oxidized lead compounds and lead sulphide being induced. It involves two principal stages, a first, or roasting stage, in which a part of the lead sulphide is united with oxygen to form lead sulphate (PbS becoming $PbSO_4$), or is changed into lead oxide (PbO); secondly, a reaction stage proper in which, under increased temperature, the oxygen compounds react on the unchanged sulphide, with the metallic lead and sulphurous acid as products. This stage may be better understood from the following equations commonly employed to represent the reactions taking place, though practically these reactions are much more complex than can be thus indicated:

$$PbSO_4+PbS= 2\,Pb+2SO_2 \text{ or}$$
$$2\,PbO+PbS= 3\,Pb+SO_2.$$

The reverberatory furnace consists essentially of a combustion chamber (*fire box*) communicating by means of an opening or flame passage with a second chamber (or hearth) in which the materials to be treated are brought under the influence of the heated gases and products of combustion from the fire box. The lower part of the hearth (*sole*) is separated from the fire box by a dam (or *fire-bridge*) and is surmounted by a low flat arch, continuous with the top of the fire box, and with the opening connecting the two chambers. One or more flues, at the end of the furnace furthest from the fire box carry the products of combustion and the volatile products of the reactions on the hearth to a stack or chimney common to several furnaces. Various openings (or *work doors*) give access to the fire box and to the hearth for purposes of introducing the charge and for manipulating it during the operations. Certain proportions between the various parts have been found best adapted to certain kinds of work, but these vary in different furnaces according to the purposes for which they are intended, and according to local condition, such as character of fuel and ore. Commonly, however, the length of the grate bars of the fire box is equal to the breadth of the hearth. The length of the hearth is from 1 to 2.5 times the breadth, and the height of the arch above the fire-bridge is usually from one-third to one-half that of the arch above the sole.*

The reverberatory reaction method has been variously modified in so far as the arrangement of the parts of the furnace and the details of operations are concerned, but three chief systems of conducting it are recognized and named from localities, furnishing typical examples. These three systems are:

a—The Carinthian.
b—The Flintshire or English, and
c—The French.

In all processes there remain on the sole of the reverberatory materials rich in lead, which give rise to a stage of treatment additional to the two already mentioned, having for its object the recovery of a further amount of lead. The features in conducting the reverberatory reaction method become, therefore, as follows:

First stage—period of oxidation. The ore to be treated is spread over the sole of the reverberatory and gradually brought to a low red heat. The charge is stirred in order that the oxidation may be as uniform as practicable, and the temperature is so controlled that the materials being treated may not be agglomerated or fused—conditions which would seriously operate against oxidation. During this period, lead oxide and lead sulphate are produced. The duration of the operation varys according to the arrangements of the furnace, and

---

* On these points consult Havrez, quoted in supplement to Crookes & Röhrig's translation of Kerl's Metallurgy.

according to the weight and grade of the charge. The presence of carbonate or other oxidized lead ores in greater or less amounts proportionally shortens the time required for oxidation. Under all circumstances the oxidation period is brought to a close before all the lead sulphide is changed. With regard to the time required, experience alone can determine for each special instance; in general, it may be stated that the time necessary for the first stage in the treatment of 450 lbs. of 65 to 72 per cent. galena ores with soft wood fuel at Bleiberg (Carinthia) was three hours, while in the English furnaces from 16 cwt. to 1 ton of 70 to 75 per cent. ore requires two hours (for *first firing*) with coal as fuel.

SECOND STAGE OR DEOXIDIZING PERIOD: The atmosphere in the furnace is rendered less oxidizing and the roasted materials are brought rapidly to a cherry-red heat. At about this temperature the reactions between the lead sulphate (or lead oxide) and the unchanged lead sulphide begin, resulting in the production of metallic lead and sulphurous acid. The lead runs down over the sole of the furnace and collects in a basin or sump in the sole, and is tapped thence into a kettle exterior to the furnace (Flintshire), or runs directly as reduced into the exterior iron kettle (Bleiberg-Carinthian). The residues of the Flintshire furnace are either pasty and are raked out (gray or drawn-slag) or in certain cases are melted and tapped, (run-slag) and are passed to other treatment; those from the Bleiberg are pasty and are treated in the same furnace.

During this stage frequent manipulations of the charge are necessary in order to bring about intimate contact of the lead sulphate (or oxide) and the lead sulphide. The operation is terminated when the lead ceases to flow. The temperature of the furnace is then somewhat lowered, and a roasting (or first stage) again brought about, and then again a second period, thus alternating the roastings and reactions, as long as lead flows and the residues have not passed into a state of liquid fusion. The length of time necessary is largely influenced by the care and skill of the workmen in the first or roasting stage.

The French method is different, in so far that the roasting stage is prolonged until the roasted mass is caked together, and drops of lead appear, and the second or deoxidizing period is characterized by the introduction of carbonaceous matters (wood and coal slack). The lead is collected in a sump. The process is adapted to ores more highly silicious than are usually deemed suitable for being worked by either the Bleiberg or the Flintshire method; but the consumption of fuel is greater, the yield is smaller, and the loss of lead greater.*

---

*Percy's *Metallurgy of Lead,* page 245; Crookes & Röhrig (*Kerl's*) *Metallurgy*, page 58.

Third or Supplementary Stage: The matters remaining on the sole of the furnace after the repetitions of the two proper stages, are still rich in lead, existing as oxide, sulphate, silicate and sulphide, but in such forms of combination or in such physical condition that further reactions among themselves are not possible by a continuance of the ordinary operation. To restore the physical condition necessary for the reactions between the lead sulphate and sulphide, and to reduce the oxidized compounds to the condition of metallic lead, lime or charcoal is introduced into the residue and the temperature of the furnace is raised. Charcoal is oxidized at the expense of a part of the oxygen lead compounds producing carbonic acid, which, escaping through the materials, renders them more porous and induces a condition favorable for further reaction between the remaining oxidized lead compounds and the lead sulphide. Lime acts mechanically, rendering the materials less fusible and more spongy, thereby allowing further reactions as above. There is also a possible chemical action of the lime, but on this point metallurgists are at variance.

The choice between charcoal and lime as reagents in the third period is influenced chiefly by the manner in which the roastings and reactions have been carried on, but is limited also by the fact of the ores being argentiferous or non-argentiferous—for silver bearing ores lime being preferable as rendering a cleaner slag, that is, one freer from silver. In the English furnaces lime is most commonly employed, while at Bleiberg the residues of two charges are treated with charcoal (*pressing*), resulting in the production of less pure lead (*press lead—Pressblei*).

As will be seen in the section devoted to the details of air-furnaces throughout this State, the supplementary stage is rarely resorted to. This is manifestly advantageous, so far as the production of a uniformly good quality of lead is concerned, but in all other respects is wasteful, and therefore to be condemned. At present the abundance of high grade ores may perhaps palliate this want of economy, but the treatment of these rich residues in the reverberatory must eventually become part of the method.

The residues from the reverberatory are, when rich enough, passed to treatment in slag furnaces by methods which are of the type of the blast or cupola furnaces, and will be comprehended from the descriptions given of these on subsequent pages. The Missouri furnaces, as before noted, usually disregard these residues. Occasionally, however, when conditions of transportation are favorable, they are sold to some of the few more complete establishments at which slag furnaces are run, and are there treated for a special brand or *slag lead.* Analyses of these residues will be found in the section in which the details of the air furnaces are given. Annexed are analyses of Flintshire *Gray Slag* for comparison:*

	Per cent.	Per cent
Lead sulphide	0.90	......
Lead sulphate	9.85	18.73
Lead oxide	48.87	38.08
Zinc oxide	7.52	12.00
Lime	12.68	13.50
Alumina	3.01	0.71
Ferric oxide	2.86	Ferrous oxide.. 3.89
Silicic acid (combined)	12.52	12.32
Residue	0.92	......
	99.66	99.23
Metallic lead	52.88	48.142

In addition to lead in the residues, there is a further source of loss of metal, arising from the volatility of lead and its compounds. This loss is inherent in all methods of lead separation, but varies greatly in degree in the different forms of treatment. At well arranged establishments means are provided for the condensation or collection of this volatilized lead matter or fume, by the introduction of a series of condensing chambers or other apparatus between the furnace and the stack; but these methods have not become of general use in this State where abundant and high grade ores and cheap raw fuel, coupled perhaps, with a laudable desire on the part of the smelter to produce an uniform grade of soft lead, have diverted attention from these apparently minor, but, in reality, important adjuncts to economical smelting and to the sanitary condition of operatives as well as of a neighborhood.

The composition of lead fume will be seen from the annexed analyses, I, being fume from the reverberatory at Granby, Newton county, by myself, and II, from reverberatory at Pontgibaud, by Rivot:

	I.	II.
Lead sulphate	76.750	39.00
Lead sulphide	.188	4.50
Lead carbonate	6 189	35.00
Lead oxide	8.371	Zinc oxide... 2.70
Ferric oxide	1.147	......
Antimony oxide	.903	.....
Arsenious oxide	trace.	1.50
Zinc chloride	049	......
Zinc sulphate	1.675	2.30

---

* Percy "*Metallurgy—Lead*," pages 235 and 239.

	I.	II.
Cadmium sulphate	trace.	......
Calcic sulphate	.923	......
Magnesic sulphate	.660	......
Insoluble residue	2.510	Silica and clay..13.20
	99.365	98.20
Per cent. metallic lead	65.156	58.253
Per cent. metallic zinc	0.676	3.09

The composition of the fume and its amount are, of course, largely influenced by the composition of the ore and the character of the treatment which the latter receives. The quantity is much greater than is realized by our smelters, being fully five per cent. of the fire assay value of the lead contained in the ore. Eilers, in a paper on the avoidable waste at American Lead Smelting Works,* estimates that the loss by fume and dust at the Utah Smelting Works, averages fully nine per cent. of the value of the original contents of the ore.

In the reverberatory air-reaction method, the presence of small amounts of blende—not exceeding twelve or fifteen per cent. of the charge—is regarded as advantageous, since it renders the mass, after oxidation, less fusible, and in better condition for frequent alternations of oxidations and deoxidations. On the other hand the presence of blende is disadvantageous, because it increases the loss of lead by volatilization, the specifically light zinc oxide which is produced in the roasting, assisting in carrying mechanically the denser lead compounds out from the furnace. The action of the zinc oxide in this direction may be inferred from the fact that zinc white or oxide, as produced directly from the ores is in such condition that an ordinary flour barrel will contain only from thirty to thirty-five pounds of the material.

If the ores are argentiferous, the loss of silver is greatly increased through similar action of zinc compounds.

The experiments of Malaguti and Durocher,* made with argentiferous zinc blende, indicate that the loss of silver in the presence of zinc compounds may be very considerable, reaching as much even as 70 per cent. of the original contents of the ore, but ranging usually between 15 and 60 per cent, according to the richness of ore in silver and zinc, and according to the management of the furnaces.†

Pyrite in the ore is beneficial in the first stage, aiding in the con-

---

* *Transactions Am. Inst. Mining Engineers*, Vol. III., pp. 98, *et seq.*

* *Annales des Mines*, XVII, 1850.

† In *Journal of Franklin Institute*, Feb., 1871, this matter has been discussed at some length by the writer of this report.

version of lead sulphide into lead sulphate and oxide. In the second stage, the iron oxide, formed by the oxidation of the pyrite, makes the charge less fusible, while any unoxidized pyrite decomposes lead sulphate, aiding in the separation of metallic lead. An excessive amount tends to the formation of matte, and thereby to loss of lead. If the pyrite is arsenical, the quality of the lead is impaired by the presence of arsenic. Chalcopyrite acts in a manner analogous to that indicated for pyrite, though smaller amounts work more injuriously than are required for the latter, since matte will be formed, and the quality of the lead will be injuriously affected by the presence of copper. Stibnite, and other antimonial minerals, even in small amounts (2—3 per cent.) are objectionable, causing caking of the charge, increased loss of lead and silver by volatilization, and impure lead.

Calcite, barite, siderite and fluorite (fluor spar), within certain limits, are either beneficial or not objectionable. Quartz, clay and other silicates in small amounts (1—6 per cent.) make a fusible charge preventing good roasting and interfering with the reactions.

In cases where argentiferous ores are treated, the lead produced in the earlier parts of the processes is richer than that of the later periods, as silver sulphide is more readily decomposed than lead sulphide.

B—*Method of desulphurization and precipitation by metallic iron:* This method is very limited in its application. It may be employed with silicious ores,* unsuited to method A, but has been largely replaced by blast furnace methods in the treatment of such ores.

The reaction, in theory, is represented by the equation:

$$PbS + Fe. = FeS + Pb.$$

In practice, three products (besides fume) are formed: slag, matte (iron sulphide with some lead sulphide) and lead. The amount of iron added never exceeds 35 per cent. of the weight of the charge. The operations are carried on in reverberatories of smaller dimensions than those used for method A, so that higher temperature may be obtained. The sole of the reverberatory is arranged with a sump, in which the three chief products collect in the order of their gravities, and are thence drawn off through the tap hole.

The method is expensive in fuel, labor, tools, and in the iron for the desulphurant. When argentiferous ores are treated there are

---

* Kerl's *Handbuch der Metallurgischen Hüttenkunde*, Vol. II, p. 89.

produced rich mattes and slags which require treatment in the blast furnace. The advantages it possesses over the blast furnace methods are found in the comparatively small cost of the furnace plant, and in the fact that raw fuel can be used.

II—BLAST FURNACE METHODS.

These are of very general utility and comprehend a great variety of details in furnace arrangements and in the manner of conducting the operations. In general the furnace is an upright shaft into which the materials subjected to treatment are charged along with the fuel (charcoal or coke) and into which is forced a current of air from one or more tuyeres. Formerly the furnaces were of rectangular section, somewhat pyramidal in elevation, narrowing toward the top and generally with a single tuyere. Within a few years radical changes have been made in the shape of the furnaces; the size has been greatly increased and the number of tuyeres has been multiplied. The improved lead-blast furnaces are in form of an inverted truncated cone, or an inverted truncated pyramid with a rectangular horizontal section with tuyeres on both the long sides, or with a hexagonal or octagonal section and tuyeres on all sides. With these changes have come increased effectivness of fuel and other important economical considerations. In all cases, whether the old forms or the improved forms be used, the cost of establishment is much more expensive than for the reverberatories. Nevertheless, for argentiferous ores, or for ores poor in both silver and lead, but with silicious gangues that cannot be separated by mechanical means, the blast furnaces or cupola processes are of great value, and necessarily of wide application. An extended range of modifications, the results of local experience with regard to the character of the ores and the available fuel, are to be observed in these methods, but they may be generally referred to two types of treatment:

A.—Direct fusion of the raw ore with iron as a desulphurant. (*Niederschlagsarbeit.*)

B.—Treatment of previously roasted ore with or without the addition of metallic iron. (*Roest-Reductions-Arbeit.*)

Method A is adapted to rich lead ores containing little silver, and only small amount of foreign sulphides, but associated with silicious gangues. Foreign sulphides make the mattes richer in lead and silver, or increase loss by volatilization, or make the charge of difficult fusion or finally impair the quality of the lead, this last not essentially peculiar to this method. Low grade ores require cheap iron and relatively high market rates for lead. Within certain limits, the higher the temperature the more complete the decomposition of the lead sulphide by the iron, and less lead sulphide unites with the ferrous sulphide in the matte. A certain quantity of slag-forming matters is necessary, as slag is required to prevent oxidation of lead in the furnace hearth, but the slag must be a fusible one to avoid increased consumption of fuel and greater loss by volatilization; at the same time it must not form at a temperature lower than that required for the desulphurization of the galenite and for the formation of mattes with the practicable minimum of lead.

Allowing cast iron to contain five per cent. of impurities, it will be found that one part of such iron will extract 3.64 parts of lead if ferrous sulphide ($Fe S$) is formed. Usually di-ferrous sulphide (Fe_2S) is produced. Less iron is necessary when the ores contain chalcopyrite and blende, and is also desirable to avoid decomposition of these sulphides. An increased amount of the iron is required when the ores are sensibly arsenical. Increased amount is prejudical towards the silver sulphide, throwing it into the matte.*

Method B. Ores containing so much foreign sulphides as to be unfitted for method A, or so rich in silicious gangues as to be unadapted to the reverberatory reaction; or argentiferous ores which cannot be concentrated mechanically without loss of silver, are suitable for treatment by this method. In some of its many modifications it is suited to a great majority of cases, especially in the treatment of the lead and silver bearing ores of Utah, Nevada and Colorado. The preliminary roasting does away largely with the use of iron, and is performed generally in reverberatories and is finished by raising the temperature sufficiently to agglomerate the roasted mass or better to fuse it. *Basic* ores containing much lime, etc., are better scorified by the addition of silica. Neutral ores, or those which contain in themselves the slag-forming constituents in the proper proportions, require no fluxes in the cupola, but *acid* ores or those containing quartz or silicious matter are fluxed by ferric oxide or by lime. Frequently a small amount of scrap iron or of forge scale is added to desulphurize any galena which may have escaped oxidation during the roasting. Production of matte is avoided as much as possible, unless the ores are notably copper-bearing and the saving of that metal is an object. The chief products, besides fume, are slag and lead.

---

* For illustrations of this process consult Kerl *loc. cit.*, pages 135 *et. seq*, or Crookes and Röhrigs translation, pages 98 *et. seq*; also Rivot *Metallurgie du Plomb et de l'Argent*, page 450. Further particulars would encumber this report unnecessarily.

The slags are compounds of silica (from the ores or from addition of acid fluxes) with lime, ferrous oxide and a small amount of lead oxide—the lime or iron oxide being added when necessary (in the case of acid ores) to take the place of the lead oxide in the slag, and prevent loss of lead. The most desirable slag is that which is either a singulo silicate (the oxygen of the bases: oxygen of the acid=1:1), or mixtures of a singulo silicate with a bisilicate (in which, oxygen of base: oxygen of acid=1:2). The singulo silicates are liable to enclose particles of metallic lead and thereby cause loss of metal; the bisilicates are not open to this objection, but require higher temperature for their formation, and are therefore more expensive in fuel, and cause greater loss by volatilization and greater reduction of foreign oxides. The bisilicates do not act so strongly on the furnace lining.

As noted above, the formation of matte is avoided and this constitutes one of the advantages of the preliminary roasting. When, however, it is necessarily produced, as in the case of silver and copper bearing ores, it is returned to the furnace or is treated by more elaborate processes for the separation of the valuable metals. In Utah and Nevada it is usually disregarded.

The proper fuel is charcoal or coke, or a mixture. The latter gives the best yield of metal; charcoal gives a lower heat, but one more uniformly distributed over a larger space than that of coke, which furnishes higher heat in a small space near the tuyeres. Besides providing under the influence of the blast the temperature necessary for the chemical changes and slag fusion, the carbon of the fuel is a deoxidizing agent for the lead compounds in the roasted ore. The carbon also reduces iron oxide, and the metallic iron thus formed is a desulphurant for lead sulphide that may be present. Excessive reduction of iron is to be avoided, for iron deposits are then formed in the hearth of the furnace, (*sows*, *bears*, *etc.*) Low cupola furnaces are best in this respect; high furnaces are more economical of fuel and avoid so great a loss of lead in fume.

The lead and slag (and matte when formed), are drawn off at intervals at the breast of the furnace by the ordinary process of tapping, or by means of the mechanical device of the syphon tap.

This form of treatment not being required for the Missouri ores, excepting in one instance, and offering no advantages over the reverberatory method so far as this State is concerned, it has not been considered necessary in this connection to dwell at any length on the cupola methods, nor, in fact, to give anything more than such an outline as is necessary to an intelligent comprehension of the subject of the metallurgy of lead. Those interested in it will find ample illustrations in the works of Kerl and Rivot, before quoted, and, for the more improved furnaces, in the reports of United States Commissioner Raymond, (especially for 1872, page 379; an excellent resumé of the subject adapted to the non-professional reader, and instructive to the professional metallurgist), in transaction American Institute of Mining Engineers, and in Church's *Notes of A Metallurgical Journey in Europe.*

III—HEARTH METHOD.

Hearth treatment is in fact an air re-action method, in kind like that described for the reverberatory, but with the difference that the oxidation and reduction go on simultaneously under the influence of a blast of air. Lead sulphate and lead oxide are produced by the air thrown in through the tuyeres, and act, at once, on the lead sulphide.

The Scotch-hearth has been replaced in this country by a modified apparatus generally known as the American-hearth. The latest, used in Missouri, is the American water-back hearth of somewhat larger size and with three tuyeres instead of one in the older form. The upper part is formed on three sides by a double-walled box of cast iron, technically known as the *tuyere-plate*, through which passes three tuyeres. A circulation of water is maintained through this iron box, and serves the double purpose of protecting the tuyeres and the castings. Back of the tuyere-plate is the *wind box* which receives the blast from the blowing apparatus. The hearth bottom or box is also of cast iron set in masonry which carries the upper iron parts of the hearth. The hearth-box contains the molten lead or lead bath upon which the charge floats and upon which the operations of smelting are performed. The work stone or table, also of cast iron, slopes downwards and forwards from the front top of the hearth-box toward the iron pot into which the lead from the reduced ore passes and accumulates. This pot is built over a small fire place, the fire in which keeps the lead in a fluid condition for ladling into the moulds. The apparatus is surmounted with a hood for carrying off the fumes and the products of combustion.

The walls of the tuyere-plate are commonly about one and a half inches thickness. The nozzles of the tuyeres through which the blast reaches the charges, are from one to one and a half inches in diameter. The tuyeres enter through the back wall of the hearth at a height of from one three inches above the surface of the lead-bath contained in the box and constituting as before stated, the sole upon which the charges are worked.* They are about six inches apart from each other, and the two outer ones of the system are about the same distance from their respective sides, so that the back of the tuyere-plate is usually about 28 inches. Its height is 16 inches. The sides are 1 foot 10½ inches from back to front, for the inside walls, and 2 feet 5½ inches for the outside walls. From inner to outer wall, outside measurement, 7½ inches; space between walls, 4½ inches.

The hearth is operated by first kindling a wood fire upon the surface of the lead in the hearth box; on this fine charcoal is thrown, well ignited and the blast turned on. Some residues or so-called slags, from the previous smelting are thrown on this fire, and, as soon as the lead in the bath is properly melted, the galena ore, crushed to the size of a pea, is thrown on in charges of about twenty pounds at a time. A little caustic lime is added as a flux, the mass covered with fresh charcoal and the whole allowed to remain undisturbed for about five minutes. At the end of this time, thorough contact of charcoal and ore is brought about by continual stirring accompanied by breaking so that the blast may have passage and full oxidizing effect. These manipulations are continued for some fifteen minutes when a new charge of twenty pounds of ore is added. Lead soon begins to flow over the inclined work-table into the kettle whence it is ladled or run by syphon into the moulds.

A shift of eight hours will treat about 3000 pounds high grade ore, yielding by the work, about 68 per cent. of metallic lead, and requires two workmen. At the end of this time the metallic bath is cleared of residue for the next shift; the so-called slags are sorted and that which is rich enough is crushed and washed for separate treatment in the same hearth, yielding about 30 per cent. lead. The poor residues and the residue from the treatment of the rich slag are either disregarded, or in a few cases, passed to further treatment in the slag furnace. About thirteen bushels charcoal ore used per shaft or for the treatment of 3000 pounds of ore. Both ore and charcoal are delivered by laborers to the smelters, and as the laborers can readily serve several hearths, a reduction of cost of working is always effected by having a number of hearths side by side.

Under the conditions surrounding these operations, there is

---

* "It may be asked why should the contents of the ore hearth be kept floating on the melted lead? The answer is that if the bottom were formed of brick or other solid refractory material, inconvenience might be caused by accretion of slaggy matter, whereby the level would be changed and the working of the furnace deranged. A bottom of iron would be rapidly acted upon. Let the reader try to suggest a substitute for the simple and usual hearth-box filled with molten lead, and he will probably be puzzled." Percy, *Metallurgy of Lead, 289.*

necessarily a considerable loss of lead by volatilization. The scoriaceous character of the residue make it also rich in unchanged ore and shot lead. The hearths require, therefore, high grade ores, and have the further disadvantage of being a costly form of apparatus. On the other hand, they will treat a much greater amount of galena in a given time than will the usual reverberatories in the same time. With a single American-Scotch hearth the cost of production of 1000 pounds of lead is about $5, and for the production of the same amount from a reverberatory under equal conditions, the expense is $4.90 independently in each case of the cost of the plant. By increase of number of hearths, the expenses of operating are decreased much more rapidly than by an increase in the number of reverberatories.

There are few details from which calculation can be made of the amount of loss by volatilization in the hearth methods in this country. In 1849 an experimental ore hearth, similar in principle to that formerly worked at the Rossie (New York) Lead Mines, and in which wood was used for fuel, was erected at Bleiberg, in Carinthia, and during that year and the years 1850 and 1851, the following results in regard to the loss, other than that caused by the slag, were obtained:*

Year.	Average per cent. lead in ore.	Per centage loss of lead, exclusive of slag.
1849	71.52	10.42
1850	71.70	9.74
1851	70.27	11.38

The slag from the hearth cannot be homogeneous; analyses will be found in the part of this report given to the illustrations of the Missouri hearth furnaces. The following is Plattner's analysis of the slag produced in the above experiments at Bleiberg:

Silica	5.260	per cent.
Sulphuric acid	5.038	per cent.
Lead oxide	37.710	per cent.
Ferric oxide	19.500	per cent.
Zinc oxide	19.200	per cent.
Molybdic acid	0.460	per cent.
Lime	8.856	per cent.
Magnesia and manganous oxide	1.417	per cent.
Alumina with traces potash, soda and copper oxide	1.760	per cent.
	99.201	

At Przibram, Bohemia, the American hearth was tried with ore yielding, by assay, 74.88 per cent. lead; the yield, by the treatment,

---

*Percy, Metallurgy of Lead, page 292.

was 60.88 per cent., not including the lead subsequently extracted from the slag: the slag amounted to 45.94 per cent. of the ore put in treatment. The reported produce of the residue is enormous when compared with the experiences in this country.

PURIFICATION OF LEAD.

The purity of the resulting lead, whatever the process employed in its production, depends, as before mentioned, largely on the purity of the ore smelted. In cases where argentiferous ores have been run through any of the methods the *work lead* produced is refined when necessary, and the silver, if small in amount but still great enough to be extracted, concentrated (Pattisonized) and finally separated from the lead by cupellation, or by treatment with zinc and subsequent separation of the zinc and silver and purification of the desilverized lead. Even when not sufficiently rich in silver to warrant attempt at separation of the two metals, the lead in most cases requires *refining*, *improving* or *softening*. The object of these practices is to remove or to lessen the amount of the foreign metals, copper, zinc, iron, antimony, arsenic, nickel and bismuth, which render the lead hard and sonorous and unfit for many of its industrial applications.

Arsenic, antimony, bismuth and copper are the most objectionable, the first two being prejudicial in almost all uses (excepting for special alloys as shot metal and type metal), but are more so towards the use of lead for mechanical purposes than for white lead manufacture; the reverse is true in general for copper, for it is not so objectionable in lead intended for mechanical purposes as for corrosion into white lead. Zinc rarely exists in serious amounts; one and a half being the highest per centage of this metal possible in lead*; iron in notable quantities, passes into the lead only when the latter has remained in a melted condition in contact with iron.

The methods of softening in common use are:

a—By process of atmospheric oxidation, the lead being melted and oxidized on the surface, the oxidation of a part of the lead inducing a more rapid and more thorough oxidation of most of the foreign metals, especially of antimony. The dross which is formed is removed, a fresh surface oxidized and these operations repeated until the lead has reached the required degree of softness. The time required, and consequently the proportion of lead oxidized depends on the quality of lead, on the temperature and on the amount of surface of melted lead exposed to the air. The operation may be carried on

---

*According to Mathiessen and von Bosse.

either in a common reverberatory, or in one in which the ordinary sole has been replaced by an iron pan, or in one into which an artificial blast or current of air is driven. [At Pontgibaud, where a cast iron bottom was used, the charge was about 20 tons hard lead, the time required was three days (including charging and discharging the furnace), and the amount of soft lead produced was 94.7 per cent. of the original lead.*] According to Richardson† the results of treatment of "many hundreds of tons of hard lead," gave the annexed results:

Good Spanish hard lead	93.2 per cent. soft lead.
Hard lead from crystalized dross	90.3 per cent. soft lead.
English slag lead	87.1 per cent. soft lead.
Slag lead from Spanish dross	66.9 per cent. soft lead.
Slag lead from refuse products	67.3 per cent. soft lead.
Chinese tea lead (containing tin)	76.4 per cent. soft lead.

b—By a process of oxidation brought about by the addition of oxidizing agents. [Baker's method, with acid sodic sulphate and sodic or potassic nitrate; nitre, soda ash and lime in method of Pontifex & Glassford. In this latter the mixtures recommended are, 3 parts sodic nitrate, 4 parts soda ash, and 4 parts of burned lime for lead containing from 5 to 15 per cent. antimony; and equal parts of each reagent in cases of lead with less than 5 per cent. antimony. The furnace employed is the ordinary calcining pan-furnace, and about 55 lbs. of the mixture are required for 9 or 10 tons of the lead for a twenty-four hour's operation.]

c—By skimming with boards (England), or with iron scrapers (Germany), the dross which forms on melted lead. [Applicable to the purer leads, containing little copper.]

d—*Poling* or stirring the melted lead with a stick of green wood. [Applicable to rather pure leads, removing largely the antimony and copper at a great saving in labor, time, fuel and lead over the reverberatory methods of working.] This method is of very general application throughout Missouri, the leads being produced from pure ores by either the reverberatory or hearth furnaces. It is also used on slag leads which are not so pure as leads produced directly from the ores. (See results of analyses of Missouri leads tabulated in chapter III.)

The dross which is incidental to these or other processes of softening may be treated by various methods (preferably in cupola furnaces), producing an impure or *hard* lead, which is either sold as such

---

*Phillips *Mining and Metallurgy of Gold and Silver*, pp. 482, *et seq*.

† *Watt's Dict. of Chemistry*—Art.—Lead by Richardson, Vol. III, p, 518.

or purified, and slag. At the furnaces in this State the dross is either disregarded or passed to treatment with residues, or, without consideration of the effect on the lead, occasionally returned and treated with original ore.

The treatment of slags or residues more properly belongs to a class of blast or shaft furnace methods—the common form of apparatus being a low shaft with a simgle tuyere being of the type known as *Krumofen.* The method as modified at the Missouri furnaces is described at some length in the chapter devoted to the illustrations furnished by the furnaces of this State. In general terms it consists of the reduction of the lead compounds through the action of carbon either as coke or charcoal, saving as much lead as is practicable, but the greater part remaining in combination with the silica as a true slag. The substitution of a cheap base (lime or iron oxide) for the lead oxide in the slags and the recovery of the greater part of the lead is not attempted as in the shaft furnace methods proper.

There are economical considerations in favor of this wastage of lead under the conditions which attend lead smelting in this State, and the pecuniary loss by it is, perhaps, more apparent than real.

In consequence of the fact that the residues contain a considerable concentration of some of the foreign metals of the ore, and as a further consequent of the inherent features of shaft furnace methods, slag leads are always harder than these produced from the ore directly either by the reverberatory or the hearth treatment. They are softened or refined, to some degree, by partial oxidation in the reverberatories, according to the plan above given. They do not give results as satisfactory when used for chemical purposes, such as for corrosion into white lead, but for many mechanical purposes they serve equally well.

CHAPTER II.

ILLUSTRATIONS OF LEAD SMELTING IN MISSOURI.

The illustrations of the lead smelting operations which we have been able to collect comprehend data from most of the establishments in the State. They include in general, details of the form and size of the furnaces; weight of charge and time of elaboration; consumption of fuel and necessary amount of labor. The pecuniary outlay, whether necessary for the establishment of the works or for the cost of treatment, has not been entered upon; nor would such a course have been desirable or of any benefit when comparison will have to be made of the results reached by the different furnaces. In fact such a course would tend rather to confuse and to give erroneous ideas, since it could not reach entire accuracy.

The result obtained by the several furnaces have been illustrated by a full line of chemical work, including analyses of the various products. The ores have not been assayed to determine their value in lead, because average samples could not be obtained, and results of specimens would have been more than useless. No estimations, by chemical processes, of the value of the ore put in treatment, are made at the furnaces.

The reverberatory treatment will be first considered and subsequently that of the ore-hearths. The section on slag treatment includes nearly all the establishments at which it is practiced. It will be found of interest, in view of the great mystery with which it has been surrounded by the lead smelters of the State.

REVERBERATORY METHOD IN MISSOURI.

Southwestern Lead District—(I). The Granby Mining and Smelting Companies chief establishment is situated at Granby, Newton county, at a distance of one and three-fourths miles from Granby Station, Atlantic and Pacific Railroad. A switch road connects the station and furnaces.

The ores are furnished by the Granby mines in the vicinity of the furnaces, as well as by various diggings at Joplin and Oronogo, in Jasper county, and are usually bought from individuals working, on leases, the lands of company. Preparations are, however, being made to smelt at Joplin and at Oronogo the ores from the company's estate in the vicinity of these places, a plan which in its realization must be of great benefit on the score of economy.

The preliminary operations of crushing are done by a Blake's breaker and a pair of smooth faced rolls. The concentrating machinery for such ores as require it, consists of an eccentric jig, a Cazin's patent one plunger and a sliding lever jig.

The furnaces include, besides six hearths, and two furnaces for slag treatment, four reverberatories of which two are of the ordinary pattern in use throughout the State (the common *air furnace*), the remaining two being larger and modelled on the plan of the English or Flintshire lead furnaces. The latter have been in operation within the last two years, and are of the first of the pattern started in the State, as far as can be ascertained.

(a) Flintshire Practice: The hearths are 10 feet long by 8½ feet width; the soles being formed from slag from the furnaces well agglomerated by strong heating, and shaped into the ordinary sump near the middle door on the front or working side of the furnace. The bridge is one foot in height above the sole at that point. The fire-box is five feet long by three feet wide. The charging is done through the common hopper arrangement in the arch at a point opposite the work door nearest the fire bridge; the charge is manipulated through the six work doors, three on each side. The tap hole opens from the sump, though the wall of the furnace below the middle door on the front side and connects with the exterior iron kettle from which the lead is ladled into moulds.

The furnaces run on a great variety of ores, of which no valuation is made by preliminary assays. The weights of the charges are, therefore, very variable. For "number one *block*" mineral it is generally about 1500 pounds worked off in twelve hours; for "number one *wash* mineral," the same amount elaborated in about fourteen hours; for carbonate or "dry bone" mineral, one thousand pounds in six hours, and for "chats," (*i. e.*, galenite on chert or dolomite, which is calcined at a low heat to make the rock more easily broken by hand, preparatory to hand sorting or rough washing), 1000 pounds worked in ten hours.

The charge is spread over the sole and the roasting done at a low temperature, the time rarely exceeding two hours, or till the charge begins to agglomerate. When the roasting is deemed finished the fire is urged and the lead runs into the sump, the tap hole being closed with a plug of lime. When the charge has been No. 1 block mineral, the tapping is done at the end of ten and a half hours after charging. Before tapping, the matters on the hearth are thickened by the addition of wood ashes and charcoal and filled up around the sides of the sole, to allow the lead to drain into the sump. After well draining the plug of the tap hole is withdrawn and the lead runs into the kettle, where it is polled and then ladled into the moulds.

After tapping the fire is urged so that more lead may be drained from the residues. This metal is tapped from the sump as before; the residue or gray slag is drawn, the sole of the furnace is repaired, if necessary, and a fresh charge is introduced.

Two hands are required at a furnace, working shifts of twelve hours in the winter season or eight hours during the summer. Each furnace consumes six cords of wood during twenty-four hours, or forty-two cords per week, representing the treatment of 35,000 pounds of average mineral, with a furnace yield of about sixty-six per cent. or 23,000 pounds polled lead. The amount of wood required is therefore 2.4 cords per ton of 2000 pounds of mineral; the labor for the same amount of ore is two days (with average postes).

The so-called slags or residues are passed to the slag-furnace for further treatment, but as they do not receive there a separate operation, but are run in mixture with the residues from the other reverberatories and the hearths, it is impossible to learn anything respecting their yield of metallic lead, nor of their quantity produced from a given weight of charge. They are not homogeneous, so that chemical analyses will show a varying per centage of lead. A specimen representing the residue from a run of block mineral in the furnace, gave by mechanical separation, 1.02 per cent. of mechanically enclosed or "shot" lead, and after the removal of this the residue showed the following result on chemical analysis—the lead sulphate being determined in this as well as in all other analyses of slags, by solution in sodium hyposulphite:

Silica	12.557	per cent.
Lime	0.642	per cent.
Magnesia	trace	
Ferric oxide	3.220	per cent.
Alumina	0.642	per cent.
Antimony oxide	0.170	per cent.
Zinc oxide	2.264	per cent.
Lead sulphate	2.509	per cent.
Lead sulphide	22.084	per cent.
Lead oxide, by difference	55.939	per cent.
	100.000	

A direct determination of the metallic lead, by humid analysis, gave, including the mechanically enclosed metal, 73.703 per cent. A comparison of this result with that of Percy's analysis of the gray slag from the furnace near Holywell, Flintshire, cited in the preceding chapter, (page 30), and the large amount of unchanged sulphide (22 per cent.) would seem to warrant the conclusion that the roasting period has been too short, with too high a temperature. The charge has evidently agglomerated before the maximum possible of lead has been worked from it, and a setting up stage might have supplemented the routine work with advantage in the yield of lead.

The composition of the pig lead from the same charge—the sample representing ten pigs—is given below. In this connection, it may be stated that the scheme of analysis followed for this and the succeeding estimations of the qualities of the Missouri leads is substantially that of Fresenius * for soft leads, with the exception that the silver was commonly estimated by cupellation instead of by the method of precipitation as chloride:

GRANBY (FLINTSHIRE) LEAD.

Arsenic	0.01640	per cent.
Antimony	0.00077	per cent.
Silver	0.00029	per cent.
Copper	0.01210	per cent.
Iron	0.01711	per cent.
Zinc	0.00066	per cent.
Nickel	a trace	
Lead, by difference	99.95267	per cent.
	100.00000	

(b) *Treatment in the ordinary Air Furnace:* Two of this form are in use at the Granby works. Each receives an average charge of 1,500 lbs. of No. 1 ore, worked off in twelve hours, equivalent to 42,000 pounds of mineral per week as the aggregate work of these furnaces. During the week there are consumed twenty-one cords of wood. The yield of similar ore is about the same in both forms of reverberatory. The wear and tear of the common form is claimed to be somewhat greater than that of the Flintshire, and hence the campaign is shorter. On the other hand the cost of the furnace is much less than that of the new form—being usually about five hundred dollars per furnace, including sheds—so that the cost of treatment is not so much swelled by the item of interest on investment.

The following are the dimensions of the common reverberatory at Granby. The form described is usual throughout the lead regions of the State, the difference being in size and in the relative dimensions of the parts:

Length of hearth	9 feet.
Width of hearth at fire bridge	3 " 2 inches.

---

* Jahresbericht für Chemie, 1870.

Length of fire bridge	2	" 6 inches.
Thickness of fire bridge at top	0	" 8 inches.
Thickness of fire bridge at base	2	" 6 inches.
Width of fire box	2	" 6 inches.
Length of fire box	5	" 6 inches.
Inclination of sole	9½ °	
Height of arch above sole at fire bridge	1	" 2 inches.
Height of arch at side opposite fire bridge	1	"

The fire box is an L attachment to the furnace, and enters the hearth at a distance of thirteen inches from the front or pot end of the furnace. The flue is 8 x 12 inches in section. The charging or working door is at the upper or flue end of the hearth, and the discharging and cleaning door is at the lower end. The sole is a cast-iron plate with six inches depth of slag or residue melted upon it. The corners at the pot end are rounded, for facility in reaching the charge and cleaning the furnace.

The charge, crushed to size of a pea, is thrown in through the door at the upper end and pushed down evenly over the hearth. A low fire is kept up and the charge roasted during one or one and a half hours, varying according to the amount of carbonate and the richness of the mineral. During the roasting the charge is constantly stirred, so that all parts may be worked, in turn, down the sole towards the fire bridge or hottest part of the hearth.

At the end of the roasting period, the heat is increased; the reactions, already described, are set up and the lead flows down the hearth, through the discharge door into the kettle at the front end of the furnace. Should the materials begin to enter into fusion, they are thickened or stiffened by the addition of wood ashes. After the lapse of from nine to twelve hours, the lead ceases to flow; the residue is drawn and the somewhat cooled furnace is ready for another charge.

It will be noticed that the process differs from that described for the Flintshire furnaces at the same establishment, in that the lead flows at once into the kettle, and that no second firing is given. In many respects the method is allied to that practiced at Bleiberg, but it differs from this last in the size of the furnace and in the details of the operation.

The residues are not homogeneous, so there are difficulties in arriving at even an approximation to a knowledge of their average composition. A sample yielded 3.52 per cent. of metallic lead mechanically held in the slag, and after removing this, had the annexed composition:

Silica	21.396 per cent.
Lime	4.650 per cent.
Magnesia	3.948 per cent.

Ferric oxide	3.680	per cent.
Alumina	0.152	per cent.
Zinc oxide	7.146	per cent.
Lead sulphate	2.349	per cent.
Lead sulphide	20.929	per cent.
Lead oxide	34.914	per cent.
	99.063	
Total metallic lead	54.82	per cent.

The polled pig lead gave, by analysis of a sample representing sixteen pigs, the following result:

Arsenic	0.01122	per cent.
Antimony	0.00077	per cent.
Silver	0.00080	per cent.
Copper	0.05091	per cent.
Iron	0.01582	per cent.
Zinc	0.00090	per cent.
Nickel	0.00281	per cent.
Lead, by difference	99.91777	per cent.
	100.0000	

In the year 1875, the Granby company received weekly from the mines at Joplin about 79,000 pounds of mineral, all smelted as before stated, in the furnaces and hearths at Granby. The weekly receipts, at the same time from the mines at Oronogo, were about 60,000 pounds. The three hearths at Oronogo, which had been idle for a long time, were to be put in operation by the close of the year 1875.

The Webb diggings, three miles south from Oronogo, were yielding 50,000 pounds of mineral each week.

(2.) *Reverberatory Treatment at Joplin, Jasper county.*—At the works of Messrs. Davis & Murphy, six reverberatories of the ordinary pattern are run, treating weekly about one hundred and fifty thousand pounds of mineral. A charge of somewhat less than eighteen hundred pounds is therefore elaborated in each twelve hour shift in each furnace. No information could be obtained respecting the consumption of fuel. The residues together with those from other reverberatories and hearths in the district are treated in a slag furnace similar to that used elsewhere in the State, and which is described when treating of slag smelting.

The same firm has one other air furnace at Bently Diggings, five miles west of Joplin, and a second other, seven miles east from Joplin, at Scotland. The latter is run on ores from the Birch and the Grove Creek Diggings—the former, one and a half miles, and the latter, one-fourth mile from the furnace site. Fifteen hundred pounds of ores, yielding one thousand pounds of metal or 66⅔ per cent.,* are

---

*Schmidt, *Broadhead's Report*, p. 500.

worked off in a nine hours shift, presumably with the ordinary amount of labor, but with what amount of fuel we are not able to state.

The following are the receipts and produce of the ore for three years:

	Ore—pounds.	Lead—pounds.
1872	2,060,800	1,344,000
1873	6,061,000	4,074,000
1874	6,784,000	4,536,000

giving an average yield of 66.7 per cent, on the supposition that no slag lead is included in the amount of metal produced.

At Lone Elm near Joplin, Messrs. Moffett & Sergeant, run one air furnace of common size, treating fifteen hundred pounds of high grade ore per eight hour shift. The furnace runs actively; forty-five hundred pounds of mineral being run through in twenty-four hours with the consumption of two and a half cords of wood. Calculated to the ton of ore put in treatment, the fuel required is 1.11 cord of wood, and the labor is 2.6 days of eight hour posts.

The residue resembles those generally produced, and gives, on analysis, as follows:

Silica	18.657	per cent.
Lime	3.789	per cent.
Magnesia	trace.	
Ferric oxide	2.147	per cent.
Alumina	2.350	per cent.
Antimony	trace.	
Zinc oxide	11.650	per cent,
Lead sulphide	8.340	per cent.
Lead sulphate	2.741	per cent.
Lead oxide	50.088	per cent.
	99.862	

The sample, which was selected with all the care practicable, gave also 3.86 per cent. of metallic lead mechanically enclosed. The lead in combination amounts to 53.35 per cent., and the total lead is, therefore, 57.21 per cent. These slags are sold for treatment in the slag furnace of Messrs. Davis and Murphy, and their market rate is about $21 per ton, against $18 per ton paid to the same establishment for its hearth residues.

The pig lead from the reverberatory is not kept distinct from the metal produced by the other mode of smelting, but all is refined by poling in an iron kettle. The kettle holds 12,000 pounds of lead; the labor of two men is required in casting about 450 pigs of the polled lead in twenty-four hours, and the loss in refining is said not to exceed one per cent. of the raw lead introduced. I have no analysis of this lead more recent than that published in February, 1875. The result of which is annexed:

Arsenic	.01189	per cent.
Antimony	.08168	per cent.
Copper	.00409	per cent.
Iron	.03650	per cent.
Zinc	.15910	per cent.
Silver	.00994	per cent.
Sulphur	trace.	
Nickel	trace.	
Lead, by difference	99.69680	per cent.
	100.00000	

The refined lead is chiefly sold to Messrs. John T. Lewis & Bros. of Philadelphia, by whom it is used for white lead corrosion, and I am informed by the members of this well established firm that the corrosions are very satisfactory, in respect to both quantity and to the quality of the resulting white lead. It is extremely probable that the amounts of zinc and of some of the other metals in the above result are in excess of what is usual in this lead, at least more recent analyses are reported to show such to be the case. The Lone Elm is being largely substituted for the Tarnowitz (Silesian) lead at the Philadelphia establishment.*

The Lone Elm Company received in the year ending August 1, 1875, 6,500,000 pounds of mineral.

The *Joplin Mining and Smelting Company* operates one reverberatory treating fifteen hundred pounds of average ores in each twelve hours shift. The residues are sold to Messrs. Davis & Murphy. The receipts of mineral for the last three years were about 15,000,000 pounds.

Messrs. Pichar & Brothers have three air-furnaces of the common form, each treating twelve hundred pounds of high grade ores in one eight hour shift, with an average consumption of 0.9 cord of wood, equivalent to 1.5 cords per ton of two thousand pounds of mineral. The firm receives about fifty tons mineral per week.

(3.) *The Dade County Mining and Smelting Company*, with works at Corry, Dade county, began smelting in May, 1875, running one reverberatory until December of that year, when a second and

---

*Arrangements have been completed for the collection of fume from the Lone Elm Furnaces. The collected material is utilized for pigments, and the results are reported to be very satisfactory.

similar one was completed and put in operation. Each furnace works, usually, twelve hundred pound charges in an eight hour shift, but W. M. Taggart, the president of the company, writes us that twenty-four charges have been run through one furnace in six days. According to the same gentleman, the consumption of fuel is 0.4 cord of wood per charge, or 0.66 cord per ton. The residues have been analyzed and show the composition given below:

Silica	23.678
Lime	12,735
Magnesia	4.321
Ferric oxide	9.134
Alumina	3.366
Zinc oxide	1.635
Lead sulphide	15.559
Lead sulphate	2.504
Lead oxide, by difference	27.068
	100.000

The sample analyzed contained no traces of arsenic, antimony or copper, and gave no mechanically enclosed or shot lead. The lead compounds named contain metal equal to 40.494 per cent. of the slag.

From the date of inaugurating the enterprise up to January 9, 1876, the company had shipped 297,797 pounds or 3,590 pigs of lead. A specimen of the metal forwarded us by the president, showed the following composition:

Arsenic	Trace
Antimony	Trace
Silver	.00177
Copper	.00448
Zinc	None
Nickel	None
Iron	.00128
Lead, by difference	99.99247
	100.00000

(4) The other reverberatories treating ores from the southwestern region are: 3, belonging to S. B. Corn, near Joplin; one at Baxter Springs, Kansas, belonging to Messrs. Sarver & Co.; one at Brookline, Greene county, and one belonging to the Conoley Mining and Smelting Company in Christian county.

B—REVERBERATORIES OF THE CENTRAL REGION.

(1) The *Pioneer*, located at Pratt's Mill, (section 26, township 43, range 14, west), Cole county, uses ores from the Dawson & Eaton, and the Bacon, Metler & Co. diggings, near Decatur, Cole county, and from the Long's diggings, Miller county.

The reverberatory, which was built in 1871, is of the ordinary form, with the hearth 7½ feet by 4 feet. The fire-box is 5 feet long and 22 inches wide. The fire brick lining is of bricks from the Cheltenham works, near St. Louis. The furnace runs usually about four months in the year.

A charge of 1,800 pounds high grade mineral is elaborated in a twelve hour turn, by the work of one smelter and one helper, and with the consumption of three-quarters of a cord of wood, or at the rate of 0.83 cord per two thousand pounds. The residues contain:

Silica (with traces baric sulphate)	24.233 per cent.
Lime	7.460 per cent.
Magnesia	1.548 per cent.
Ferric oxide	4.920 per cent.
Alumina	1.437 per cent.
Antimony oxide	0.351 per cent.
Arsenic	trace.
Zinc oxide	1.454 per cent.
Lead sulphate	2.452 per cent.
Lead sulphide	35.730 per cent.
Lead oxide, by difference	20.414 per cent.
	100.000
Total metallic lead	51.081 per cent.

Most of the lead-oxide is in combination with the silica; this and the large amount of unchanged sulphide shown by the analysis would seem to indicate, that the time of roasting has been too hurried. As far as yield of metal is concerned, a setting up and second firing would have been advantageous.

The lead is of excellent quality. The result of its analysis is as follows:

Arsenic	trace.
Antimony	0.00347
Silver	0.00105
Copper	0.00946
Iron	0.00496
Zinc	0.00517
Nickel	0.00125
Lead, by difference	99.97464
	100.00000

The brand of the metal is "ONE."

(2) The *Eagle Furnace* (Sec. 23, T. 43, R. 14, W.,) near Pratt's Mill, Cole county, works ore from the Eureka and Scott diggings, on the same section. The hearth is 9 feet 8 inches by 3 feet 8 inches (greatest width); the fire chamber 5 feet by 2 feet. The mineral is

usually mixed with some carbonate ore; 1,500 pounds are charged at a time and worked off in twelve hours, with about the same relative consumption of fuel given for the Pioneer furnace. The residues are not treated further; they contain:

Silica	7.132 per cent.
Lime	2.939 per cent.
Magnesia	.336 per cent.
Ferric oxide	1.052 per cent.
Alumina	.699 per cent.
Antimony oxide	trace.
Zinc oxide	.945 per cent.
Lead sulphide	18.686 per cent.
Lead sulphate	1.855 per cent.
Lead oxide	65.505 per cent.
	99.149 per cent.
Lead-shot in original slag	0.10 per cent.
Metallic lead in combination	78.31 per cent.
Total lead in residue	78.41 per cent.

The pig lead gave the following result on analysis:

Arsenic	0.00313 per cent.
Antimony	trace.
Silver	0.00029 per cent.
Copper	0 00820 per cent.
Iron	0.00461 per cent.
Nickel	trace.
Zinc	0.00057 per cent.
Lead, by difference	99.98320 per cent.
	100.00000

(3) The *Gum Spring* furnace is located in S. E. ¼ of Sec. 8, T. 42, R. 17, W., Morgan county, and is operated by the Jackson company. The ore worked is obtained exclusively from the company's diggings, and is entirely galenite mineral. Two charges, each of 1,500 pounds, are worked off in 24 hours with the consumption of 1¼ cords of wood, or 0.83 cord for a ton of mineral.

(4) *Buffalo* furnace, Sec. 1, T. 41, R. 19, W., Morgan county, operated by Hon. G. Stover of Versailles, Morgan county; runs on ore from diggings belonging to the furnace owner, situated on the same section. A charge is 1,500 pounds, two to three charges being worked off in 24 hours. The furnace was built in January, 1874. No dimensions or further particulars were obtained regarding this, or the preceding establishment.

The residues contain as under:

Silica	6.359 per cent.
Lime	5.625 per cent.

Magnesia	.855 per cent.
Alumina Ferric oxide	1.875 per cent.
Antimony oxide	.075 per cent.
Arsenic oxide	trace.
Zinc oxide	1.199 per cent.
Lead sulphide	11.796 per cent.
Lead sulphate	2.703 per cent.
Lead oxide, by difference	69.513 per cent.
	100.000 per cent.
Total metallic lead	76.588

The brand of pig lead produced at the Buffalo is among the very best examined. It shows the composition given below:

Arsenic	trace.
Antimony	0.00281 per cent.
Silver	0.00114 per cent.
Copper	0.01024 per cent.
Iron	0.00556 per cent.
Zinc	0.00136 per cent.
Nickel	0.00115 per cent.
Lead, by difference	99.97774 per cent.
	100.00000 per cent.

The pigs weigh 83 pounds each, and are branded with the figure of a buffalo.

(5). The *Star* furnace, S. 21, T. 42, R. 18 W., Morgan county, is owned by J. P. Clark, of Versailles, and uses chiefly ore from the Coffin Spring Creek, belonging to the furnace owner. The fire box is 7 feet by 2 feet; the hearth is 9½ feet by a greatest width of 3 feet. The fire brick are made by the Oak Hill Company, and last usually about four months. Three charges of 1,500 lbs. each, are treated every 24 hours, by the labor of two smelters and one helper. The residues yield by analysis:

Silica	14.349 per cent,
Lime	5.932 per cent.
Magnesia	1.007 per cent.
Ferric oxide	2.323 per cent.
Alumina	.371 per cent.
Antimony oxide	.086 per cent.
Arsenic oxide	.959 per cent.
Zinc oxide	.886 per cent.
Copper oxide	trace.
Lead sulphide	5.334 per cent.
Lead sulphate	5.085 per cent.
Lead oxide, by difference	63.868 per cent.
	100.000 per cent.

The lead compounds in the above contain 67.19 per cent. metallic lead, in addition to 0.25 per cent. of shot lead in the original residue. The lead gave on analysis the following result:

Arsenic	0.01086 per cent.
Antimony	0.00513 per cent.
Silver	0.00029 per cent.
Copper	0.01376 per cent.
Iron	0.00462 per cent.
Zinc	0.00090 per cent.
Nickel	0.00085 per cent.
Lead, by difference	99.96359 per cent.
	100.00000 per cent.

The lead is more arsenical than any analyzed; probably the charges contain a notable amount of arsenical pyrites or some arsenical copper ore.

(6). *Wyan Spring* furnace, S. 32, T. 42, R. 17 W., Morgan county, is owned by T. M. Avery, of Chicago, but is operated under the superintendence of —— Gooding, of Versailles. The ore is obtained from the workings connected with the furnace, and is almost entirely galenite.

The hearth has 10 feet length and a greatest breadth of 5.25 feet. The fire box is two feet wide and six feet long. The fire bricks used in the construction are made at Oak Hill and Marine, Illinois. A charge consists of 1,800 lbs. mineral, and is worked off in 24 hours by two smelters and one helper, and with the consumption of 1.13 cords wood. The rate of expenditure of fuel is therefore 1.25 cords wood per two thousand pounds ore put in treatment.

The roasting period lasts commonly one hour, when the temperature is raised. A second roasting and firing, and finally a pressing of the residues follow, so that, while the consumption of fuel is greater, the residues are worked cleaner, as exhibited in the annexed result of analysis, and the gain in lead is decidedly disproportionately greater than the expense entailed by the use of the additional fuel:

Analysis of Wyan Spring Residue.

Silica	18.203 per cent.
Lime	17.782 per cent.
Magnesia	1.755 per cent.
Ferric oxide	5.229 per cent.
Alumina	.294 per cent.
Antimony oxide	.065 per cent.
Zinc oxide	.996 per cent.
Lead sulphide	16.456 per cent.
Lead sulphate	2.104 per cent.
Lead oxide	27.005 per cent.
	99.889 per cent.

The residue contains no appreciable amount of mechanically enclosed lead; the total lead contained in the lead compounds is 40.76 per cent. of the slag.

The resulting lead has the following centessimal composition:

Arsenic	0.00338
Antimony	0.00048
Silver	0.00057
Copper	0.01002
Iron	0.00428
Zinc	0.00139
Nickel	0.00202
Lead, by difference	99.97786
	100.00000

The furnace runs about two weeks in each month, in which time is smelted the total product from the diggings of the company, as well as some purchased ore.

(7). *Bond's Air* furnace—S. E. qr. N. E. qr., S. 16, T. 40, R. 17 W., works chiefly ore from the Buck Creek diggings, about three miles distant, southerly from the furnace. It was built in 1867, by the Messrs. Bond, but is now leased by them to Messrs. Wangelin, Bradbury & Co., of Jefferson City.

The following dimensions have been furnished us:

Length of hearth	8 feet.
Greatest width	4 feet.

Narrowing down however, towards each door, to a width of ten inches.

Length of fire box	4 feet.
Width of fire box	2 feet.
Depth of fire box	2 feet.
Height of fire bridge above grate	2 feet.

The slope of the hearth is eighteen inches in the eight feet length.

The fire brick lining is from the establishment of Messrs. Evans & Howard, St. Louis, and is reported to last during two campaigns of a furnace. The furnace runs from four to six months in the year.

The charge is fifteen hundred pounds worked off in twenty-four hours, consuming three-fourths cord of wood. The yield is 12 pigs of 90 pounds each, or 1,080 pounds of metal, equivalent to 72 per cent. from the ore put in treatment.

The roasting period covers about three-fourths of an hour, but the comparatively clean character of the residues would seem to indicate that it has been repeated. There appears to have been a setting up of the residues also. A sample of the residue gave the following composition on analysis:

Silica	20.211 per cent.
Lime	7.791 per cent.
Magnesia	1.641 per cent.
Ferric oxide	4.459 per cent.
Alumina	.557 per cent.
Antimony oxide	.132 per cent.
Zinc oxide	.417 per cent.
Lead sulphide	26.152 per cent.
Lead sulphate	1.932 per cent.
Lead oxide, by difference	36.708 per cent.
	100.000

A little baric sulphate is included in the per centage of silica, but no separation was deemed necessary.

The residues contain no shot lead. The total per centage of metallic lead contained in all the lead compounds is 58.047.

The brand of pig is "GRAVOIS." It gives on analysis, as follows:

Antimony	0.00495
Silver	0 00084
Copper	0.00556
Iron	0.00411
Zinc	0.00181
Nickel	0.00195
Lead, by difference	99.98078
	100.00000

(8) The *Linn Creek* furnace, Sec. 25, T. 39, R. 17 W., Camden county, is owned by Draper & McClurg, and was built in 1871. The fire box is 4.5 by 2 feet. The dimensions of the hearth are 8 feet by 4. The interior lining is of fire brick from the establishment of Messrs. Evans & Howard of St. Louis. The campaigns of the furnace have been, for the year 1871, three months; for 1872, four months; for 1873, six months, and for 1874, nine months, with new lining after each campaign.

The ore treated is mainly from the Buck Creek diggings, eight and a half miles distant from the furnace. The charge is usually 1,500 lbs. elaborated during the 24 hours with the consumption of one cord of wood. It yields twelve pigs of 84½ lbs. weight each, or 1,014 lbs. metal, equal to 67.6 per cent. of the ore treated. The residues hold no shot lead and contain:

Silica	26.342 per cent.
Lime	5 234 per cent.

Magnesia	.889 per cent.
Ferric oxide	4.073 per cent.
Alumina	1.187 per cent.
Antimony oxide	trace.
Arsenic oxide	trace.
Zinc oxide	.519 per cent.
Lead sulphide	36.445 per cent.
Lead sulphate	4.529 per cent.
Lead oxide, by difference	20.872 per cent.
	100.000

The lead compounds are equivalent to 54.029 per cent. of metallic lead in the residues.

The pig lead is branded "Draper & McClurg," and gives the following result on analysis:

Arsenic	0.01423 per cent.
Antimony	0 00639 per cent.
Silver	0.00043 per cent.
Copper	0.01511 per cent.
Iron	0.00376 per cent.
Zinc	0.00136 per cent.
Nickel	0.00135 per cent.
Lead, by difference	99.95737 per cent.
	100.00000

A small furnace has been lately erected in Benton county for the treatment of ores obtained in the explorations now being actively pushed in the sub-carboniferous rocks of the vicinity. At the time of our visit the furnace was not running.

The other reverberatories in counties included in the central lead district, and of which no particulars can be given, are as follows:

Marmaduke	Sec. 19;	T. 49;	R. 19 W.;	Saline county.
Old Scott	Sec. 26;	T. 49;	R. 19 W.;	Cooper county.
Collin's	Sec. 19;	T. 49;	R. 18 W.;	Cooper county.
Handlin	Sec. 9;	T. 42;	R. 17 W.;	Morgan county.
Otterville	Sec. 21;	T. 45;	R. 19 W.;	Morgan county.
Eanes & Berry	Sec. 32;	T. 44;	R. 14 W.;	Moniteau county.
Eureka	Sec. 23;	T. 43;	R. 14 W.;	Cole county.
Pratts Mill	Sec. 26;	T. 43;	R. 14 W.;	Cole county.
Pioneer	Sec. 25;	T. 41;	R. 14 W.;	Miller county.
Grassroot	Sec. 23;	T. 41;	R. 14 W.;	Miller county.

The total number of air furnaces in the central distrit is 19.

The following analysis of the Handlin (Morgan county) furnace residue is added:

Silica	16.860 per cent.
Lime	9.723 per cent.

Magnesia	2.484 per cent.
Ferric oxide Alumina	5.940 per cent.
Antimony oxide	0.215 per cent.
Arsenic oxide	trace.
Zinc oxide	1.829 per cent.
Cupric oxide	nil.
Lead sulphide	23.322 per cent.
Lead sulphate	3.061 per cent.
Lead oxide	36.607 per cent.
	100.041

The residue contains 0.28 per cent. of free or mechanically mixed lead, while the total content of metallic lead is 56.348 per cent.

The Eanes & Berry furnace, according to Schmidt's report (page 558), works usually charges of twelve hundred pounds weight. Attempts to increase the weight to a ton, or even to fifteen hundred pounds were unsuccessful, having invariably been found to be disadvantageous in regard to the yield from the ore. The dimensions of the furnace are not given.

(C.) THE SOUTHEASTERN REGION.

(1) *St. Joe. Furnaces:* The estate of the St. Joe. company includes eighteen hundred acres of land, chiefly in St. Francois county. The mining operations are mainly concentrated upon the deposits in Sec. 33; T. 38; R. 4 E., about six miles from Cadet, a station on the Iron Mountain Railroad. The character of these deposits and their geological position are so closely allied to those of the well known Mine la Motte property, and both have been so fully described elsewhere, that repetition in this report would be superfluous.

The ores receive mechanical treatment and concentration before furnace treatment. Their composition is shown in the results of analyses of the galenite from the different Missouri localities (ante. pp. 16 and 17). The furnace yield is said to range between 65 and 70 per cent. metal.

The smelting establishment includes eight reverberatories of the common form, but with larger hearth area than those in general use in the State. The dimensions are:

Length of hearth	14 feet.
Width of hearth	3 feet.
Length of fire box	5 feet.
Width of fire box	4 feet.

So that the areas of the hearth and grate are to each other nearly as two to one. The charge is larger, being two thousand pounds mineral, worked off in eight hours, or three such charges being treated in one day. The fuel consumed is three-fourths of a cord of wood per charge or per ton. There are required two days (of eight hour's) labor per ton of ore treated.

The residues are saved for treatment for nickeliferous matte and lead at such time when the proper furnaces have been decided upon and erected. Their value for nickel and cobalt as well as the full composition is shown in the following result of analysis:

Silica	26.252	per cent.
Lime	9.908	per cent.
Magnesia	3.704	per cent.
Ferric oxide	11.533	per cent.
Alumina	3.223	per cent.
Arsenic	0 059	per cent.
Antimony	trace.	
Zinc oxide	trace.	
Copper sulphide	0.981	per cent.
Cobalt sulphide	0.523	per cent.
Nickel sulphide	0.777	per cent.
Lead sulphate	3.455	per cent.
Lead sulphide	37.792	per cent.
Lead oxide, by difference	4.793	per cent.
	100.000	

A special estimation of lead by humid analysis gave a total of 38.695 per cent in the residue.

The slags are richer in unchanged sulphide and poorer in oxide of lead than any thus far examined. Inferentially the heat has been too great for advantageous roasting, or the time too short for the weight of charge on the sole; but as these residues are intended for further treatment by a method looking to the winning of the lead and the production of a nickel matte, their richness in metallic lead is for the present a secondary matter.

Analysis of St. Joe Reverberatory Lead.

Arsenic	0.00183	per cent.
Antimony	0.00675	per cent.
Silver	0.00405	per cent.
Copper	0.06394	per cent.
Iron	0.00137	per cent.
Zinc	0.00082	per cent.
Nickel	trace.	
Lead, by difference	99.92124	per cent.
	100.00000	

(2) *Des Loge* furnace, in St. Francois county, was started in

June of the present year. The ores treated are similar to those reduced at the St. Joe, being, in fact, extracted from the same deposit, the mines of the two companies being only about one thousand feet apart. The ore from the mine yields, by humid analysis, upwards of nineteen per cent. metallic lead, and contains the following amounts of foreign metals: Copper, 0.798; zinc, 0.611; nickel and cobalt (not separated) 0.137; iron, 2.240, chiefly as pyrites, some little, however, occurring as oxide in the limestone gangue.

An excellent system—the most complete in the State—has been adopted for the mechanical concentration of the ores, and the capacity is reported to be equal to the handling of one hundred and twenty-five tons of raw ore daily. A Blake's breaker, No. 5, makes the preliminary crushing of the ore, whence it passes to two setts of rolls, the larger of which has a twelve inch face, the smaller being fourteen inches in diameter, with a fourteen inch face. The crushed ore is sized by four revolving screens or trommels, which yield seven grades of mineral, corresponding to 19 (and over), 12, 8, 6, 4, 2 and 1-thirty-second of an inch diameter. The largest size is returned to the rolls, the others are passed to jigs, the coarser being treated in those of the Rittinger system; the finest being classified into four grades, of which the two coarsest are treated in an eccentric jig and the other on a Rittinger table. A seventy-horse-power engine furnishes the motive power. Eight men are required for a post of twelve hours, representing the treatment of some sixty tons raw ore. The establishment was designed by Mr. Ferdinand Stolinski and erected under his personal supervision. In the following table are given the yields of the products in metallic lead and the content of foreign metals, the numbers corresponding to the recognized grades at the works:

	Lead—per cent.	Iron—per cent.	Zinc—per cent.	Copper—per cent.	Nickel and Cobalt—per cent.	Antimony.
No. I	63.61	1.559	1.276	0.167	0.573	trace.
No. II	76.42	0.067	0.237	0.040	0.042	trace.
No. III	67.96	0.823	0.670	0.091	0.078	trace.
No. IV	76.93	0.655	0.689	0.132	0.004	trace.
No. V	74.42	0.928	0.433	0.092	0.039	trace.
No. VI	73.68	1.761	0.249	0.226	0.039	trace.
Coarse "chats"	35.238	6.185	1.006	2.735	0.480	trace.
Middlings	31.076	9.911	2.002	2.627	Ni O. 168 Co.O. 857	trace.
Fine "chats"	11.620	27.521	trace.	2.622	1.107	trace.

In No. I.–VI. (inclusive) the lead was estimated by fire assay, the mean of two results being given; the trace of antimony was found in ten grammes of the materials. The lead in the "chats" and "middlings" was reached by weighing as sulphate in the ordinary process of humid analysis, and the results are higher by at least five per cent. than if the usual fire assay had been made.

One furnace of the Flintshire pattern has been put in operation. Twenty-five hundred pounds of dressed mineral are charged at a time. The charge is elaborated in ten hours. Thirty pounds of fluor spar (calcium fluoride) are added to a charge, with a view probably to getting rid of some of the silica of the charge. The average slag shows a composition, as follows :

Lead sulphate	1.299	per cent.
Lead sulphide	40.105	per cent.
Lead oxide	9.313	per cent.
Nickel and cobalt oxides	0.0911	per cent.
Copper oxide	0.903	per cent.
Ferric oxide	10.874	per cent.
Calcium fluoride	0.711	per cent.
Alumina	0.227	per cent.
Lime	12.552	per cent.
Zinc oxide	2.303	per cent.
Magnesia	3.218	per cent.
Silica	16.987	per cent.
	99.404	

A sample of the pig lead gave the following:

Lead, by difference	99.93383	per cent.
Iron	0.00307	per cent.
Zinc	0.00398	per cent.
Copper	0.05872	per cent.
Silver	0.00045	per cent.
Arsenic, antimony, nickel	traces*	
	100.0000	

An analysis of the *dross* or skimmings from the lead pot was also made and is here introduced to show the concentration of certain metals in the process of refining by poling in the lead pot. The estimation of the lead and foreign metals (excepting silver) was made, the other constituents not being determined. In one hundred parts of the dross there are—

Lead	87.597
Iron	0.638
Zinc	0.313
Nickel	0.703
Copper,	0.987
Antimony	trace
Silver	not estimated.

The lead was estimated as sulphate.

---

*In 100 grammes.

[*The Ash Grove Mining Company*, operating in T. 30, R. 24 west, have taken out upwards of forty thousand pounds of ore in explorations of less than four months. I have no information concerning the character of the deposit, nor of the details of the smelting, if any have been started.]

(3.) *The Saline Valley Mining Company*, operating the Avon mines, Sec. 12, T. 35, R. 7 E., Ste. Genevieve county, have two reverberatories for ore treatment, and a slag furnace for residues. The ores occur similarly to those at La Motte and St. Joe.

(4.) *The Frumet Company* (owning 1,300 acres in Secs. 28, 33 and 34, T. 40, R. 3 E, Jefferson county), have one reverberatory of the Flintshire pattern, and a cupola for residues. A sample of the lead produced gave the following composition of analysis:

Arsenic	0.00004
Antimony	0.00007
Silver	0.00179
Copper	0.04399
Iron	0.00282
Zinc	0.00240
Nickel	0.00063
Lead, by difference	99.94826
	100.00000

II.—ILLUSTRATIONS OF HEARTH TREATMENT.

A general outline of hearth treatment has been given on preceding pages. The forms of apparatus in use in the State are either the single tuyere, old fashioned hearth, or the more recently introduced American water-back hearth, commonly with three tuyeres. This latter form has been described at sufficient length for purposes of this report in the section devoted to the general practice of lead metallurgy. The details which will be given under the present heading, though somewhat meagre, will be found sufficiently full for reaching general conclusions with respect to the economics of the method. This is the more especially true since the variations in the dimensions of the hearths and the weight of charge treated on them are not wide as in the case of the reverberatory, and since as a general thing the purer and higher grade ores alone receive treatment. The results of analyses of slags or residues which are given in this section, while perhaps of value to those who may submit them to further treatment in the slag furnaces, can give no correct ideas of the thoroughness with which the lead may have been extracted, unless, indeed, the whole conditions of weights of materials introduced, of loss by volitilization, and of amount of residue produced, are given as factors in our calculations. The fact that these residues generally contain unconsumed fuel, and frequently, also, the lime which has been added as the so-called flux, may make the per centage of lead seem small in some instances, while, if it were possible to calculate the wastage back upon the original ore, it is really much greater. The difficulties in this respect are even more serious than those which obtain in the case of the reverberatory treatment, where, if it were possible to know the true and full composition of the ore, and the residue, and the weight of the latter produced, the chief disturbing element would be the foreign matters introduced by the wear of the furnace linings. Unfortunately, no such data as we desire exist for any of the Missouri furnaces, for no attempts are made to control or improve empirical results by stochiometrical calculations.

The order adopted for the reverberatories will be followed in considering the hearth method.

A.—SOUTHWESTERN REGION: (1.) The Granby Company operates six of the improved hearths. The common duty of a hearth is about 3,000 pounds No. 1 block mineral worked off in an eight hour shift. The labor required for this amount of ore is two smelters and one helper—the latter serving two or more hearths by delivering ore and coal to them. Two shifts are usually worked in a day for six days, equivalent to the treatment of 216,000 pounds of mineral, yielding 181,200 pounds of metal, or 70 per cent. For the production of this amount of metal there are consumed 660 bushels of charcoal and 60 bushels of lime for so-called flux. The latter costs 25 cents per bushel.

The lead produced is somewhat purer than that produced by the reverberatories, as will be seen from the subjoined analysis, and a comparison of these results with those given while treating of the Flintshire and ordinary air furnace results at Granby. This result may be due to features inherent in the process, or it may be ascribed to greater purity of ore put in treatment:

Granby Hearth Leads.

	I.	II.
Arsenic	.00019	0.00124 per cent.
Antimony	.00198	0.01085 per cent.
Silver	.00045	0.00057 per cent.
Copper	.00479	0.00780 per cent.
Iron	.00220	0.00367 per cent.
Zinc	.00142	trace.
Nickel	.00047	0.00087 per cent.
Sulphur	trace.	trace.
Lead, by difference	99.98850	99.96905 per cent.
	100 00000	100.00000 per cent.

No 1. is an analysis already published (in *Report of Board of Curators of State University* for the years 1874-75); No. 2. is a recent analysis of a sample representing 23 pigs.

The Granby Mining and Smelting Company have also three hearths at Oronogo, Jasper county, which have not, however, been operated for some time, the ores mined at that place having been sent to Granby for treatment.

(2.) At Lone Elm, Jasper county, Messrs. Moffett & Sergeant have three hearths of the newer form. Each treats three thousand pounds of high grade ore per shift, with three shifts per day. Fifteen bushels of charcoal are required for the treatment of the three thousand pounds of mineral. The yield is 66.6 per cent. metal, with the following composition;

Arsenic	.00027	per cent.
Iron	.00777	per cent.
Zinc	.02071	per cent.
Copper	.00463	per cent.
Nickel	.00031	per cent.
Antimony	.00266	per cent.
Silver	.00025	per cent.
Lead, by difference	99.96340	
	100.00000	

(3.) With three hearths, the Joplin Mining and Smelting Company treat 108,000 pounds of mineral per week, which, at two eight hour shifts per day, would correspond to 3,000 pounds per shift. The yield is reported at 66.6 per cent. of lead. No details respecting these, nor the following hearths in the southwestern region have been obtained.

Riggins & Chapman	3	Hearths.
Thurman Mining and Smelting Co	1	"
S. B. Corn & Co	2	"
Neosho Manufacturing Co	1	"

These with those above described, give a total of twenty-two hearths for the district.

(b) Central Region—The central lead region furnishes but one illustration of the hearth treatment. It is situated in Sec. 17, T. 41, R. 17, W., in Morgan county; was built in 1869, and belonged to the Messrs. O'Brien. It belongs to the old form, single tuyere hearth, which is of less capacity than the improved form, treating 6,000 pounds of mineral in three eight hours shifts. The yield is usually sixty pigs of sixty-seven pounds each, or 4,020 metal, equivalent to 67 per cent.

of the ore put in treatment. The blast is furnished by bellows moved by water-power.

A sample of the residue gave, on analysis, the following result:

Silica, with unconsumed charcoal and a little baric sulphate	43.065	per cent.
Lime	5.226	per cent.
Magnesia	1.345	per cent.
Ferric oxide	3.021	per cent.
Alumina	trace.	
Antimony oxide	0.334	per cent.
Zinc oxide	0.456	per cent.
Lead sulphide	28.943	per cent.
Lead sulphate	0.432	per cent.
Lead oxide	19.219	per cent.
	99.935	

The lead compounds in the above contain 42.492 per cent. of metallic lead. The original slag showed but 0.07 per cent of shot or mechanically enclosed lead, separated in the operation of preparing the sample for analysis.

C—*Southeastern Region*—(1) The Hopewell furnace, Washington county, has been in operation since 1839.* The character of the ore treated is shown in the test of results of analysis [Nos. XXXIV and XXXV, page 17]. The hearth used is the old form, twenty inches wide by twelve deep, with a single tuyere, the blast being supplied by a double acting bellows operated by an overshot water-wheel. Of the amount of ore treated and of the consumption of fuel we were unable to learn any particulars. The usual time is a shift of eight hours or the period required to produce twenty-four pigs of sixty-one pounds each, or 1,464 pounds of metal. As this furnace is of the same character and size as the O'Brien hearth, just described, it may be presumed that the ore worked to produce this weight of metal was about two thousand pounds, giving a yield of 73.2 per cent. for the ore put in treatment. In the year 1875, this furnace received 410,000 pounds of ore, and produced 5,130 pigs or 312,930 pounds of metal, corresponding to a yield of 76.3 per cent. The metallic production probably included some slag lead, for the Hopewell residue are treated in a slag furnace; so that there is probably very little risk in assuming the yield to be about 73 per cent. as before calculated.

The residue has the following composition:

Silica	36.348	per cent.
Lime	3 604	per cent.

---

* Dr. Litton (in Swallow's Report) pp. 58 and 59.

Magnesia	1.436 per cent.
Ferric oxide	2.707 per cent.
Alumina	0.052 per cent.
Antimony oxide	0.168 per cent.
Arsenic oxide	0.218 per cent.
Zinc oxide	2.527 per cent.
Copper oxide	a trace
Lead sulphide	19.647 per cent.
Lead sulphate	2.913 per cent.
Lead oxide, by difference	30.383 per cent
The total per centage of metallic lead in the residue is	47.267
	100.000

(2) *Perry Furnace*, one and a half miles from Potosi, Washington county, is a hearth of the old pattern and of the size as that given for the Hopewell. The blast enters through one tuyere of two and a half inches diameter and is supplied by a Sturtevant blower, No. 3. The ore is prepared by crushing by hand on iron grates.

The length of time required for the production of thirty pigs of seventy pounds each, (2,100 pounds of metal) constitutes a shift, in which are worked two smelters and a helper as usual. There are commonly treated in this time about three thousand pounds of mineral with the consumption of ten bushels of charcoal. This furnace is very actively worked, its production in 1875 being 14,600 pigs, or 1,022,000 pounds of metal, corresponding to 487 shifts of thirty pigs each.

The residues are treated in the same hearth after concentration by washing. The second residue, or those resulting from the re-smelting, are sold for treatment in the slag furnaces of the vicinity. The hearth residues from the original ore have the following composition:

Silica	25.335 per cent.
Lime	4.230 per cent.
Magnesia	0.711 per cent.
Ferric oxide	1.414 per cent.
Alumina	1 912 per cent.
Nickel oxide	0 746 per cent.
Zinc	0.591 per cent.
Arsenic oxide	0.023 per cent.
Lead	48.577 per cent.
Sulphur, sulphuric acid and oxygen in combination with lead	not estimated.

The Perry pig lead gives, on analysis, the following:

Arsenic	0.09674 per cent.
Antimony	0.00502 per cent.
Silver	0.00190 per cent.
Copper	0.00499 per cent.
Iron	0.00436 per cent.
Zinc	0.00598 per cent.
Lead, by difference	99.97191 per cent.
	100.00000

(3) *Valle Furnaces*—The estate of the Valle Company comprise 4,500 acres, chiefly in the counties of Jefferson and St. Francois. The deposits on Sec. 7 and 8, T. 38, R. 5, E., were discovered as early as 1824, by Joseph Schuts, and have been worked with but little interruption since that date.

The furnace includes two hearths and a cupola for slag treatment. The hearths have each a single tuyere; the blast being furnished by a No. 6 Sturtevant blower, run by a 30 horse power engine. The hearths are 18 inches by 14 inches in section. In an eight hour shift there are treated 3,500 pounds ot ore, producing thirty-three pigs of seventy-five pounds each, or 2,475 pounds of metal of the "Rozier" brand. This yield corresponds to 70.7 per cent. For the Valle brand of metal, there is used a previously calcined and sorted ore which is charged, to the same amount, in the Scotch hearth, producing thirty pigs of eighty pounds each.

During each shift a hearth requires four bushels of charcoal and one-twentieth cord of wood. The charcoal is supplied to the company by the burners at a cost of eight cents per bushel, the company cutting the wood. Occasionally, when the mineral is not very clean, a little lime is used.

The ore is worth $29 per miner's thousand when lead is worth seven cents per pound, and $1.50 additional per thousand for each one-quarter cent advance in the market price of the metal.

The hearth residues are treated in a small slag cupola. They show the following composition:

Silica	8.971 per cent.
Lime	3.308 per cent.
Magnesia	0.100 per cent.
Ferric oxide	4.366 per cent.
Alumina	0.061 per cent.
Arsenic and antimony	traces.
Zinc oxide	18.129 per cent.
Copper oxide	0.243 per cent.
Lead sulphide	22.922 per cent.
Lead sulphate	1.945 per cent.
Lead oxide	39.390 per cent.
	99.322
Metallic lead in above	57.744

The two brands of pig lead procured at the the Vallé furnaces are composed as under:

	Rozier.	Vallé.
Arsenic	0.00925	trace.
Antimony	0.00184	0 00214

	Rozier.	Valle.
Silver	0.00615	0.00326
Copper	0.03742	0 04165
Iron	0.02497	0.00453
Zinc	0.00118	0.00294
Lead, by difference	99.91919	99.94548
	100.00000	100.00000

During the year 1875, there were smelted 602,320 pounds of ore, producing 1,801 pigs, or 135,075 pounds of Rogers' brand, and 3,589 pigs, or 287,120 pounds of Vallé brand of metal, giving a total of 422,195 pounds of lead, equivalent to 70.09 per cent. of the ore treated. The residue gave, by further treatment in the slag furnace, 929 pigs of 60 pounds each, or 55,740 pounds of Phœnix lead. [See section on slag smelting, page 75.]

(4.) *Mine La Motte.*—Historically, as well as economically, Mine La Motte is one of the most important and interesting of the lead developments in the State. Its estate includes twenty-four thousand acres in Madison county, on some parts of which lead ores appear to have been discovered, as early as the year 1723, by Renault and La Motte, the last named beginning work in the following year. The discovery of nickel and cobalt minerals in association with the lead ores in the deposits at this locality, and the very considerable importance which the production of nickel matte as an incidental product has reached, has of late greatly increased the interest which is attached to the well known and frequently described Mine La Motte.

The metallurgical operations on this property comprehend the treatment of the ores by the hearth process and the further treatment of the residues and roasted ores, notably nickeliferous, in the cupola furnace for the production of a slag lead and a nickel matte. The latter is further treated by concentration for market.

The hearths are two in number, of the newer form. They are two feet wide by twenty-two inches deep, with three tuyeres each; the blast having usually a pressure of four ounces. Each hearth works thirty-two hundred pounds of material in about six hours time, with two smelters and one helper serving the two hearths. Eight bushels of charcoal are consumed for this work. The product is twenty-five pigs of eighty pounds each, or 2,000 pounds of metal, equivalent to 68.7 per cent.

From March 27, 1875, to December 25, of the same year, 7,906 cars of ore were crushed, producing 6,786,279 pounds of clean mineral for furnace treatment. In the same time there were smelted 5,808,747 pounds of this washed ore, producing 49,104 pigs, or 3,928,320 pounds of lead, of which 41,136 pigs, or 3,290,880 pounds were from the hearths, and 7,968 pigs, or 637,440 pounds were produced by the cupola. With an average yield of 68.7 per cent., the amount of ore treated to produce the above named amount of metal from the hearths would have been 4,790,048 pounds, leaving for the cupolas 1,018,520 pounds.

The amount of residue produced by the hearths at Mine La Motte is reported at from 10 to 17 per cent. of the ore put in treatment, or, say an average of 13.5 per cent. A charge of 3,200 pounds would therefore yield 431 pounds of residue, the composition of which we find to be as under:

Silica	15.536 per cent.
Lime	10.237 per cent.
Magnesia	5.488 per cent.
Ferric oxide	15.656 per cent.
Alumina	1.883 per cent.
Nickel sulphide	1.021 per cent.
Cobalt sulphide	.719 per cent.
Cuprous sulphide	.154 per cent.
Antimony and Arsenic	trace.
Zinc oxide	1.315 per cent.
Lead sulphide	20.999 per cent.
Lead sulphate	3.904 per cent.
Lead oxide, by difference	23.087 per cent.
	100,000

With a yield of 43.011 per cent. of metallic lead.

The nickel and cobalt compounds in the above result, correspond, respectively, to 0.666 per cent. nickel and 0.466 per cent. cobalt. An analysis of a second sample gave W. C. Minger, in this laboratory, 1.242 per cent. nickel sulphide (= 0.852 metallic nickel) and 0.675 per cent. cobalt sulphide (= 0.438 cobalt). The mean, which probably represents a fair run of the residues, is 0.759 per cent. metallic nickel and 0.452 per cent. cobalt. These residues are treated further for metallic lead and a concentrated nickeliferous mass or nickel matte, by a method which will be found detailed in the section devoted to the methods of slag or residue treatment. The matte is a source of no inconsiderable revenue to the company.

The hearth lead produced at Mine La Motte we find to have the following composition:

Arsenic	0.00034 per cent.
Antimony	0.00119 per cent.
Silver	0.00345 per cent.

Copper	0.01999 per cent.
Iron	0.00248 per cent.
Zinc	0.00164 per cent.
Nickel	0.00095 per cent.
Lead, by difference	99.96996 per cent.
	100.00000

The hearths at Mine la Motte have connected with them chambers for fume condensation, in which are collected thirty-three thousand three hundred pounds of fume in three months' time, or 100,000 pounds resulting from the hearth treatment of 4,790,048 pounds of ore. The fume yields 70 per cent. metal, and is treated with residue and ore in the cupolas. The loss by volatilization in the hearths is, therefore, 2.17 per cent. of the ore, or 3.04 per cent. of the lead produced in the treatment in the ore hearth.

SLAG TREATMENT IN MISSOURI.

Slag treatment, or the extraction of lead from the residues of the reverberatory and the hearth processes, is practiced at Granby, at Davis' & Murphy's establishment, Joplin, at Hopewell, at Perry, and at the Valle furnaces. To this list may be added the more elaborate cupola process, in use at Mine la Motte, where the residues are treated mainly for the production of nickel matte, by which a metallic lead is also recovered. The character of the material to be treated has already been indicated by the various analyses given on preceding pages, in the description of the several furnaces and ore hearths. It will be seen that the residues consist of varying admixtures of more or less scorified gangue substances with lead sulphide, sulphate and oxide—the latter both free and in combination with silicic acid—and contain besides mechanically enclosed or "shot" lead. The foreign metals of the ore, which are but slightly volatile at the temperature of treatment, are also concentrated in these residues, which, therefore, produce necessarily a more impure metal than do the original ores.

The form of furnace in use is the North of England slag hearth, so modified as to increase rapidity of handling and production, but retaining at the same time the valuable features of economy of original plant and convenience of operation. It belongs to the class of furnaces distinguished as *Krummofen*, being a low shaft, usually of four feet height, with rectangular section, provided with a cast-iron bed-plate or sole sloping downwards towards the front, usually at the rate of one and a half inches to the foot, and extending outwards so as to overlap the iron pot in which the lead is received. This bed-plate carries the fire-brick lining of the sides of the furnace. A single tuyere, commonly three inches in diameter, enters the back of the furnace at a height of eight or ten inches above the sole. The "eye" or port of the furnace is about nine inches in height by one foot in width, and is closed during the operation of the furnace by well rammed, stiff clay; through the bottom of this, at the bed-plate, a wooden tap-plug passes, which is withdrawn at about an hour after the furnace is in full blast. The exterior lead pot is divided unequally by a partition descending nearly to the bottom; the larger division is kept filled with pieces of charcoal, over which the slag runs, and through which the lead filters and passes beneath the partition into the smaller division or moulding side of the pot. The slag passes thence into the water tank. The very considerable volatilization of the lead requires that the furnace should be surmounted by proper contrivances for ventilation.

The furnace is operated by tamping over the bed-plate a brasque composed of equal volumes of clay and coke, so arranged as to be about one inch in thickness at the port and four inches at the back. A fire is kindled with dry wood in small stacks arranged against each other in an inverted V shape, from front to back, and slightly above the tuyere in height, so as to form an unobstructed passage for the blast from back to front of the furnace. On this live coals are thrown above the tuyere, the furnace filled with charcoal, and the blast turned on to the extent of about half the usual amount. As soon as the charcoal is thoroughly ignited, the normal fuel or coke is spread over it to the depth of six inches, the full blast turned on and the slag spread evenly over the coke. At intervals the normal change of coke and slag (about one measure of the first to four of the latter) are thrown in, in such a manner that the slag is chiefly at the back and the coke mainly at the front of the furnace, and the furnace is kept full. The proper manipulation of the furnace for forming the nose is a matter of experience. The discharge of slag indicates the working, A sluggish dull-red flow of slag over the charcoal in the pot indicates too low heat, and loss of lead by mechanical enclosure; fuming of the slag shows matte production, while a thin stream of slag, of a bright straw-yellow color, shows a proper condition of working.

The highly basic character of the materials smelted in these furnaces, causes rapid wear of the linings, and a campaign is necessarily a very short one, rarely exceeding thirty-six hours. The furnace is blown out by ceasing to charge with residues, but continuing the introduction of the fuel till no more slag runs from the tap-hole. The repairs are readily and cheaply made. No flux is added to the charge, but when practicable the residues rich in lime are mixed with furnace bottoms for treatment.

The usual duty of the slag hearth in this State is from twelve to fifteen tons of residues, run through in the double shift of sixteen hours, requiring six workmen, as follows: One slag smelter, working the double shift, two helpers and two pot-men; one of each in each eight hour shift.

There will be used in such a run about four thousand pounds of coke, about twenty bushels of charcoal, besides the fuel for the blowing apparatus. The cost of treatment will be about as under:

Labor, per ton residues	0.4 day.
Coke, " "	266 lbs.
Charcoal, " "	30 lbs.

The furnace yield is about 23 per cent. lead. The other products are black-slag, lead-dross, wall accretions from the furnace and fume. The lead is hard lead, as noted above; it is usually refined by melting and poling in the reverberatory, but is not then as satisfactory for chemical purposes, and usually ranges in the market a half cent per pound less than the soft leads directly from the ores.

The *Granby* slag furnace has a horizontal section of 2 feet 6 inches by 3 feet. The slag gives the annexed composition on analysis:

Silica	37.648	per cent.
Lime	4.005	"
Magnesia	1.754	"
Alumina	3.268	"
Ferrous oxide	2.258	"
Lead oxide	33.778	"
Zinc oxide	14.221	"
Lead sulphate	0.359	"
Ferrous sulphide	1.927	"
	99.218	

Besides traces of antimony. Regarding the ferrous sulphide as a small amount of matte, mechanically mixed with the slag, and taking the lead sulphate as a foreign matter, there remains 96.932 per cent. of true slag, with the following per centage composition and amounts of oxygen in the respective constituents :

		Per cent. Oxygen.	Ratio of O. of Silica to O of bases.
Silica	38,84 per cent.	20.71	20.71
Lime	4.13 "	1.18	10,17
Magnesia	1.81 "	0.72	
Alumina	3.37 "	1.57	
Ferrous oxide	2.33 "	0.52	
Lead oxide.	34.85 "	2.50	
Zinc oxide	14.67 "	2.90	

The ratio is therefore 2 to 1, giving the formula of a bi-silicate slag with lead oxide as the chief base. It is doubtful whether the substitute of some other base for the lead oxide, by the use of flux, would be advantageous on the score of economy, even though saving of the furnace lining might also be obtained thereby.

After refining in the Flintshire furnace, the Granby slag lead has the following composition:

Arsenic	0.00101 per cent.
Antimony	0.04975 per cent.
Silver	0.00029 per cent.
Copper	0.02965 per cent.
Iron	0.00718 per cent.
Zinc	0.00180 per cent.
Cadmium	0.00276 per cent.
Lead, by difference	99.90756 per cent.
	100.00000

The sample represented thirty pigs. The loss of weight in the refining operation is said to be about four per cent.

The slag-furnace of Davis & Murphy, at Joplin, (similar in construction and mode of operation to that at Granby), yields a black slag, with the composition given below:

Silica	24.527 per cent.
Lime	6.130 per cent.
Magnesia	1.055 per cent.
Alumina	.361 per cent.
Antimony oxide	trace.
Zinc oxide	10.335 per cent.
Lead oxide	47.619 per cent.
Lead sulphate	2.765 per cent.
Lead sulphide	1.165 per cent.
Ferrous sulphide	5.189 per cent.
	99.146

Deducting the last three constituents as not belonging properly to the slag, the true slag will be found by calculation to have the following composition and oxygen ratios:

	Per cent. constituents.		Per cent. Oxygen.		Oxygen of silica to oxygen of bases.
Silica	27.252	=	14.53	=	14.53
Lime	6.811	=	1.94	=	8.72
Magnesia	1.172	=	0.47		
Alumina	0.401	=	0.19		
Zinc oxide	11.483	=	2.23		
Lead oxide	52.910	=	3.89		

or about 8 to 5, and showing, therefore, a mixture of singulo- and bi-silicate, with nearly 53 per cent. of lead oxide [=50 per cent. metallic lead]. This composition would seem to indicate that the slag was formed with greater corrosion of the furnace lining, but with less consumption of fuel than the Granby slag; but, on the other hand, it would be more apt to entangle metallic lead, being somewhat thick-flowing, congealing more rapidly and of high specific gravity. The original slag contains also more matte, and is less clean than the first described.

The slag-furnace at *Hopewell* is three feet in height, with a horizontal section of 2 by 2½ feet. Wood is used for fuel, and the hearth residues are fluxed with some black slag from preceding operations in the slag-furnace. The black slag shows:

			Per cent. oxygen.		
Silica	24.539 per cent.	=	13 09	=	13.09
Lime	9.896 per cent.	=	2.83	=	8.25
Magnesia	2.773 per cent.	=	1.11		
Alumina	.385 per cent.	=	0.18		
Ferrous oxide	2.039 per cent.	=	0.45		
Zinc oxide	1.619 per cent.	=	0.32		
Lead oxide	47.920 per cent.	=	3.36		
Ferrous sulphide	9.399 per cent.				
	99 570				

Besides traces of copper oxide. It contains 43.56 per cent. metallic lead, and is, therefore, somewhat cleaner than the preceding; but, like the latter, contains nearly ten per cent. of matte. The oxygen ratio is about 9 to 6, so that the slag is even more basic than that from Joplin.

At the *Valle* works, 5,000 pounds residues are run through the slag-furnace in a twelve-hour shift, with the production of 18 pigs (of

60 pounds each) of slag lead of the brand "Phœnix." The furnace yield is, therefore, a little more than 20 per cent. The furnace is run with charcoal, and slag from the same furnace is used for flux. The slag gives, on analysis:

			Per cent. oxygen.		Oxygen of silica to oxygen of bases.
Silica	20.422	=	10.89	=	10.89
Lime	6.234	=	1.78	=	14.72
Magnesia	0.872	=	0.35		
Ferrous oxide	15.628	=	3.47		
Alumina	5.233	=	2.44		
Zinc oxide	28.432	=	5.61		
Lead oxide	19.081	=	1.37		
Lead sulphate	3.779				
Cuprous sulphide	0.319				

with traces of arsenic and antimony. The oxygen ratio of 1 to 1.5 nearly, indicates a highly basic slag, or sub-silicate with less lead than any slag described, and with a proportional increase in ferrous and zinc oxides. Such a slag should run in a thin stream, and congeal very rapidly, and is very likely from its high specific gravity, to enclose metal. Besides it must be very destructive to the furnace lining, necessitating much more frequent repairs than those previously described. It is stated above that the run of the furnace producing this slag is but twelve hours, and only four thousand pounds residues.

The slag lead of the Valle furnace gives on analysis, the following composition:

Arsenic	0.01556 per cent
Antimony	0.13365 per cent.
Silver	0.00525 per cent.
Copper	0.09323 per cent.
Iron	0.00470 per cent.
Zinc	0.00235 per cent.
Lead, by difference	99.74536 per cent.
	100.00000

In all furnaces of the cupola form, the ashes of the fuel become constituents of the slag. Coke ashes contain from 55 to 65 per cent. of silica, and have alumina and ferric oxides as their chief bases, besides containing lime, magnesia, manganic oxide, and the alkalis. The oxygen ratio of the acid and bases is very nearly that of a bi-silicate, while the great variety of bases makes it a slag of somewhat ready fusibility. The ashes of charcoal on the other hand, contain but small amounts of silica, but are rich in bases such as the alkalis and lime. The alkalis have not been determined in any of the foregoing analysis of slags; they were but small in amount, and would be of no significance from a technical standpoint.

SPECIAL CUPOLA METHOD AT MINE LA MOTTE.

The fact that the hearth residues at Mine la Motte are notably nickeliferous, induces a modification of the slag methods above described, which has for its object the extraction of the lead and the simultaneous production of a matte, in which the nickel shall be concentrated in amount. Besides, some of the ore deposits which are worked at that property, yield ores sufficiently rich in nickel to be passed at once to this treatment, either in admixture with the hearth residues or separately.

The process practiced belongs to the class of true cupola methods, and is the only instance offered in the State. Two forms of furnaces have been used, one with a circular and the other with a trapezoidal section, the latter being the form in use at present.

The circular cupola is 3 feet diameter, tuyeres, 8 inches above hearth plate; height from tuyeres to throat, 12 feet. The so-called square cupola is 28 inches wide in front, 42 inches at the back and 36 inches deep. Three tuyeres, diameter 2⅝ inches, enter the back at a height of 11 inches above the hearth plate. From the tuyeres to the throat, the height is 11 feet. The usual pressure of the blast used is 10 ounces. The sole of the furnace is a mixture of one volume of coke with two of clay.

The shifts run are twelve hours duration, with five men to a shift. Fuel is coke, either from Pittsburgh or from Missouri coal, and is used in the proportion of one Pittsburgh coke to six of ore or residue, or one of Missouri coke to five of ore or residue. The ore or residue charges are about 150 lbs. each, and seven or eight are usually charged per hour. The flux is a hematite from the Iron Mountain region, with a composition as under:

Ferric oxide	94.545 per cent.
Alumina	3.922 per cent.
Magnesia	0.401 per cent.
Lime	0.129 per cent.
Silica	1.182 per cent.
	100.179 per cent.

There seems to be no regular or definite amount of flux used, it being proportioned to the demands of the working of the furnace, as pointed out by the experience of the workmen.

A twelve hour poste represents the treatment of 12,600 lbs. ore, and consumption of 1,800 pounds of Pittsburgh coke. A ton of ore requires therefore, 206 pounds fuel, and at the rate of 0.8 days labor.

It may be mentioned in this connection, that the treatment for the production of matte is largely experimental at present, especially in the direction of the form of furnace best adapted to the work. At the present writing other forms are being erected.

The production of a twelve hour shift is 63 pigs, or 5,080 pounds metal, (giving for the furnace yield of the residue 40.32 per cent), and 1,750 pounds of first matte, besides slag.

The slag lead shows the following composition on analysis:

Arsenic	0.00125	per cent.
Antimony	0.00119	per cent.
Silver	0.00564	per cent.
Copper	0.08544	per cent.
Iron	0.00334	per cent.
Zinc	0.00458	per cent.
Nickel	0.00519	per cent.
Lead, by difference	99.89337	per cent.
	100.00000	per cent.

The first matte is composed as under:

Silica	19.755	per cent.
Alumina	8.549	per cent.
Lime	11.931	per cent.
Magnesia	2.701	per cent.
Ferrous oxide	9.328	per cent.
di-Ferrous sulphide	26.133	Matte proper.
Ferrous sulphide	13.861	
Nickel sulphide	3.387	
Cobalt sulphide	1.842	
Lead sulphide	1.404	
Copper sulphide	.726	
Zinc sulphide	.423	
	100.095	

The matte contains 2.196 per cent. nickel, and 1.194 per cent. cobalt. Minger's determinations gave 2.486 nickel and 1.195 cobalt.

William C. Minger, in this laboratory, determines the slag to have the following rational composition, (including some mechanically mixed matte):

Silica	53.435	per cent.
Alumina,	6 219	per cent.
Lime	15.046	per cent.
Magnesia	8.633	per cent.
Potassa	0.708	per cent.
Ferrous oxide	0.988	per cent.
Cobalt and nickel sulphides	0 793	per cent.
Zinc sulphide	0.436	per cent.
Diferrous sulphide	13.576	per cent.
	99.834	per cent.

After deducting the mechanically enclosed sulphides, it will be found on calculation, that the oxygen of the acid and bases stand to each other as 10 (nearly) to 4.5, and the slag is nearly a true bi-silicate. The small amount of ferrous oxide is unaccountable, unless the run producing this slag has been made with slag from the same furnace as flux, and with but little hematite. The absence of lead oxide among the bases is also noticeable.

The slag is thrown aside, excepting on occasions when its presence is deemed necessary in the charge.

The first matte is roasted in heaps, containing from fifty to seventy-five thousand pounds. The time required for roasting is from four to six weeks, and the fuel consumed is 5 cords of wood and 300 bushels of charcoal. The roasted matte has the annexed composition:

Nickel sulphate	0.186
Lime sulphate	5.442
Ferrous sulphate	1.338
Silica	7.804
Alumina	2.221
Nickel sulphide	5.631
Cobalt sulphide	3.390
Cuprous sulphide	1.908
Diferrous sulphide	31.041
Tetra-ferrous sulphide	1.154
Ferric oxide	39.877
	100.000

The roasted product is passed to a second smelting in a cupola furnace, giving a concentrated matte and slag. The second or concentrated matte has been analyzed with the following result:

Nickel sulphide	8.310 per cent.
Cobalt sulphide	3.424 per cent.
Cuprous sulphide	.371 per cent.
Zinc sulphide	.930 per cent.
Tetra-ferrous sulphide	20.361 per cent.
Diferrous sulphide	57.969 per cent.
Silica	.884 per cent.
Lime	4.653 per cent.
Alumina	2.659 per cent.
Magnesia	.776 per cent.
	100.337 per cent.

It contains 5.338 per cent. nickel, and 2.220 per cent. cobalt. With this matte there is a further treatment, consisting in two roastings and a cupola smelting, which yields, finally, a concentrated matte, which is claimed to yield from 10 to 18 per cent. nickel and cobalt.

and which is passed to market. The flux used is a silicious limestone and some hematite.

The further separation of the nickel compounds and the production of commercial nickel is carried on at Camden, New Jersey, and at some European localities, among which may be named Birmingham, England. The processes are complicated and usually kept secret. They have no special interest in this connection, since they form no part of the industry of Missouri.

The following will exhibit the work of the furnace at Mine la Motte for six weeks (ending as below) in the year 1875:

Date.	Mineral delivered to furnaces—lbs.	Pigs produced (80 lbs. each.)	Matte produced—lbs.
November 20	216,071	1,750	18,000
November 27	198,214	1,497	10,000
December 4	180,356	1,265	18,000
December 11	162,500	1,302	3,000
December 18	225,893	1,824	16,000
December 25	191,963	1,571	15,000

Each one thousand of mineral is really 1120 pounds, or a "miner's thousand." The matte is all first run—not concentrated.

In the same period, the following number of cars of vein-matter were crushed, producing the accompanying weights of clean or dressed mineral:

Date.	No. cars crushed.	Clean mineral produced—lbs.
November 20	229	77,492
November 27	187	220,908
December 4	217	158,806
December 11	212	75,228
December 18	207	237,186
December 25	162	232,360

The cars for transporting the ore from the mines to the dressing works, carry from two to five tons, and are moved by gravity.

The capacity of the reduction works is not adequate to the handling of the ore produced by the mines, the latter accumulating rapidly. The metallurgical processes and apparatus are hardly settled upon, the work being, as before stated, largely experimental.

CHAPTER III.

ECONOMICAL CONSIDERATIONS OF THE LEAD EXTRACTION METHODS.

The difficulty of reaching correct conclusions respecting the economics of metallurgical operations, where work is conducted empirically more than in accordance with technical principles, is almost unsurmountable. At the best, only approximations can be made; and for these, even, many of the data are necessarily derived from sources which are not and cannot be positive, since they are often merely loose estimates based on judgment or prejudice, and uncontrolled by strict business system. Where the disposition to impart information exists, (and the writer would here state that in nearly all cases, in this State, he has found a most commendable willingness to furnish information concerning the operation of lead furnaces, which he fully appreciates, and for which he returns his sincere thanks,) there is not always the ability to furnish it in the detail which is necessary to reach even close approximations.

It is a matter of serious question whether the economical conditions of lead smelting in Missouri have ever been examined to the extent which the importance of the subject warrants. Abundant distribution of ores of great purity and of high grade, simplifies in this State, what is otherwise a complex metallurgical problem; cheap and abundant fuel and labor—the latter, not infrequently, far from being skilled, but still regarded as adequate to the mechanical operations of a reverberatory or a hearth furnace—leave abundant margin for profit, engendering the belief that the best possible is being done with the business, and preventing a consideration of those sources of possible wastage, which, if examined, might lead to improvement and thence to increased financial remuneration.

In attempting the following analysis of the surroundings of lead smelting in Missouri, and the comparison of these with those obtained elsewhere, we are met at the outset with the difficulty that no one establishment has furnished full particulars. Hence we have been forced to consider no one furnace, but to average, as it were, the details and results of all those of which we have been able to gather any particulars, and which we have detailed in our illustration given in the preceding section. The conclusions reached are, therefore, not those for any special establishment, but they may be of some value in relation to the general aspects of lead business, though even for this they cannot be positive. It will, therefore, be understood that they are not put forward as absolutely correct, but only as such close approximations as could be made from the data procurable.

It is proposed to examine into value of the reverberatory method as practiced—disregarding the few examples of the Flintshire now in use—in comparison with those of other localities; to compare in the same manner the results of the hearth smeltings among themselves, and finally to inquire which of the two methods does give or can be made to give the best results with the present economical surroundings in Missouri.

Reverberatory Methods.—The illustrations of the reverberatory method which have been given are collected in the following table, in which are given the size; the weight of the charge in pouuds; time in hours required for the elaboration of the charge and the fuel consumed for the same. From the latter have been calculated the fuel and labor required to a ton (of 2,000 pounds) of ore, together with the mean results of the twelve instances given:

Recapitulation of Results with Reverberatory Method in Missouri Furnaces.

Names.	Area Hearth square feet..	Area Grate square feet..	Charge, lbs...	Time, hours..	Wood per charge, cords	Fuel, per 2000 pounds ore...	Labor, days..
1. Lone Elm	....	....	1,500	8	0.8	1.20	2.62
2. Dade county	....	....	1,200	8	0.4?	0 66	3.32
3. Pioneer	34	9.15	1,800	12	0.75	0.84	1.66
4. Eagle	35.3	10	1,500	12	0.63	0.84	2.00
5. Gum Spring	....	....	1,500	24	1 25	1.66	4.00
6. Buffalo	....	....	1,500	12	....	....	2.00
7. Star	28.5	14	1,500	8	....	....	1.31
8. Wyan Spring	52.5	12	1,800	24	1.13	1.26	3.33
9. Bonds	32	18	1,500	24	0 75	1.00	4.00
10. Linn Creek,	32	9	....	......	....	....	....
11. St. Joe	42	20	2,000	8	0.75	0.75	2.00
12. Granby.	27	15	1,500	12	1 50	2.00	2.00
Mean						1.13	2.57

Size and Weight of Charge.—The mean area of the hearths of reverberatories in the above list is 35.66 square feet; on this hearth is charged an average weight of about sixteen hundred pounds of ore, which as far as can be ascertained, has an assay value of 78 per cent. metallic lead. The lead furnace (Flintshire) at Bagilt, near Holywell, Flintshire, from the dimensions given by Percy,* has a hearth area of 90.25 square feet, and is charged with 2,352 pounds of ore (with draughtage calculated to cover the moisture of the ore,) with an assay value of from 75 to 80 per cent. lead (say 77.5.) A furnace of similar style, cited by Rivot,† has a hearth area of 86.5 sq. feet with a charge of one ton (2,240 pounds) of 78 per cent. ore. The Bleiberg furnace, which approaches more closely the ordinary air furnace in use in this State, both as respects size, and in regard to the method of work, has, according to Rivot,‡ a hearth area of 54.5 square feet, with a charge of 456 pounds, yielding, by fire assay, 65 to 72 per cent. metallic lead (say, 68.5.)

The fire boxes of the several furnaces named above are as follows:

Missouri Reverberatories	13.37 square feet.
Flintshire at Bagilt	38.25 square feet.
Flintshire (by Rivot)	9.79 square feet.
Bleiberg	6.84 square feet.

From which the following proportions between the areas of the fire box and hearth are established:

Missouri	1:2.6
Flintshire	1:8.
Flintshire	1:9.
Bleiberg	1:8.

Fuel consumption: In the treatment of one ton of ore the Missouri air furnaces consume on an average 1.13 cords of wood. A cord of oak wood of the mixed varieties delivered to the furnaces and as usually piled, weighs about 3,800 pounds. In smelting one ton of ore there are, therefore, required very nearly 4,300 pounds of wood. The two examples of the English furnaces that we have selected require, respectively, 1,333 and 1,050 pounds of good quality coal for the ton (of 2,000 lbs.) of ore put in treatment. Karsten estimates that, in reverberatory effect, one part by weight of coal is equal to 2.6 parts by weight of good wood (probably too low an estimate for a comparison between the good quality English coal and the imperfectly sea-

---

*Metallurgy of Lead, pp. 222, *et seq.*

†*Traite de Metallurgie*, II., pp. 319, *et seq.*

‡*Ibid*, pp. 299, *et seq.*

soned mixed woods furnished the furnaces in Missouri), on which basis the weights of coal named will be equal to 3,466 and 2,730 pounds of wood respectively. The Bleiberg furnace consumed 32,000 pounds of well seasoned fir-wood.*

In so far as the consumption of fuel to a given weight of ore treated is concerned, better results are obtained with the foreign furnaces than with those worked here, and in those furnaces, it will be remembered, the hearth areas are about eight times greater than the fire box areas. The yield of lead will also influence these results.

Labor: The table gives as a mean amount of labor required for the treatment of one ton of ore in the Missouri air furnaces, 2.57 days —skilled and ordinary labor both included. The two Flintshire furnaces require of all kinds of labor, and for the ton of ore, 1.36 days, while the Bleiberg necessitates 3.98 days, this latter being increased by the fact that the residues are also worked up in the same furnace, and their treatment is included in the cost of the ore smelting.

The furnaces are, therefore, more expensive in labor, demanding nearly twice as much of this item of cost as the large English furnaces, though not so much as the Carinthian, in which the whole method of treatment is shaped rather to economy of fuel than saving of labor. The Flintshire furnaces look rather to economy in expenditure of labor than to saving in cost of fuel. To be in strict accord with the conditions surrounding metallurgical industry in this State, expenditure of fuel should be subordinated to expenditure of labor.

Furnace yield of ore: The average of the ore put in treatment and handled in the manner we have described, cannot have a furnace yield greater than 63 per cent., though this point is difficult to reach with much precision. One ton of ore would yield 1,260 pounds of metal against 1,560 pounds determined by assay to exist in it. The amount obtained is then about 80 per cent. of the assay value of the ore. A charge of 21 cwts. of good Flintshire ore yields about 14½ cwt. of lead, of which 91 per cent. is obtained directly from the ore, and 9 per cent. from slag, etc., according to Percy, or 1,381 pounds to the nett ton, equivalent to 62.84 per cent. yield from the ore. The treatment therefore recovers 81.6 per cent. of the lead, which fire assay shows to exist in the ore. The example given by Rivot, shows a recovery in the reverberatory of 80 per cent. of the lead existing in the ore, while the Bleiberg furnaces give a 60 per cent result, corresponding to 87.6 of the assay value of the ore.

The loss in treatment, from all sources, in the several practices of reverberatory smelting named are 20, 18.4, 20 and 12.4 per cent, respec-

---

* The results are obtained from the several authorities mentioned, calculations being made to reduce to the ton of 2,000 lbs.

tively, of the amount of lead which fire assay shows to exist in the several lots of mineral put in treatment. The Bleiberg practice includes a residue treatment in the same furnace by the same workmen and is therefore much less than the other, and is hardly comparable. The others it will be seen differ but slightly among themselve, and may be, for all practical purposes, regarded as the same for both the ordinary Missouri air-furnace and the Flintshire furnaces.

The examples of English smelting show that the per centage of residue yielded by a charge of ore, range between 11¾ and 25, with a mean of 17.3, and that the assay values of those so-called slags are between 40 per cent. and 55½ per cent. with a mean of 51 per cent., as determined by crucible assay. The mean composition of the twelve samples of residue, the results of the analyses of which have been given. on preceding pages, so far as the lead compounds are concerned, is as follows:

Lead sulphide	29.42	per cent.
Lead sulphate	2.89	per cent.
Lead oxide	39.74	per cent.
Metallic lead	56.88	

These estimations having been made in the humid way are higher by about five per cent. than results which would have been reached by the most carefully conducted dry assays. The mean amount of lead in our slags, therefore, does not differ greatly from the average metallic content of the residue from the English reverberatories. Hence it may safely be assumed that the amount of residue drawn from the charge of our air-furnaces is about the same as that drawn from Flintshire, namely, about seventeen per cent. of the weight introduced.

These figures show that of the total lead value of our ores, 80 per cent. is recovered in the reverboratory furnance, 11.25 per cent. passes into the residue, from which, at those cases where slag-lead smelting practiced it is partly recovered as slag lead, and 8.75 per cent. is lost, chiefly in fume and in furnace bottoms.

So far then, in the common Missouri reverberatory practice, the results are equal to the best English examples, but they are not attained with the same regard to economy in labor and fuel. The difference in these important points must be looked for either in the form of the apparatus used for treatment, or in the capacity of the workmen, for these can be the only variables in the treatment of similar ores in different furnaces and with workmen of different energies.*

---

* This point is well illustrated by Moisenet. *Traitment de la Galene au four Gallois, Ann. des mines,* Tome 1, 1860.

Representing the weight of a charge in pounds by c, the number of hours required for its elaboration by t, and the weight of ore treated in twelve hours, that is to say the activity of the furnace, by w, then $w=12\frac{c}{t}$, and the following values will be obtained for w in the several furnaces we have been comparing:

	Value of w.	Weight treated per man per 12 hours.
English	3795	1897
Missouri	1800	900
Bleiberg	475	237

The smelter at the English furnaces treats, therefore, in a given time, more than twice as much ore as the American and eight times as much as the German smelter. All this difference cannot be accounted for reasonably by the difference of energy of the workmen at the different localities, but must be looked for evidently in the construction of the furnaces.

Examining the proportions existing between the hearth and fire-box areas of the different furnaces which have been put in comparison, it will be seen that our air-furnaces have much less hearth surface in proportion to the surface of the horizontal section of the combustion chamber than any of the others. A remedy for the apparent difference between the working effects of the English and American smelter may be found in an increase of the size of the hearth till the ratio between the two areas approaches more closely that obtaining in the cases of the foreign furnaces, whereby the charge could be either increased in weight, or this remaining constant, a thiner layer would be spread over the sole. The effect of this would be more thorough oxidation and a shortening of this period as well as that of the reaction proper, by which the charge would be as thoroughly worked off in a shorter time, and the furnace activity be thereby increased. This view, though hypothetical, would seem to be warranted by those experiences cited elsewhere, and may therefore be worthy of attention from those who may appreciate the importance of decreasing the expenditure for labor required for the production of lead. It is also strengthened by what is shown in the composition of the residues, these latter showing an average of 29.42 per cent. of lead sulphide or unchanged ore, and only 2.89 per cent. of lead sulphate, showing that oxidation has not been thorough, owing, possibly, to a too highly heated hearth and a consequent partial fusion of the ore. With such conditions rapid elaboration of a charge is impossible. Tookey's results* show for the residues or gray slag from the Flintshire furnace

---

* Percy, *loc. cit.*, page 235.

0.9 per cent. sulphide, and 9.85 of lead sulphate. It may be added that in these residues nearly all the silica was found to exist in combination as a silicate decomposable by acid with the separation of gelatinous silica. It is most probably in union with lead oxide, a condition which would imply considerable action of the charge on the furnace lining, also a result of the too elevated temperature of the hearth, for the average ores are hardly silicious enough to account for this silicate of lead.

There being so many other items entering into the total expenses of furnace operations, it is deemed inadvisable to attempt, with the data at hand, a summing up of the cost of treatment. The amounts of the two chief items, labor and fuel, have been indicated and will be found to be very close to the condition realized at the Missouri furnaces. The items of wear and tear and interest are exceedingly variable, and besides could not be reached with even approximate accuracy.

Hearth Methods—Two classes of ore hearths are in use; those with a single tuyere, and those larger and with three tuyeres, or, as we have distinguished them, the older and the newer patterns.

The accompanying statement of the results of the illustration of hearth methods includes the (calculated) amounts of labor and fuel expended in the treatment of one ton of ore (2,000 pounds). In the case of the Valle hearths the fuel includes the estimated charcoal corresponding to one-twentieth of a cord of wood.

RECAPITULATION OF RESULT WITH HEARTH METHODS.

NAME.	No. Tuy.	Charge—pounds.	Labor—days.	Charcoal per charge—bushels.	Labor per ton—days	Charcoal per ton—bushels.
Granby................	3	3000	3	9.2	2.0	6.13
Lone Elm............	3	3000	3	15.0	2.0	10.00
La Motte..............	3	3200	2.5	8.3	1.56	5.19
O'Brien	1	2000	3	?	3.0	?
Hopewell............	1	2731	3	?	2.2	?
Perry	1	3000	3	10.0	2.0	6.68
Valle....................	1	3500	3	9.3	1.7	5.06

Comparing together the results of the old and new forms it will be seen that the former consumes 5.84 bushels of charcoal per ton, while the latter requires 7.44 bushels. But this increased consumption of charcoal which the three tuyere furnaces show, and which would be naturally expected, is offset by the diminished amount of labor, the new form requiring 1.85 days labor, against 2.22 days labor in the treatment of one ton ore in the hearth with the single tuyere. The items of fuel do not include any consumed for the blowing engines.

There are few data extant for comparing the results above given for the single tuyere American hearth, with those at any other locality in this country. The works on metallurgy contain statements of the cost of hearth treatment at Rossie, N. Y., which show a fuel consumption of 0.2 cord of wood per ton of ore, corresponding to a somewhat greater number of bushels of charcoal than that given above.

The figures we have collected show a mean yield of the ore in all the hearths of about 67 per cent. of the ore put in treatment, which has probably an assay value of 78 per cent. The amount of lead saved is 86 per cent. of the assay value. The Mine la Motte results give for the yeld of residue 13.5 per cent. of the ore treated, or 270 pounds weight per ton of ore. The average composition of these residues we find to be as follows:

Lead sulphide	23.86	per cent.
Lead sulphate	1.83	per cent.
Lead oxide	28.60	per cent.
Total Metallic lead	48.8	

The total lead is the result of determinations, all made by humid methods, and are at least five per cent. excessive over the results by fire assays, so that the assay value of the residues would be, say, 45 per cent. The slags from a ton of ore would contain 121.5 lbs. of lead, equivalent to nearly eight per cent. of the total lead shown by assay to exist in the ore. This is partly recovered in those cases where slag treatment is resorted to. The loss from all other sources—(fume, dross, etc.,) is therefore four per cent. of the total content of metallic lead.

Comparing these results with those calculated for the existing reverberatories of the State, it is evident that, as far as the mere operations are concerned, the hearth is somewhat less expensive both in labor and in fuel than the reverberatory. Taking the cost of these items at $2.75 per day for the labor, $2 per cord for wood, and 12 cents per bushel for charcoal, the costs per ton for ore treated, will be:

	Labor.	Fuel.	Total
Reverberatory	$6.97	$2.26	$9.23
Hearth	5.58	62	6.20

But it must be remembered that the hearth method entails additional expenses for labor and fuel for the blast, which may be roughly estimated at a total of $1.80 per ton of ore, which will increase the cost of treatment to $8. Further, the much more expensive character of the plant of the hearth, above that required for the reverberatory, will increase the aggregate by the greater item of interest. On the other hand, the wear and tear is less in the hearth, and the yield is greater.

In spite of these offsets, there is very little difference in cost of treatment or production now obtaining, and if improvements were made in the dimensions of the reverberatories, the advantages would be decidedly in favor of this method of treatment. This view is confirmatory of that taken by Dr. Percy (in criticising the comparative results in treating similar ore in the reverberatory and the ore hearth at the Greenside mines in Westmoreland, England) who states: "The wear and tear in ore-hearths is certainly less than in reverberatory furnaces; but probably insufficient to cover the considerable balance against ore-hearths, so that we may expect to see the former displace the latter in all but few localities, where coal cannot be got at moderate prices."[*]

While on this subject, it may be interesting to call attention to the results obtained by Mr. E. D. Peters[†] at the Mount Lincoln Works, Colorado. With a reverberatory, with a hearth 15 by 9½ feet, and with two and three-fourths ton of ore for a charge, the cost of treatment we find to be 1.11 days, and the fuel, 1.66 cords wood. It is noteworthy that the ore was more silicious than calcareous, and therefore of a character which, according to preconceived notions, was not well adapted to reverberatory treatment.

The cost of slag treatment has been shown to be about 0.4 day for the labor, 266 pounds of coke and 30 pounds of charcoal for the ton of residue treated, independently of the cost of the wages of the engineer and the fuel for the engine. With the same prices for labor and charcoal that have been assumed, and with a cost of twenty-five cents per bushel for coke, the items will aggregate $3.18.

For the ton of residues at the (English) Flintshire furnaces, Rivot's estimate of the special expenses are:

Workmen	1.907 days.
Coke	627 pounds.
Coal for blowing engine	367 pounds.

At the same rates for the first two items, the expenses will aggregate $9.64. The labor probably includes, however, that of the engineer

---

[*] *Loc. cit.*, p. 285.

[†] *Transaction Am. Inst. Mining Eng.*, vol. 2, pp. 310.

and fireman. The estimated cost of the proportion of the times of these added to the expenses above given for the Missouri slag furnaces will give a total of $4.28.

As was stated, the system practiced in this State is an improvement over that obtaining in the north of England; the expenses in treatment of the residues being scarcely half as great as those for the English method of work. The yield of lead is, however, greater in the latter than in the Missouri, this producing about 23 per cent. lead on the average, while that gives 35 per cent. The difference in the cost is lessened when assessed on the ton of slag lead produced. The costs are then as $18.60 to $27.68.

This reduced cost in treatment will show that in the plan of residue treatment practiced in Missouri, the wastage of lead in the slag is more apparent than real. The value of the lead is more than compensated for in the reductions in the costs of labor and fuel, to say nothing of the additional expense of flux. It is probably more nearly in accordance with the conditions of our local metallurgy than any of the present plans of direct treatment of the ore.

The details we have given enable us to calculate with some approximate accuracy at least the special expenses attendant upon the complete treatment of a ton of ore, including the residue as well as those necessary to the production of a ton of metal, including that from the residue. These will be found to be as under:

Reverberatory treatment of one ton mineral	$9 23
Slag treatment of 340 pounds residue	72
Total	$9 95

The yield will be:

From the reverberatory	1260 pounds lead.
From the slag	78 pounds lead.
Total lead recovered	1338 pounds.

or 66.6 per cent. of the ore, equivalent to 86.8 per cent. (nearly) of the assay value of the lead. The cost of one ton of lead is therefore $14.88, exclusive of the general expenses of the establishment.

The apparent loss of lead from all sources, according to these calculations, is 13.2 per cent., while the real loss, taking into account the actual value of ore in lead, determined by exact analytical processes, cannot be less than seventeen per cent. of the metal. This difference between the real and apparent loss is certainly all assessable on volatilization, but does not cover the full wastage from that cause. The attention of smelting establishments in the State should be directed to this loss and more adequate means be taken to provide against it. At present the waste is almost entirely neglected in Missouri.

By including the cost of residue treatment, we are in condition to make a more just comparison of the costs of treatment in the Missouri air-furnace and the Bleiberg form, which it will be remembered includes a working of the slags in the same furnace. With the assumed prices for labor and fuel the aggregate of expenses for the Bleiberg furnace will be found to be $12.62 per ton of ore, yielding in the furnace 60 per cent., or twelve hundred pounds of metal. The cost of production is, therefore, $21 per ton of resulting metal. The yield is 87.6 per cent. of the assay value of the lead in the ore. Evidently the elaborate and close work of the Carinthian furnace, expending such an excessive amount of labor, is not adapted for profitable employment in this State, and cannot compete with our present wasteful system with existing high prices of labor, though relatively cheap fuel can be obtained.

These estimates confirm the views that have been advanced respecting the reverberatory system practiced in Missouri. It is evident from them that all that remains to be done is to increase the activity of the air-furnaces, which result can be satisfactorily reached only by increasing the size of the hearths. If the reader will refer to the recent reverberatory experience in Colorado, noted in this section, he will find that the labor per ton of ore is reduced to 1.11 days, while the fuel is only 1.66 cords, by having a hearth with 142 square feet area, which accommodates two and three-quarters tons of ore at a charge. Obviously a step in the right direction is being made by the introduction in this State of furnaces modeled more upon the type of the Flintshire or English reverberatories, which we have seen give such excellent results. The Granby furnaces of this pattern, the new furnace at the Des Loge works, and that at the Frumet, appear to us, on theoretical grounds, to be a decided advance toward greater economy of treatment, though it is to be regretted that, through want of data, we have not been able to examine into this matter as thoroughly as the importance of the subject demands.

ADAPTABILITY OF MISSOURI LEAD TO CERTAIN INDUSTRIAL USES:

The character of the ores of lead and the metallurgical treatment that they receive, result in the production of a metal which experience has shown is well adapted to the manufacture of white lead. Lead suitable for conversion into this important product is usually denominated "soft lead"; it must be capable of being rapidly corroded, that is, of being converted into basic carbonate of lead, under the conditions to which it is exposed in the stack, and of yielding a product which is of pure white color. Commonly in the history of the white-lead manufacture in the eastern states, the preference has been given to imported leads, more especially to those of continental Europe. These have generally resulted from three fining of more impure or work leads, incidental to the extraction of silver. The process now usually adopted for this latter purpose, consists in melting the silver-bearing lead with metallic zinc, whereby the silver is largely alloyed with the zinc, from which it is separated by distillation or otherwise. The details of the process are not necessary in a report of this character; but they result in the production of leads of remarkable purity, which have been and are still, deservedly in high repute among white lead corroders.

Missouri leads are not of a character to warrant any such treatment—not being as we have seen, argentiferous. Nevertheless they have given excellent results on corrosions, both in respect to degree of corrodibility and to character of product, and have been long used among white lead manufacturers in St. Louis. Since the beginning of the year 1875, they have also found their way into eastern markets, coming into successful competition with well established European brands, and in fact, largely displacing them. This has been effected in the face of difficulties arising from or encouraged by prejudice, and from the fact that, though the Missouri leads were generally of lower market rates, manufacturers are not disposed to change the routine of their process, or are fearful of destroying what may have been recognized as a distinguishing feature of their product.

Recognizing the value of Missouri lead as a raw material for white lead manufacture, the writer of this report, as early as May 1875, called attention to the matter in the following introduction to a paper on the composition of Missouri leads:

For the purpose of White Lead corrosion, some of the brands of European leads have secured a prominence, and are preferred, notwithstanding their relatively high prices. How far this preference is justified by the inherent qualities of the lead, evidenced in the results of corrosion obtained, or how far it may be the result of a prejudice, perhaps, created and fostered by trade manipulations, are questions difficult of settlement. The imported and prized leads are generally the products of refining processes to which the Missouri metal cannot be economically submitted because unwarranted by a sufficient amount of silver. On the other hand, the established purity of the ores of the lead districts of Missouri should admit of the production of a metal which by a simple process of so-called refining and improving, or rather softening, ought to approximate at least to the purity of the much prized imported brands. The circulation, therefore, of correct knowledge of the composition of the pig leads produced from the ores of the great lead area of the Mississippi Valley regions must be of value if they should be able to show that much of the prejudice against the domestic and in favor of the foreign lead, is ill-founded or is based only (as may be the case with white lead manufactures) on traditional results of corrosions obtained. On this point a recent writer on the metallurgy of lead* gives the following: "The best white lead, i. e., the whitest, was supposed by many to be made from what is known in the market as 'W. B. selected and refined pig lead,' (the initials being those of Walter Blackett, the producer, and now applying equally to the name of his successor, Wentworth Beaumont.) This pig lead is the produce of certain mines on the Greenwich Hospital estate, in the north of England, which are leased to Mr. Beaumont. It fetched a higher price to the amount of from 5 p. c. to 7½ p. c. than any other pig lead. So great was the prepossession in favor of this pig lead for the mauufacture of white lead, that attempts to persuade London manufacturers to make a trial of the best and purest soft lead from other sources were fruitless, notwithstanding it was offered at about 30 s. per ton, i. e., 7½ p. c. less than W. B. lead. Commercial competition, however, seems at length to be uprooting prejudices which, as in the case in question, have probably no better foundation than traditional credulity. It is to be hoped that now-a-days the world is growing wiser, and will judge an article of manufacture by its quality and not by its pedigree. Many illusions connected with articles of manufacture have been dispelled, in recent years, with much gain to the public, and possibly it will be found that plenty of lead is produced in Great Britain and elsewhere, quite as suitable for conversion into white lead as that bearing the mark 'W. B. selected and refined.'"

Since the analytical results which were introduced by these remarks were made public,the prejudice against our leads has disappeared very rapidly before a knowledge of their composition; but it may not be out of place in a report of this character to go further into a consideration of this important subject.

Of the foreign metals present, copper, iron and nickel forming colored compounds may be regarded as those which may most seriously impair the color of the white lead. But it cannot be admitted that the minute amounts of these metals which are present in any soft lead, or in any lead which would be selected by an experienced corroder, would be powerful enough to affect color to the unlimited extent which has been claimed for them. Their potency is greatly overrated in this respect. In fact the whole matter of impaired color of

---

* *Percy's Metallurgy*, p. 508.

corrison is one which is by no means settled. The rose tint frequently noticed was ascribed to copper oxide, and to iron oxide, but in 1869, W. Baker claimed that it should be ascribed to silver, which, if present to the extent of 1–13 of an ounce to the centner of the corroded lead, would tint the whole product of white lead.* Bannow and Kràmer† ascribe the same coloring to the presence of lead suboxide formed by imperfect ventilation of the stack during the corrosion period. Lorscheid,‡ to the same compound existing through insufficient carbonic acid in the stack.

The annexed table includes all the results of the analyses which have been given on the preceding pages, together with results of analytical investigations into the composition of some other American leads and a few produced at foreign establishments. The latter are all well known and favorite brands. The leads from Missouri furnaces, it will be seen, are of average purity with those others that are given. They contain no bismuth, but have the other foreign matters present in no greater amounts than the favorite brands.

---

* *Phil. Magazine*, May 1869, p. 324.

† Jahresbericht für Technologie, 1872.

‡ *Ibid.* 1873.

TABLE showing the Composition of Commercial Leads produced in Missouri,

BRAND.	FURNACE METHOD.	ANALYST.
Granby	Reverberatory (Flintshire)	This Report
Granby	Reverberatory (ordinary)	"
Lone Elm	" "	"
Dade County	" "	"
Pioneer	" "	"
Eagle	" "	"
Buffalo	" "	"
Star	" "	"
Wyan Spring	" "	"
Bond's	" "	"
Linn Creek	" "	"
St. Joe	" "	"
Frumet	" "	"
Frumet	" "	Chauvenet and Blair
Granby	Hearth	This Report
Hopewell	"	"
Perry	"	"
Vallé (Vallé Furnace)	"	"
Rozier (Vallé Furnace)	"	"
Mine la Motte	"	"
Granby	Cupola	"
Phœnix (Vallé Furnace)	"	"
Mine la Motte	"	"
St. Louis Smelting and Refining Co.	Refined from Territorial Work Lead.	"
Pittsburg (*a*)	Refined from Territorial Work Lead.	Dr. Wuth
Lautenthal (*b*)	Refined from Clausthal Lead	Hampe
Lautenthal (*c*)	Refined from Clausthal Lead (1870)	Schollmeyer
Altenau (*d*)	Refined from Clausthal Lead (1871)	Hampe
Pontifex & Wood		Laboratory at Clausthal.
W. B —best selected		" "
Stolberg		" "
Eschweiler Company		Fresenius
Pirath & Jung, Commern		"
Real Compania, Asturania		"
Herbst & Co., Call—1867		"
Mechernich—1869		"

a—Cited by Eilers, *Trans. Am. Inst. Mining Eng.*, Vol. III, p. 322. *b* and *c*—Cited by Eilers, *Raymond's Report*, 1874, p. 445. *d*—*Ib.*, p. 454—produced by poling lead obtained in the treatment of rich foreign silver ores and dressed lead ores of the Upper Harz.

together with Results of Analyses of some other Domestic and Foreign Brands.

Lead........	Antimony..	Arsenic	Bismuth....	Copper	Silver	Iron.........	Zinc.........	Nickel.	Cadmium .	Other matters..
99.95267	0.00077	0.01640	....	0.01210	0.00029	0.01711	0.00066	Trace	....	....
99.91677	0.00077	0.01122	....	0.05091	0.00080	0.01582	0.00090	0.00281	....	....
99.96340	0.00266	0.00027	....	0.00463	0.00025	0.00777	0.02071	0.0031	....	....
99.99247	Trace	Trace	....	0.00448	0.00177	0.00128	Nil	Nil	....	....
99.97469	0.00347	Trace	....	0.00346	0.00100	0.00496	0 00517	0.00125	....	....
99.99320	Trace	0.00313	....	0.00820	0.00029	0.00461	0.00057	Trace	....	....
99.98722	0.00281	Trace	....	0.01024	0.00114	0.00556	0.00136	0.00115	....	....
99.96359	0.00513	0.01086	....	0.01376	0.00029	0 00462	0.00090	0.00085	....	....
99.97786	0.00048	0.00338	....	0.01002	0.00057	0.00428	0.00139	0.00203	....	....
99.98078	0.00495	Nil	....	0.00556	0.00084	0.00411	0.00181	0.00195	....	....
99.95737	0.00639	0.01423	....	0.01511	0.00043	0.00376	0.00136	0.00135	....	....
99.93124	0.00675	0.00183	....	0.06394	0.00405	0.00137	0.00082	Trace	....	....
99.94826	0.00007	0.00004	....	0.04399	0.00179	0.00282	0.00240	0.00063	....	....
99.95700	Trace	....	....	0.04100	....	0.00200	Nil	Nil	....	....
99.96905	0.01085	0.00124	....	0 00780	0.00646	0.00367	Trace	0.00087	....	....
99.97408	0.00803	0 00583	....	0.00585	0.00219	0.00145	0.00156	Nil	....	....
99.97291	0.00520	0.00674	....	0.00409	0.00190	0.00436	0.00598	Nil	....	....
99.94548	0.00214	Trace	....	0.04165	0.00326	0.00453	0.00294	Nil	....	....
99.91019	0.00184	0.00925	....	0 03742	0.00615	0.02497	0.00118	Nil	....	....
99.96996	0.00119	0.00034	....	0.01999	0.00345	0.00248	0.00164	0.00095	....	....
99.90756	0.04976	0.00101	....	0.02965	0.00029	0.00718	0.00180	Nil	0.00296	....
99.74526	0.13365	0.01556	....	0.09323	0.00525	0.00470	0.00235	Nil	....	....
99.89337	0.00119	0.00125	....	0.08544	0.00564	0.00334	0.00458	0.00519	....	....
99.91813	0.00538	0.00066	0.03559	0.02457	0.00210	0.00819	0.00493	0.00045	....	....
99.99844	0.00051	....	....	0.00007	0.00042	Trace	0.00038	....	....	Sulphur 0 00018
99.983573	0.001184	....	0.008498	0.000954	0.000500	0.004930	0.000361	Trace	Trace	....
99.983139	0.005698	....	0.005487	0.001413	0.000460	0.002289	0.000834	0.000680	....	....
99.976719	0.005743	....	0.009580	0.002475	0.001000	0.003272	0.000140	0.000196	....	Chlorine 0.000875
99.91360	0.00320	....	....	0.07580	0.00200	0.00220	0.00320	....	....	....
99.95200	0.00590	....	....	0.03180	0.00500	0.00170	0.00370	....	....	....
99.95000	0.00500	....	....	0.03830	0.00250	0.00240	0 00180	....	....	....
99.94060	0.00210	....	0.00200	0.05010	0.00440	0.00080	....	....	....	....
99.98329	0.00921	Trace	....	0.00268	0.00275	0.00121	....	....	....	....
99.98597	0.00133	Trace	0.01041	0.00057	0.00040	0.00124	0.00008	Trace Ni. and Co.	....	Trace Co. Mn.
99.98740	0.00450	....	....	0.00510	....	0.0520	....	0.0010	....	Trace Mn.
99.99474	0.00118	Trace	....	0.00234	....	0.00090	....	0.00075	....	....

It is not unreasonable to believe that the cause assigned by Messrs. Bannow and Krämer, or something analagous to it, is adequate to account for the impaired quality—much more so than are the minute quantities of the foreign metals—and that most cases of tinted corrosions made from soft leads may be referred to internal conditions of the white lead stacks. If this view be correct, then much of the virtue claimed for extraordinary freedom from impurities disappears so far as the color of the white lead is concerned, or if still existant, finds its cause of existence in trade prejudices or manipulations, which deterred manufacturers from changing to any lead which differed, however slightly, in composition from that which they had been accustomed to employ.

The maximum amount of these coloring metals which may exist in a lead without impairing the quality of the white lead made therefrom, has not been fixed, but any lead, which would yield a product colored, say, greenish from copper compounds, would not be a soft or corroding lead. Doubtless this limit is passed in the slag leads of Missouri, (mean amount of copper, .06944 per cent.), but it is not reached in the reverberatory leads from southwestern and central ores, (with .01329 per cent. copper) for many of these have been practically tried, and besides their mean amount of copper is not much greater than the mean result of six leads analyzed by Fresenius (with .01046 copper) all of which are recognized and employed as soft corrodable leads. If the one gives an uncolored product, then the other set may reasonably be expected to give the same under similar conditions of corrosion.

The result of analysis of the lead of the St. Louis Smelting and Refining Company prepared in refining the work, lead from territorial ores, shows upwards of 0.02 of copper, and this lead has given most satisfactory results with both St. Louis and eastern corroders.

Instances of practical experiences such as these, must carry conviction that either these small per centage of certain metals do not work with prejudice to the color of the white lead, or that prepossession in favor of certain brands of lead has been created and sustained by commercial operations, and are not warranted by sufficient differences in composition.

Differences in favor of or against a brand of lead, when such small amounts are taken into consideration, may readily arise from errors inherent even in the most carefully conducted analytical investigations into their composition, or from variations in drawing samples for analysis. On this last point Schweitzer* found in the

---

**American Chemist*, June, 1876 ; p. 457.

same pig of lead, by cupelling, seventeen samples from different parts, amounts of silver ranging between 79.83 ounces to the ton and 104.93 ounces.

The other foreign metals may affect the corrodibility of the lead but on this score the knowledge with regard to the small amounts existing in soft leads is but little more extended or certain than in respect to the coloring power of the metals already named. Much that has been ascribed to difference in composition of the leads corroded is, without doubt, to be traced really to conditions of temperature and of the atmosphere in the stacks. These are matters upon which there is little if any information. It is within the experience of every manufacturer that differences in the percentage of metal corroded are as great for the same metal in different or even in the same stack as for different leads in separate corrosion heaps. Till more knowledge is obtained in regard to the thermal conditions of corrosions and the composition of the atmospheres of the stacks, this point must remain in obscurity, or be—unphilosophically it would seem—charged to minute variations in the amounts of a given foreign metal in the lead.

Antimony is generally regarded as prejudicial and bismuth favorable to rapid and thorough corrosion. The latter is rapidly oxidized if intimately mixed with the lead, but if concentrated in parts of the buckles (as may happen in casting) it oxidizes slowly itself, and prevents or retards further corrosion of the lead.* The presence of bismuth (0.03559 per cent.) in the lead of the St. Louis Smelting and Refining Company may account for the favor with which this lead has been received, and also for the prejudice in favor of certain European brands.

On the other hand, the extended and carefully conducted experiments of Hampe with Hartz and other brands of lead of recognized purity, and with prepared alloys of bismuth and lead, simultaneously corroded at the Hartz Lead Works at Osterode, do not support this view of this action of bismuth. Neither do they substantiate the opinion generally advanced that the bismuth compounds impart a tinge of gray to the white lead, which, though disappearing on grinding and levigation, nevertheless, cause the white lead to turn yellow in the dark.†

Some diversity of opinion exists concerning the action of anti-

---

* Endeman on *Bismuth in lead in the Manufacture of White Lead.* American Chemist, June, 1876.

† A translation of Hampe's paper (by Prime) can be foand in American Chemist, June, 1875.

mony. This metal is generally regarded as retarding corrosion in white lead stacks. Hasenclever † shows that it has such effect in the lead pans used for evaporating sulphuric acid. The rapid wear of such pans is due to the greater purity of the metallic lead; antimonial lead lasts much longer. On the other hand, common experience with metallic constructions shows that small amounts of foreign metals hasten oxidation.

On the whole, the subject of the conversion of lead into white lead by the so-called Dutch method is but little understood, and many so-called explanations of unsatisfactory results are the vaguest hypotheses. The experiences of eastern corroders have been fully as satisfactory with the Missouri soft leads as with European leads longer in use.

Manufacturers generally hold that the residual blue lead, or kernels remaining uncorroded, contain the foreign metals of the lead in increased amount. This reputed concentration has not been established, and the matter is one not easily settled. Hampe writes concerning this: "Finally, with regard to the frequently imputed enrichment of the foreign metals in the metallic residue (the so-called lead kernels) obtained in the preparation of white lead, it is impossible to give a definite answer to this from the following analyses. It must, however, be borne in mind that such pure leads are not well adapted to settle this question, since the differences are so small that it is impossible to say whether they are essential, or whether they are due to the uncertainty caused by the taking of samples for analysis from the residues. An enrichment of the bismuth in the finely divided lead was found only in the bismuth alloys after separating the residues from the white lead by washing."

The result of our analysis of a sample of unwashed white lead manufactured in St. Louis is given below, (II), as well as the composition of the lead from which it was corroded, (I), and of the blue lead or kernel separated from the same (III). The foreign metals only are given.

Missouri Lead and Products.

	I.	II.	III.
Arsenic	0.00019	None.	0.00217
Antimony	0.00198	0.00076	0.00158
Silver	0.00045	0.00034	0.00050
Copper	0.00479	0.00926	0.01381
Iron	0.09020	0.00085	0.00315
Zinc	0.00142	0.00035	0.00770
Nickel	0.00047	0.00007	0.00055

---

† Berg and Huettenmann. Zeitung, Nov. 30, 1872; quoted by Hampe (as above.)

Some of the results obtained by Hampe in the experiments above alluded to, are appended for comparison.

LAUTENTHAL PRODUCTS (Hampe.)

	Original.	I. Oxidation White Lead.	II. Kernel from Oxidation.
Bismuth	0.008498	0.006276	0.011091
Copper	0.000954	0.000431	0.001149
Antimony	0.001184	0.000903	0.001334
Silver	0.000500	0.000500	not est.
Iron	0.004930	0.000728	trace.
Zinc	0.000361	0.000128	trace.
Cadmium Nickel	trace.	trace.	

SILESIAN PRODUCTS (Hampe.)

Copper	0.0012	0.000566	0.000938
Bismuth	trace.	trace.	trace.
Cadmium	0.00046	0.000360	0.000500
Antimony	0.00131	0.000444	0.001388
Silver	0.000166	0.000130	0.000166
Iron	0.00120	0.000903	0.000360
Zinc	0.00035	0.000257	0.00031

MECHERNICH PRODUCTS (Hampe.)

Bismuth	0.00022	0.000197	0.00024
Copper	0.00051	0.000408	0.000411
Cadmium	0.00037	0.000200	0.000450
Antimony	0.00320	0,002218	0.00310
Silver	0.00032	0.000300	0.00038
Iron	0.00075	0.000937	0.000634
Zinc	0.00052	0.000337	0.00020

WHITE LEAD MANUFACTURE IN ST. LOUIS.—The census of 1870 makes no mention of this industry in St. Louis, though the business had then attained such magnitude that the value of the product is estimated by Mr. Cobb to have exceeded one and three-quarter millions of dollars. In 1875 the production was, according to the best estimates, 11,200 tons, an increase of about ten per cent. over that of 1874. According to the reports of the St. Louis Exchange, the white lead industry in 1875 had the following status in St. Louis:

Number of establishments	6
Capital invested	$2,033,000
Cost of raw materials	1,196,000
Value of product	3,292,500

Litharge and red lead or minium (oxides of lead) are incidental products of the white lead establishments. They find consumption in a number of industrial operations, notably in the manufacture of glass and in connection with the painter's trade.

For glass-making purposes, the lead oxides must be as free as possible from those foreign oxides which have coloring properties, especially oxide of copper. In the English markets the red-lead from Snailbeach metal finds preference among glass-makers, and brings, by five per cent., a higher price in the market. It contains from 0.0021 to 0.0065 of copper, the mean of five results by Percy, being 0.00492 per cent. No nickel or cobalt were detected, though especially sought after.

The following determination of the amounts of foreign metals in a sample produced in St. Louis, from Missouri lead, will be of interest in this connection as well as of value as collateral testimony bearing on the questions of the amount and effects of the impurities in Missouri soft leads:

Foreign Metals in Red Lead from Missouri Metal.

	Per cent.
Iron	0.10640
Zinc	0.04460
Copper	0.00207
Antimony	0.00920

besides traces (in one hundred grammes) of arsenic and cadmium.

The growing favor that Missouri soft lead is finding among the corroders of the eastern cities is shown by the fact of the increase of shipments of lead eastward during the last three years. In 1873 the shipments exceeded those of 1872 by 153,178 pigs; 1874 showed an increase in the same direction over 1873 of 2,498 pigs, while the amount of 1875 exceeded that of 1874 by 102,135 pigs.

Lead for Mechanical purposes: Mr. Henry Cobb* estimates that from 1858 to 1873, the St. Louis shot tower, produced $9,000,000 worth of its product, with an annual production of nearly $600,000.

According to the same writer, there are two establishments for pipe and sheet lead in St. Louis, yielding products with an annual value of $200,000.

While for mechanical purposes, such extraordinary purity of the lead is not expected, yet, excepting for certain special alloys, (solders, type metal, shot,) the most satisfactory results are, of course, realized by leads of high grade. The question of the corrodibility of lead pipe and the influence on health and comfort by the water it conducts, are intimately connected with the purity of the lead. Dr. Paul Schweitzer†

---

*Appendix A—Broadhead's Report, page 684.

†American Chemist, June, 1876.

of the University of Missouri, found in one gallon of filtered water, which had stood in lead pipe for one month—the water being rain water, collected in zinc-lined tank:

1.079	grains metallic zinc.
0.537	grains metallic iron.
2.503	grains metallic lead.
0.082	grains metallic copper.
0.049	grains metallic arsenic.

The pipe was evidently made from an inferior quality of lead.

The purest soft lead obtained directly from the ores produces the sheet lead best adapted for sulphuric acid chambers, resisting corrosion better, and being, therefore, more durable. The soft lead is generally selected for sheet purposes; hard leads are preferred by manufacturers for making pipes—antimony being sometimes added. What grounds, beyond prejudice, are held for such practice cannot be given.

THE ZINC INDUSTRY

OF

MISSOURI.

ZINC INDUSTRY OF MISSOURI.

The distribution of the ores of zinc in Missouri may be regarded as co-extensive with that of the lead ores, especially in the southwestern and southeastern regions. Three mineralogical species constitute the great bulk of the ores, viz: Sphalerite or blende, calamine and Smithsonite. The ratio of the annual production of these by the Missouri mines, may be estimated to be as follows: Blende : calamine : Smithonite=100 : 138 : 130. The blende is produced chiefly by the mines of Jasper county, and in less amount by those of Newton; the calamine ores by Newton, Dade and the southeastern region, in the order of abundance, while Dade county leads in Smithsonite ores, followed by the southeastern lead mines.

Sphalerite (Glocker), commonly called blende, and by the miners "black jack," is zinc sulphide, and contains by theory, 67 per cent. zinc and 33 per cent. sulphur.

Its crystalline forms belong to the isometric system with a dodecahedral cleavage. Frequently it is compact and massive; sometimes fibrous and radiated; occasionally in botryoidal and other imitative shapes.

The colors are commonly brown, yellow and black, and occasionally green. When pure, yellow or white. Hardness from 3.5 to 4. Specific gravity 3.9 to 4.2. A cubic foot of solid blende, therefore weighs from 243 to 262 pounds.

The southwestern lead mines furnish, commonly, either a coarsely crystalline variety, with broad cleavage faces and of either yellowish or brown color, occasionally dark and nearly black; or a granular variety, frequently of an adamantine lustre, and with a light yellow or brown, and not infrequently a red color.

In the central region sphalerite has not been found in sufficient quantities to have become a commercial article. It is there generally noticed, according to Schmidt,* as dark green, yellow or red crystalline specks, or as agglomerations of crystals between broken chert

---

* Broadhead's report, page 508.

(at the Eureka diggings, Cole county), or surrounds by barite (Coffin Springs, Morgan); as well-developed twin-crystals, translucent and of an amber yellow color, in black clay (Stocker's diggings, Morgan), and with galenite in contact with barite (Collins' diggings, Cooper.)

Three varieties of sphalerite are admitted by Dana. These are: (1)—The *ordinary*, containing little or no iron; colors, white to yellowish brown, sometimes black; (2)—*Marmatite*, the ferriferous variety, with a dark brown to black color, and with the proportion of the iron sulphide to the zinc sulphide, ranging from 1 to 5 up to 1 to 2; (3)—Przibramite, a variety carrying cadmium, always less, however, than five per cent. in amount.

The following are analyses of sphalerite from Missouri localities:

	I.	II.	III.	IV.	V.
Zinc	61.934	61.963	64.67	65.920	64.870
Iron	0.788	0.588	0.53	0.320	0.370
Cadmium	trace	none	....	0.509	0.723
Copper	0.043	none	....	....	....
Antimony	0.002	none	....	....	....
Sulphur	31.269	not est.	not est.	not est.	not est.
Calamine	3.603	....	....	....	....
Gangue	2.552	not est.	2.05	0.25	1.41

No. I—From Newton county, analysed in this laboratory; contained some intermixed calamine which was calculated from the silicic acid, (see Report A, p. 15.) II—From Jasper county. III, IV and V analysed by Chauvenet; first, from Bellew shaft, Granby; second, from Porter's diggings, Joplin; third, from Leadville.

Blende is usually silver bearing. If it occurs in association with galenite, there is commonly more silver in the sphalerite than in the galena. Malaguti and Durocher's,* researches show that most blendes contain small amounts of silver, up to 0.88 per cent., which cannot originate in the associated galenite. The Missouri blendes are not more argentiferous than are the galenas. Blendes found in gold-bearing districts, and even elsewhere, are auriferous to a greater or less extent.

CALAMINE (Smithson) is hydrous zinc silicate, with the following theoretical composition: Silica, 25.0; zinc oxide, 67.5, (containing metallic zinc, 44.2); water, 7.5.

Its common crystalline forms are hemihedral forms of the orthorhombic system. More frequently granular and massive; also fibrous, mammillated, botryoidal and stalactitic. Frequent colors, white and yellowish and brown; lustre, vitreous; hardness, 4.5 to 5. Specific

---

* *Annales des Mines, 1855.*

gravity, 3.43 to 3.49, a cubic foot of the solid mineral, therefore, weighing from 214 to 218 pounds.

The mineral is decomposable by the mineral acids, the silicic acid separating as a gelatinous mass.

The following are the results of analyses of calamine from localities in this State:

	I.	II.	III.	IV.	V.
Silicic acid	25.489	27.51	26.83	23.32	24.264
Zinc oxide	66.813	63.05	66.37	67.15	66.347
Ferric oxide	1.114	1.22	0.65	0.61	1.785
Lime	trace.	1.21	trace.	....	0.078
Water	7.502	7.10	6.46	8.59	7.705
	100.918	100.09	100.31	99.57	100.179
Metallic Zinc	53.62	50.37	52.97	53.95	53.25

I. By C. R. Winters in this laboratory; II, III and IV, by Chauvenet, (Broadhead's Report, page 393.) All the specimens analyzed were of the fine-grained, sub-crystalline variety, mixed with streaks of the crystalline variety, and were all from shafts at Granby. V, from Valle diggings, analyzed under my instruction by L. R. Grabill.

Smithsonite (Beudant,) chemically, is zinc carbonate, and has the following theoretical composition: zinc oxide, 64.8; carbonic acid, 35.2. The zinc carbonate is generally replaced by more or less of the isomorphous compounds, ferrous carbonate, manganous carbonate, calcic carbonate and magnesic carbonate. The pure mineral would contain by theory 51.6 per cent. metallic zinc.

When crystallized its forms belong to the hexagonal system. Its common modes of occurrence are as earthy masses, stalactitic, crystalline incrustations, reniform or botryoidal. Color, white, often with a grayish or greenish tint; more commonly brownish, owing to the presence of ferric oxide resulting from the oxidation of ferrous carbonate.

Hardness, 5; specific gravity, 4 to 4.45. A cubic foot will weigh from 250 to 276 pounds.

Several varieties, depending on differences in amounts of foreign carbonates are recognized, but the separation of the varieties is not very complete or definite. The following are given by Dana: (1) Containing less than four per cent. of any other carbonate and without copper; (2) *ferriferous*, containing more than twenty per cent. of ferrous carbonate (*zinceisenspath;* the *capnite* of Breithaupt); (3) *manganiferous*, containing upwards of five per cent. manganous carbonate; (4) *cupriferous*, (the *Herrerite* of Del Rio) a specimen of which from Abarradon, Mexico, gave Genth, 1.48 of cuprous carbonate, and 93.74 of zinc carbonate, besides manganous, calcic and magnesic carbonates.

The following are the results of analysis of some specimens from Missouri:

	I.	II.	III.	IV.	V.
Zinc carbonate	83.000	93.440	97.60	98.057	93.87
Ferrous carbonate	1.932	4.040	......	none.	1.04
Calcic carbonate	0.803	1.525	......	0.525	2.28
Magnesic carbonate	0.861	0.454	......	0.489	.70
Ferric oxide	3.046	0.800	1.21	none.	2.09
Alumina	2.573	0.261	......	none.	......
Silica	7 390	0.477	1.22	0.902	.34
Water	0.165	nil.	nil.	nil.	......
	99.770	100.277	100 03	99.973	100.32
Metallic zinc	43.019	48.618	50.37	....	....

No. I, from a shaft in Dade county, 12 feet below the surface, analyzed in this laboratory by Mr. Jas. A. Pack; No. II, from same shaft at 35 feet depth, analyzed in the laboratory by A. H. Ohmann-Dumesnil: No. III, from Hardshaft diggings, Granby, by Chauvenet, (Broadhead's Report, 394;) No. IV, white, massive, from Taney county, by the writer; No. V, from Washington county, Valle mines, by the writer.

	VI.	VIII.	VIII.	IX.
Zinc carbonate	90.081	89.538	90.287	91.557
Ferrous carbonate	0.059	0.101	2.594	0.656
Calcic carbonate	2.230	3.343	2.030	2.557
Magnesic carbonate	3.001	2.736	2 268	2.318
Ferric oxide Alumina	3.828	3.527	1.467	1.389
Silica	0.436	0.600	1.492	trace.
Water	.....	......	......	1.086
Zinc sulphide	0.597	0.961	......	......
	100.233	100.806	100.138	99.555
Metallic zinc	47.23	47.20	47.02	47.62

VI, from Hopewell, Washington county; VII, from Mine la Motte; VIII, from St. Joe mines, analyzed in this laboratory by W. R. Brown; IX, from Granby, contains a little hydrozincite intermixed.

Hydrozincite (Kenngott,) or hydrous zinc carbonate, contains: zinc oxide, 75.3, (metallic zinc, 60.4;) carbonic acid, 13.6; water, 11.1. Though noticed at a number of Missouri localities it is, so far as quantity is concerned, of very subordinate importance as an ore of zinc. It is found at the Crabtree diggings (Granby) containing calamine;*

---

*Schmidt *loc. cit.*, page 394.

and Gage[†] notes its occurrence at the Valle mines as an incrustation on Smithsonite.

This mineral is the zinc-bloom of some writers and the Marionite of Elderhorst, who notes[‡] its occurrence in Marion county, Arkansas.

The following two species of zinc-bearing minerals are found in small quantities, but are of no significance from a technical standpoint:

Buratite, a hydrous zinc carbonate containing copper oxide, (zinc oxide, 44.7; copper oxide, 29.2; water, 9.9: carbonic acid 16.2=100,) is found as a coating on zinc ores, at the southwestern mines, notably at those of Oronogo, Jasper county.

Goslarite is found in solution in the mine waters from Oronogo, resulting from the oxidation of sphalerite (see Report A, page 25.)

The following species, which at some localities are important sources of zinc, have not been noticed at Missouri localities:

Zincite (Haidinger): Zinc oxide, contains by theory, 80.26 per cent. metallic zinc.

Franklinite (Berthier): Is composed of ferrous, manganous and zinc oxides combined with ferric and manganic oxides with from 17 to 25 zinc oxide. It occurs abundantly in association with the last named species at Stirling Hill and other localities in northern New Jersey.

Williamite (Levy): Is zinc silicate with 72.9 per cent. zinc oxide (= 58.5 metallic zinc) and 27.1 per cent. silicic acid.

GENERAL METALLURGY OF ZINC.

The fact that metallic zinc volatilizes at a temperature below that which is necessary to reduce zinc oxide, renders all methods for its extraction necessarily those of distillation, for it must be obvious that under such conditions, collections of the metal by liquification or fusion is impossible.

The reducing agent employed in practice is carbon, in the form of coal dust. The products of the reduction are, therefore, mainly carbonic acid and zinc vapor. At a temperature slightly below that at which zinc vapor may be condensed into liquid metal, the carbonic acid may act as an oxidizing agent on the zinc, producing carbonic oxide and zinc oxide again. Hence, to prevent the zinc from reverting to the condition in which it existed in the retort, viz: Zinc oxide, it is necessary to conduct the vapor into a suitably cooled receiver, which must be of small proportions in comparison with the volume

---

[†] Broadhead's Report, 610.

[‡] First Report, Geol. Survey, Arkansas, 1858, pp. 153 and 154.

of the retort or apparatus in which the reduction and volatilization of the zinc takes place. By properly proportioning the size of the condenser or receiver, the amount of reproduced oxide of zinc may be reduced to a minimum, though in practice some is always produced.

Carbonic acid is a necessary product of the reaction which forms the zinc vapor, and as it is in great volume, objectionable in so far as it may, at the proper temperature, reoxidize the zinc, effort should be made in practice to reduce its amount as nearly as possible to that which will be formed by the reduction. If, therefore, the ore contains carbonic acid it should be submitted to a preliminary calcination (in a kiln or a reverberatory furnace.) Water also oxidizes zinc vapor, at certain ranges of temperature, so that a calcination is also advantageous in order that the loss of zinc by reoxidation may be reduced to the minimum possible in practice.

In the case of blende ores it is necessary to submit the ore to a preliminary oxidizing roasting to convert the zinc sulphide into zinc oxide. In such a roasting zinc sulphate is always produced, and as this compound requires a bright red heat for its decomposition (into zinc oxide and sulphuric acid) the expenditure of fuel in such roastings is sensibly increased. While blende ores are richer in zinc than are the oxidized ores (excepting zincite) their treatment is more expensive than is that of the latter, and they must therefore rate at lower price in the market than the calamine or smithsonite ores. Kerl states that at Borbeck and Oberhausen 128 pounds of coal are required to roast sufficient blende to produce one hundred pounds of zinc, while for the calcination of enough calamine ore (mixed silicate and carbonate) to produce the same amount of metal only 13 to 15 pounds of coal are consumed.

The distilling and condensing vessels are made of refractory clay, which must be of high quality. As these vessels are subject to a somewhat rapid wear through scorification and accidental breakage, an abundant and convenient supply of good clay is necessary to all zinc establishments. In fact the quality of the clay and its abundance influence to no slight extent the minor characteristics of the zinc extraction processes.

All the methods employed in practical extraction are based on the general principles already enunciated. The variations are in the forms of the apparatus and the furnaces induced by local considerations, such as the relative values of ore, coal and clay and the greater or less skill of the operatives. Three methods are recognized, deriving their names from the countries in which they have been most successfully conducted. They are,

The English method.
The Silesian method.
The Belgian method.

The first is not so extensively practiced as formerly, having been supplanted by the others. The Belgian method with minor modifications is employed in this country, and is therefore the only method which need be considered in this connection.

In this plan of treatment the distilling vessel or retort is cylindrical and closed at one end. The usual dimensions are as follows:

Interior length	39¼ inches.
Interior diameter	6½ inches.
Thickness of walls	1½ inches.
Thickness of bottom	2½ inches.
Capacity in cubic inches	1294.

A number of these retorts are arranged in rows in a furnace in such a manner that the open end slopes slightly downwards and outwards. The combustion chamber of the furnace is fed with coal for the production of sufficient heat around and in the retorts to produce the temperature necessary for the reducing action of the carbon on the zinc oxide above mentioned, an intimate mixture of the calcined ore and coal (both in the state of powder) having been introduced through the open end of each retort. The lower row or the lower two rows of retorts are usually kept empty during the working of the furnace, as they are in position to receive too much heat. The retorts thus used to break the heat are technically called "canons." The upper rows of retorts, where the temperature is less, is charged usually with mixed coal and the dross and other products of the works rich in zinc. The furnaces are commonly built back to back in blocks of four.

The retorts are charged with a mixture of the calcined ore, two parts; with non-caking coal, one part. About forty pounds of the mixture constitutes a charge for each retort. After the charging, the receivers or condensers are attached to the open or outer end of the retort and properly luted. The receivers are of clay and have the form of the frustum of a cone, with the following dimensions:

Diameter at base	3½ inches.
Diameter at top	2 inches.
Perpendicular height	14¾ inches.
Thickness of walls, from	1¼ to ¾ inches.

The capacity of the receivers, calculated from these dimensions, is about eighty cubic inches, so that the volumes of the retorts and the receivers are to each other as 16 to 1. The object of the relatively small condenser, as before stated, is to reduce the oxidation of the zinc vapor by the carbonic acid, to the smallest possible limit. Some zinc oxide, however, forms, and most of it is collected by attaching to the small or outer end of the receiver, sheet iron cones, tapering to a small opening, which are known as prolongs.

The charge is worked off in twelve hours, when the receivers are detached, the residue withdrawn from the retorts and a fresh charge introduced. The upper retorts, or those charged with the rich zinc dust and dross, and situated in the less highly heated parts of the furnace, are charged but once in twenty-four hours.

The wear of the retorts adds very considerably to the expenses of zinc production by the Belgian methods. It is customary to estimate that the consumption of raw materials is, for *ore*, 35 per cent.; for *coal*, including both fuel and the reducing agent, 50 per cent.; and for *clay*, 15 per cent. These figures are not widely different from those obtained in practice with careful management, and will show that the average life of a retort cannot be much longer than twelve days, or, say through from twenty to twenty-five charges.

The durability of the distilling vessels is largely influenced by the character of the ore put in treatment. Oxide of iron and oxide of lead cause more rapid wear of retorts, from their disposition to form, at high temperatures, fusible compounds with silica. Proper mixing of acidic and basic ores will obviate this difficulty, to some extent, in the case of the iron compounds; lead is prejudicial to a greater extent, and its injurious effects are not so readily controlled.

The character of the ore, in respect to the presence or absence of certain foreign matters, influences also the character of the metallic zinc produced. Among the substances which operate injuriously in this way, may be named: arsenic, antimony and lead.

ZINC PRODUCTION IN MISSOURI.

The inauguration of the manufacture of speltre, or metallic zinc, dates back only to 1867, it having been produced in that year by Mr. Geo. Hesselmeyer at Potosi. Two years subsequently the works at Carondelet were put in operation. Three establishments at Carondelet now consume annually about fifteen thousand tons of Missouri zinc ore, the remainder of the production of the mines finding consumption outside the State, chiefly at La Salle, Illinois.

Carondelet is advantageously located for the site of zinc producing establishments, both as regards coal and clay; besides, the ore is within easy access, either over the Iron Mountain or the Atlantic and Pacific Railroads. The clays from the vicinity of Cheltenham are of good quality as regards refractory properties, and make distilling vessels which have even more than the average durability of similar vessels. The character of the Cheltenham clays are indicated by the following analyses of the samples A. and B.; the latter (not complete) was freed from moisture before analysis:

	A.	B.
Sand and undecomposed silicates	13.972	56.137
Silicic acid	39.157	
Alumina	30.499	32.515
Ferric oxide	1.481	1.020
Magnesia	1.349	not est.
Lime	1.168	1.603
Alkalis	trace.	trace.
Water of combination	9.948	10.570
Moisture	1.806	none.

Analysis A, shows the clay to be composed of 79.6 per cent. of plastic or true-clay material, the per centage composition of which is silica, 49.19; alumina, 38.31; water, 12.50. This composition is not widely different from that calculated for pure kaolinite from the symbol Al_2O_3 2 SiO_2, $2H_2O$, requiring, silica, 46.3; alumina, 39.8; water 13.9.

[The following notes of Dr. Adolph Schmidt (made 1872, but not heretofore used) respecting the coal and clay mines of Parker, Russell & Co., Oakhill, near St. Louis, are inserted in this connection.

Strata below the coal found in a boring on the Russell Mining ground, pit No. 3, in the year 1856:

1.	Blue shale, mixed with clayish nodules of iron ore	3′ 8″
2.	Limestone colored by iron	1′ 2″
3.	Gray, hard slate	2′
4.	Blue slate	1′ 2″
5.	Limestone, colored by iron	1′ 2″
6.	FIRE CLAY	2′
7.	Layer of imperfect and impure coal	2″—6″
8.	FIRE CLAY	4′ 6″
9.	Blue slate	5′10″
10.	Brown slate	1′ 4″
11.	Blue slate	1′10″
12.	Sandstone in layers, shaly, micaceous	
13.	White sandstone	6′ 5½″
14.	Blue limestone	6′ 8″

The section to this point has been found correct and agreeing with the shafts sunk to mine the clay. * * * *

The fire clay is irregularly stratified, wavy formation, and is only found at one single place in the whole property, where it seems to have a mere local development. It is not known between which of the regular strata the clay lies, because this shaft has been dug through old mining materials. Where the clay was found, no coal was found.

The clay looks gray, owing to an admixture of organic matter, but becomes white on burning. It is the exclusive material for a large manufactory of good fire brick, which the company is working on this ground*.

Analysis of this clay, made by Chauvenet & Blair, of St. Louis:

Hygroscopic water	2.96 per cent.
Water of composition, and organic matter	7.27 per cent.
Silica	64.32 per cent.
Alumina	22.82 per cent.
Ferrous oxide	1.75 per cent.
Lime	0.45 per cent.
Magnesia	0.12 per cent.
Soda	0.54 per cent.
Potash	0.23 per cent.
Sulphur	0.12 per cent.
Phosphoric acid	0.00 per cent.
	100.58 per cent.]

A sample, recently analyzed in this laboratory, showed a less amount of silica and more alumina. The result was, as follows, (material dried at 102° C.):

Silicic acid	52.846 per cent.
Alumina	34.214 per cent.
Ferrous oxide	0.665 per cent.

---

* Reference has been made, in the chapter on lead smelting, to the use of the fire bricks made from the clays of this vicinity, for linings for the reverberatories. The composition of two different specimens have been determined by Mr. James A. Pack and Mr. L. R. Grabill, under my direction. The results are as under:

	Laclede. (Pack.)	Mitchell (Grabill.)
Silicic acid	56.307 per cent.	60.394 per cent.
Alumina	35.275 per cent.	35.900 per cent.
Ferrous oxide	0.316 per cent.	1.404 per cent
Ferric oxide	0.088 per cent.	....
Lime	3.316 per cent.	0.741 per cent.
Magnesia	3.560 per cent.	0.732 per cent.
Potassa	0.925 per cent.	0.745 per cent.
Soda		0.247 per cent.
	99.887	100.192

Lime	1.001 per cent.
Magnesia	0.763 per cent.
Potassa	0.084 per cent.
Soda	0.033 per cent.
Water and organic matter	10 838 per cent.
	100 444

This analysis expresses the composition of the clay as a whole, without drawing the distinction between the plastic portion or the true clay and the mechanically intermixed silicates and sand. It will be noticed from this and the preceding analytical results that the clays are free from injurious amounts of those substances (lime, magnesia, ferric oxide and the alkalis) which impair the refractory properties of the clays. The best Stourbridge clay gives silica, 63.30; alumina, 23.30; lime, 0.73; ferrous oxide, 1.80; water and organic matters, 10.30 (Percy); the finest china clay, kaolinite, from Cornwall, according to the same authority, has the following composition: Silica 46.29; alumina, 40.09; lime, 0.50; ferrous oxide, 0.27; combined water, 12.67 (sample freed from hygroscopic water before analysis).

For the purposes of the Carondelet zinc establishments, the St. Louis county clays are tempered by mixing with chamotte of old retorts, freed as far as practicable from the residue of previous distillations. The usual mixture is two of clay to one of chamotte. The distilling vessels prepared from this mixture are, as before stated, of excellent quality, and, as far as ordinary wear and tear are concerned, will last through at least twenty-eight charges.

The ores are calcined for purposes of removing carbonic acid and water, the calcination being generally effected in kilns. It is impossible for us to reach the average composition of the ores used at the several works without a large series of analyses, especially since the ores are from so many different sources and vary greatly, both in mineralogical composition and grade. Below are given the results of three analyses of samples of the calcined ores after crushing. The results were obtained under my direction by Mr. G. A. Duncan (I) and John W. Pack (II and III)—the first and second being samples from the Missouri Company's works, and the third from the Martindale works. The results may serve to give some idea of the character of the material put in treatment, and indicate that I and II were more largely carbonate ore, and that III had a larger amount of calamine.

	I.	II.	III.
Silicic acid	10.280	10.210	29.447
Zinc oxide	75.240	75.360	70.039
Ferric oxide and Alumina	1.909	2.120	0.527

	I.	II.	III.
Manganous oxide	1.634	1.650	....
Lime	4.340	4.430	0.285
Magnesia	3.390	3.250	0.084
Arsenic	trace	trace	0.540
Antimony	0.280	0.289	trace
Carbonic acid	3.090	3.520	trace
Sulphur	....	....	0.349
	100.163	100.829	101.271
Per cent. zinc	60.387	60.483	56.213

All the samples contained faint traces of lead and copper oxides; number III evidently had some blende. The average loss in calcination, estimated by the increase of zinc oxide, must be from 22 to 24 per cent. of the original weight of the raw ore.

The coals used are usually exploited in the Illinois coal fields, which are within convenient access of the furnaces at Carondelet. Forty-five pounds calcined ore, which has been crushed after calcination, are mixed with one-half the volume of coal, and the mixture constitutes a charge for a retort. The yield is about seventy per cent. of the theory, but of the thirty per cent. wastage, a portion is recovered by treatment in the upper retorts of the various furnaces of the zinc dust, drippings and skimmings. The ratio of production of these rich products, as nearly as can be ascertained, is about one of dust to three of skimmings and drippings. They show, according to the analysis of Mr. Pack, the following composition:

Zinc	75.899 per cent.
Antimony	0.372 per cent.
Arsenic	0.321 per cent.
Lead and Copper	traces.
Iron	2.052 per cent.
Carbon	1.221 per cent.
Sulphur	0.026 per cent.
Oxygen in combination with metals	11.398 per cent.
Sand, etc	9.608 per cent.
	100.797

The specimen analyzed was from the Martindale works. The oxygen corresponds to 57.740 per cent. of oxide of zinc.

Another source of loss is in the unvolatilized zinc remaining in the retorts along with the non-volatile matters of the ore and the ashes of the coal. The amount of waste from this source is, of course, exceedingly variable. A specimen of old retort with adhering scoria gave 4.152 per cent. zinc, besides 0.1510, lead; copper, a trace; arsenic, 0.4348; antimony, 0.9488; sulphur, 0.8560; and iron, 2.2774 per cent. The latter, probably, in part as oxide and belonging to the clay and ashes.

The loss from this and all other sources, is estimated to be about twelve per cent. of the zinc produced.

The Martindale Zinc Works has two blocks of four furnaces each; in all eight furnaces with 108 retorts each, or 864 retorts.

Carondelet works has 88 retorts to a furnace, and six furnaces. The number of retorts is therefore 528.

The Missouri Company's furnaces have 74 retorts each, giving for the four furnaces 296 retorts.

The total production of these furnaces for the year 1875, was 4,650 tons of spelter.

The zinc produced by these works was analyzed in 1874, by Mr. John Pack, under my personal supervision and instruction. His results are appended. No. I being from the Missouri Company's works, and No. II from the Martindale:

	I.	II.
Sulphur	.0035	.0741
Silica	.1346	.1374
Carbon	.1775	.0006
Iron	.7173	.2863
Lead	.1001	.0061
Copper	.1123	.0018
Arsenic	.0603	.0590
Antimony	.0249	none
Zinc, by difference	98.6995	99.4347
	100.0000	100.0000

MANUFACTURE OF ZINC WHITE IN MISSOURI.

The use of Zinc White, or oxide, as a substitute for white lead, seems to have been suggested in 1845, by Leclerc in France, and for the discovery, the Montaigne medal was awarded him by the French Academy. The earlier methods employed for its production necessitated the reduction of the ore to spelter, and the subsequent burning of this zinc metal in a separate vessel, by which it was converted into oxide. In the years 1847 and 1848, English patents, numbers 12,001 and 11,846 were issued to Rochaz* and to Newton† for improved methods in the manufacture of pigments in the direction of "treating

---

*Newton's *London Journal*, *Conjoined Series*, vol. 43, pages 246 *et seq.*

†*Ibid.* pp. 256 *et seq.*

zinc ores and manufacturing of oxide of zinc." The novel features in these improvements consisted in the reduction of the ore and the oxidation of the zinc in an apparatus furnished with a coercitive blast.

The manufacture in this country began at the New Jersey Zinc Works in 1850, and was done by what is known as the direct method, that is, from the ore without the separate or distinct production of the metal, and was the first successful issue, on a practical scale, of the attempts at this manufacture. The practice then in use, consisted in firing the ore interstratified with fuel in heaps; suddenly cooling with water for facilitating the subsequent crushing in a stamp-mill; mixing the crushed ore with about forty per cent. of its weight of coal; charging into a muffle furnace, and collecting the resulting oxide in a suitable apparatus.

This method of manufacture was continued with varying success till sometime in the year 1855, when the process, now used universally in this country, and generally known by the name of its inventor and patentee, Wetherill, was substituted for it. The Washington County Zinc Company, near Hopewell, manufactured oxide under the extension of this patent, using ores produced in this State, and though not in operation at the present writing (February, 1876,) the cause of the suspension of the operations is certainly not to be sought for in technical reasons. The works and the mode of production are therefore entitled to consideration in a report of this character, the more especially since the ores of zinc are so abundant in the State, and the product has intrinsic value as a legitimate substitute for white lead in certain applications.

The furnace used is a low-arched chamber, the sole or bottom of which is formed of wide grate bars, placed contiguously, and provided with numerous conical perforations. Below these bars is an air chamber which is supplied with a low pressure blast, from a centrifugal blower. The arch of the upper chamber has two openings, one connecting with the stack and the other connecting by means of an iron pipe with the apparatus for condensation of the oxide. Both of these openings are so arranged as to be closed at will, and as the stages of the operations require. Through the shell and on each side of the furnace, and at the level of the grate bars are several openings or work doors, which give access to the upper chamber for the purpose of charging and discharging and of manipulating the charge. These openings are closed, when necessary, by banking up ashes or loam around them.

The furnace is operated by first spreading a layer of bedding coal over the grate bars and igniting the same, the combustion being aided by the blast from the air chamber beneath. Upon this is spread a layer of crushed ore intimately mixed with fine coal or dust for reducing purposes. As soon as this layer becomes sufficiently heated to effect the reduction of the oxide of zinc—indicated by the peculiar blueish-green flame of burning zinc appearing—the flue connecting with the stack which has been opened is closed, and that leading to the condensing apparatus is opened, at the same time the work doors are closed up by banking the ashes around the openings. The process is aided by frequent rabblings or stirrings, and is terminated when the zinc flame is no longer to be seen—usually after the lapse of from four to six hours. The scoriaceous mass or residuum is now withdrawn, a fresh layer of bedding coal is introduced and ignited either by the sufficient heat of the grate bars, or by some of the residue which is, for that purpose, allowed to remain on the sole of the furnace, and the operations repeated as above.

The furnaces are usually built in blocks ; a flue of brick connects them with a sheet iron pipe four feet in diameter, which opens into a brick chamber or tower, where the oxide meets with water to aid in condensation and thence passes into bags of strong cotton fabric, where the oxide is collected, and the gaseous products of the furnace filter through the pores of the bags into the exterior atmosphere. The draft of the furnace is aided by an exhaust fan placed intermediate to the furnace and the bags.

The method of collection in bags was early introduced at the New Jersey works, and has been found to be the only effective means. It is capable of extension for the collection of other valuable metallurgical products, such as fume, and has hardly received the attention that its simplicity and thoroughness warrant.*

The *rationale* of the operation is not difficult to understand. The reduction of the oxide of the zinc ore is effected by carbonic oxide, the products of the action being carbonic acid and metallic zinc, the latter in the state of vapor. At a temperature somewhat less than that necessary for the reduction, which condition would be found in the upper part of the charge, the vapor of zinc will burn in an atmosphere of carbonic acid, producing carbonic oxide and zinc oxide. The reduction of the ore and the combustion of the zinc vapor therefrom go on almost *pari passu*, observations showing that almost all the oxidation is effected previously to the zinc leaving the charge and not in the space of the chamber above the charge. When this latter does take place there is produced and collected in the flues, a larger amount of so-called *blue-powder*, a mixture of finely divided metallic

---

*On this point see the author on Loss by Volatilization, *Jour. Franklin Institute, Feb., 1871.*

zinc and oxide. The formation of this blue powder was a serious objection to the successful practical working of the muffle-furnace; when, however, the furnace now in use is charged properly the production of this substance is reduced to a minimum.

Blende-bearing ores should be roasted before treatment. Generally, such roasting is done in a reverberatory with a double sole. If galenite is present in the ores it is largely carried over by the specifically light oxide of zinc, and appears in the product chiefly as lead oxide or lead sulphate. At the Keystone Zinc Works, near Tyrone, Pennsylvania, galeniferous zinc ores, containing from six to twelve per cent. lead, were treated by this process, and the so-called Bartlett white lead was made in a similar manner by burning the mixture of galena and blende from the buddles of the Silver Hill mine, Davidson county, South Carolina. The annexed analyses of these products by the author* are introduced for illustrating the volatility of lead under the circumstances discussed:

	Keystone Oxide.	"Bartlett White Lead."
Zinc oxide	73.246	72.083
Lead oxide	....	0.274
Antimony oxide	....	trace.
Ferric oxide	....	trace.
Lead sulphate	25.084	23.968
Zinc sulphate	0.574	0.810
Zinc chloride	....	0.839
Ferric chloride	....	0.071
Antimony chloride	....	trace.
Cadmium chloride	....	0.256
Cadium sulphate	0.186	....
Calcic carbonate	0.729	....

A process for converting lead ores into a paint material susceptible of taking the place of white lead, by a method similar to that described above has recently been patented.† The product is unexceptionable in color, and mixes readily with oil. Some recent analyses by the writer show the following to be the compositions of two samples prepared by this method of burning, with slight differences, however, in the modes of collection, "A" being condensed without, and "B" by use of water:

	A.	B.
Zinc sulphate	0.4112	0.3488
Ferric sulphate	trace.	....

---

* *Watt's Dictionary of Chemistry*, II supplement, page 725, from *Chemical News*, XXIII, 236.

† Now in use at Lone Elm, Joplin, in connection with the hearths, for the condensation of fume.

	A.	B.
Lime sulphate	0.0151	0.0045
Lead sulphate	38.9874	39.5008
Lead carbonate	2.9822	17.9145
Lead oxide	53.3813	37.3707
Zinc oxide	4.2938	4.6441
Ferric oxide	0.0551	0.1104
Sand	0.0091	0.0221
Organic matter, chiefly fibre from bag	0.0045	0.0024
	100.1397	99.9183

The quantity of residue left on the grate bars after proper burning of the zinc ore by the Wetherill process is about equal in weight to the weight of the ore introduced. It is a scoriaceous mixture of the foreign matters of the ore, ashes of coal, unconsumed coal, and some oxide of zinc. Its quality in respect to the amount of this latter has been found to range between 2.5 per cent. and 22 per cent, being always smaller when the charges are of small depth, and greater as the charges are increased in thickness. The thickness of the charge spread over the bedding coal is usually from four to six inches, at both the Lehigh, (Pennsylvania,) and the New Jersey, (Newark,) works, and the amount of oxide in the residue under such conditions of charging and working never exceeds four per cent. Any notable increase in thickness is accompanied by increased wastage of oxide in the residue, irrespective of the time that may be given, after certain limits, to the elaboration of the charge, so that economy in both time and zinc is secured by thin charges. Hence the necessity of crushed ore and dust coal for the charge, and of small coal (usually chestnut size) for the bedding fuel.

The greater energy of blast which would be required for thicker charges might also cause a greater calorific intensity resulting, in interference with the proper reactions, and possibly causing, also, a slagging or semi-fusion of the charge, interfering with the disengagements of zinc vapor.

The effect of adding to the depth of charge in the Wetherill furnace, is illustrated in the annexed table * of results of analyses of

---

* Condensed from testimony and argument in *Wetherill vs. the N. J. Zinc Co.*, Philadelphia, 1871.

the residues of certain experimental charges in the New Jersey furnaces:

Thickness of Charge.	Time of Working.	Per Cent. of Oxide in Residue.	Analyst.
6 to 8 inches.........	5 hours..............	2.03 per cent	T. C. Garrett.
6 to 8 inches.........	6 hours..............	1.20 per cent	Author.
6 to 8 inches.........	6 hours..............	3·203 per cent	Author.
7½ inches........	14 hours	9.75 per cent .... 10.54 per cent........... 8.06 per cent	Charles Roepper. Author. T. C. Garrett.
No measure	5½ hours	6.49 per cent	Charles Roepper.
No measure	11 hours	6.57 per cent	Charles Roepper.
14 inches........	8 hours..	7.95 per cent	Charles Roepper.
14 to 17 inches.....	13½ hours...........	8.05 per cent	Charles Roepper.
16 to 19 inches......	28 hours	13.05 per cent	Charles Roepper.
16 to 19 inches......	28 hours	10.45 per cent	Author.
16 to 19 inches......	28 hours	10.23 per cent	Author.

The New Jersey Company works ore of high grade containing a large proportion of willemite, zincite and the zinciferous iron ore, Franklinite. The last named species is rich also in manganese, so that the residue is notably both iron and manganese bearing, and is used for mixing with iron ores for treatment in the blast furnace for the manufacture of an excellent quality of *spiegeleisen*. Experience has shown that where the residues contain more than four per cent. oxide of zinc they are not adapted to blast furnace treatment, and their introduction results in serious and expensive disadvantage to the charge in the furnace, so that the profitable utilization of this valuable matter depends on the close working of the oxide furnaces. The Lehigh Company works under the double disadvantage of starting with lower grade ores, and of producing a residue which cannot be utilized, and is therefore forced to clean up the charge as closely as is practicable. The Missouri Company has not run its residue down as cleanly as that of either of the above named establishments, probably on the grounds of starting with rich ores and of economy in fuel, but whether the practice has, in reality, been subservient to increase of profit is a point still open to demonstration.

The Hopewell furnaces are eight in number, built in one block, each with a hearth of 4.5 feet by 6 feet, and with a height of 27 inches from the grate bars to the centre of the arch. The bedding coal for a charge is about 250 pounds weight of Pennsylvania anthracite spread evenly over the grate surface and giving therefore a thickness of one and a half inches. The charge proper consists of 400 pounds ore and 200 pounds of small coal and dust thoroughly mixed before introduction. The coal is mixed Pennsylvania anthracite and that from Spadra, Arkansas. The ore comes chiefly from the Washington county lead mines, and contained occasionally a small amount of galenite. Sometimes Dade county smithsonite was employed. A representative lot of specimens gave the following result on analysis:

Zinc oxide	53.997 per cent.
Ferric oxide	4.263 per cent.
Alumina	0.632 per cent.
Ferrous oxide	1.039 per cent.
Lime	2 393 per cent.
Magnesia	1.546 per cent.
Lead oxide	trace.
Cupric oxide	0.052 per cent.
Cadmium oxide	0.173 per cent.
Antimony oxide	0.170 per cent.
Carbonic acid	24.302 per cent.
Silica	8.510 per cent.
Water	2.414 per cent.
Sulphur	0.242 per cent.
	99.633

It is evident from this result that the ore is a mixture of about two-thirds smithsonite and one-third calamine, containing, however, a fractional per centage of blende and foreign sulphides.

The charge indicated required nearly six hours for its elaboration. The ore yielded a little more than thirty per cent. of oxide, leaving therefore as stated above, a large part of the zinc in the residue. This is confirmed by the annexed result of analysis of the residue:

Zinc oxide	25.426 per cent.
Ferrous oxide	12.528 per cent.
Alumina	4.247 per cent.
Lime	5.311 per cent.
Magnesia	1.692 per cent.
Ferrous sulphide	8.723 per cent.
Cuprous sulphide	0.175 per cent.
Silica	31.740 per cent.
Carbonaceous matters	10.040 per cent.
	99.882 per cent.

The quality of the oxide varies widely according to the foreign matters in the ore, being influenced to the greatest degree by the lead and the sulphur. Several brands of the oxide have been produced the character of each being indicated by the analytical results given below:

	I.	II.	III.	IV.
Zinc sulphate	1.153	0.203	1.001	0.824
Calcic sulphate	0.008	0.041	0.080	0.011
Lead sulphate	4.970	3.243	1.271	0.249
Lead oxide	1.047	4.598	0.399	0.814
Copper oxide	0.063	0.186	0.043	0.021
Ferric oxide	trace	trace	trace	trace
Zinc oxide, by difference	92.749	91.729	97.206	98.081

I is "Star No. 1;" II, "Star No. 2;" III, "Star No. 3;" IV "Oxide."

Two samples produced entirely from Dade county ores, were analyzed under my directions by Mr. P. E. Blow, and the results were published in the Report of the Curators of the State University for the year ending June, 1875. These results are reproduced below:

	V.	VI.
Zinc sulphate	0.3001	0.4998 per cent.
Calcic sulphate	0.1114	0.1400 per cent.
Cadmium sulphate	....	0 2024 per cent.
Zinc chloride	0.0948	traces
Cadmium chloride	0.9446	traces
Lead sulphate	0.0195	0.0042 per cent.
Lead oxide	0.4995	1.1488 per cent.
Ferric oxide	0.0322	0.0976 per cent.
Zinc oxide, by difference	97.9979	97.9062 per cent.
	100.0000	100.0000 per cent.

V was of good color and body; VI was somewhat "off color," having a decided buff tint and marked impure. The ore from which they were produced contained 0.842 per cent. cadmium oxide.

While the works were in operation, they consumed annually three thousand tons of ore and twenty-seven hundred tons of coal for all purposes. The product was from eight hundred to a thousand tons of oxide of the various grades.

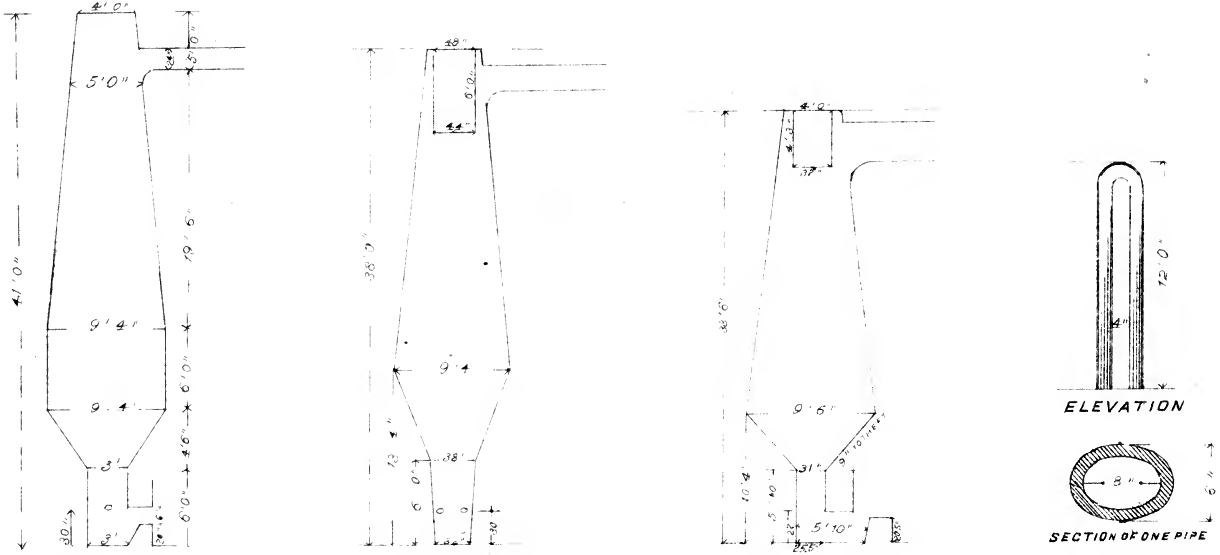
The chief use of zinc oxide or white is as a basis for a pigment. It is rather more expensive than white lead, but is claimed by some to have even better covering properties in the ratio of 13 to 10. Further, it is not so apt to change color, retaining its pure white tint much longer than the best white lead, and is not blackened when exposed to emanations containing sulphurretted hydrogen. A further and more limited use of the oxide is in the conversion of caoutchouc into the material known as ebonite.

The shipments of oxide from Hopewell, by the Washington County Zinc Company, amounted in 1875, to 2,446 barrels, weighing 489,000 pounds, of the several grades. The value of the shipments is given at $44,010.

THE IRON INDUSTRY,

WITH ESPECIAL REFERENCE TO THE MANUFACTURE OF

CHARCOAL IRON.



SCOTIA. **MOSELLE.** **MERAMEC.** **SCOTIA**

PLATE I.

CHARCOAL FURNACE'S on A & P. R.R.

Beoan & Carter Lith. Jeff City, Mo

IRON INDUSTRY OF MISSOURI.

MARAMEC FURNACE.

The Maramec furnace (T. 37, R. 6 W., Secs. 1 and 2), owned by the heirs of Thomas James, and operated under the management of William James, Esq., of St. James, is the oldest establishment now in operation in the State. Smelting work was begun at the location as early as 1824.

The present furnace is of thimble top style, its dimensions being given in the accompanying section (Plate I.) It has two tuyeres of three and three-quarter inches diameter, delivering blast of three pounds pressure. The blowing apparatus is worked by water-power, furnished by the ample spring at the works. The diameter of the blower is 4 ft.; stroke 4 ft. 6 in.; two cylinders; number of revolutions per minute, 16. The calculated capacity of the apparatus is 3,612 cubic feet of air per minute; or, with an average daily production of fourteen tons iron, one ton of iron requires 13.86 tons of air.

The ore is furnished by the Maramec bank, near the furnace. [For description of this deposit by Schmidt, see report of 1872, pp. 144, *et seq.* I]. A carefully selected sample gave the following result:

Ferric oxide	73.365 per cent.
Ferrous oxide	2.237 per cent.
Alumina	7.534 per cent.
Magnesia	2.044 per cent.
Lime	trace.
Silica	14.740 per cent.
Phosphoric acids	trace.
Sulphur	0.057
	99.997
Metallic Iron	53.094 per cent.

The yield shown by the above analysis is about an average of that produced by working. During the three months of December, (1874), and January and March, (of 1875), there were required 4,472, 4,345 and 3,926 pounds of ore, respectively, to produce one ton (2,240 pounds) of iron, corresponding to 50.6, 51.6 and 57 per cent. yield for the ore, or a mean of 53 per cent.

Flux: The limestone used for flux is nearly a true dolomite, as is shown by the annexed result of analysis:

Calcic carbonate	52.611	per cent.
Magnesic carbonate	42.415	per cent.
Ferric oxide and alumina	1.330	per cent.
Insoluble matters	4.201	per cent.
	100.557	

A special examination showed it to contain 0.036 per cent. of sulphur and 0.005 of phosphoric acid (=0022 of phosphorus.)

The usual charge consists of 520 pounds ore and 100 pounds flux, one hundred such charges being introduced daily. In January, 1875, 1,578,280 pounds ore were run through in 2,772 charges, and in March of the same year, 2,006,410 pounds of ore were used in 3,562 charges, or at the rate of 569 pounds of charge for the first and 563 pounds for the second.

Fuel consumption: The records of the furnace show a consumption of 153 bushels of charcoal per ton of pig iron produced. The charcoal contains, 8.66 per cent. ash; 497 hygroscopic water, and 86.37 carbon. Estimating the weight per bushel to be at the rate of eighteen pounds, 1.221 tons charcoal are required per ton of pig iron, or 1.053 ton of pure carbon to the ton of iron.

Products: The average daily production of iron is about fifteen tons. Taking the month of April, 1875, to represent the character of the product, it will be found that the relative amounts of the three grades of iron are as (No. 3), 10 to 10.5, (No. 1) to 44 (No. 2). The chemical composition of these several grades are found to be as below: *

	I.	II.	III.
Carbon, graphitic	2.024	2 000	2.662
Carbon, combined	2.276	0.887	0.288
Silicon	1.378	1.246	1.337
Phosphorus	trace	trace	trace
Sulphur	0.048	0.136	0.091
Manganese	0.904	0.717	0.852
Iron, by difference	94.370	95.010	94.770
	100.000	100.000	100.000

---

* For methods of Analyses followed, see Fresenius *Quantitative Analysis*, Fourth *English* Edition, pp. 658, *et seq.*

From the data given above in connection with the analyses of the raw materials used, the amount of *slag* produced to the ton of iron will be found to be 0.858 ton. Of this quantity the ore contributes, 0.569 ton; the flux, 0.183 ton, and the charcoal 0.106 ton. The composition of the slag is indicated in the annexed analysis in which the sulphur is regarded as combined with a part of the calcium into calcium sulphide.

Composition Maramec Slag.

Silica	64.746 per cent.
Alumina	7.729 per cent.
Ferrous oxide	2.282 per cent.
Lime	16.973 per cent.
Magnesia	5.349 per cent.
Soda	1.812 per cent.
Potassa	0,702 per cent.
Phosphoric acid	0.572 per cent.
Calcium sulphide	0.134 per cent.
	100.299
Sulphur in slag	0.059 per cent.
Phosphorus in slag	0.249 per cent.

The percentage of oxygen of the silica is 34.53, and that of the bases, 11.68, giving a ratio of 1 to 3 (nearly) and constituting, therefore, an acid silicate.

Combining the results of the operation of the furnace with further details furnished by the chemical investigations, the distribution of the raw materials—solid and gaseous—entering into the production of one ton of average pig iron at the Maramec furnace, may be ascertained with sufficient accuracy to form the basis for calculations concerning the calorific effect of the fuel. The raw materials are ore, flux, charcoal and air, and the products are pig iron, slag and the gases escaping from the tunnel-head. Tabulated, the relations of these factors and products will stand as follows, the calculations being made to the ton of iron produced:

			Iron.	Slag.	Gases.
ORE, 1.966 tons, containing:					
Ferric oxide	1.371	Iron	0.960	......	......
		Oxygen	......	.....	0.415
Ferrous oxide	0 054	Ferrous oxide	......	0.048	......
		Oxygen	......	......	0.006
Silica	0.500	Silicon	0.010	......	......
		Oxygen	......	......	0.010
		Silica	......	0.480	......
Lime and magnesia			......	0.041	......

FLUX, 0.339 tons containing:			
Silica	......	0.014	......
Lime	......	0.100	......
Magnesia	......	0.069	......
Carbonic acid	......	.....	0.154
CHARCOAL, 1.221 tons containing:			
Carbon	0.030	......	1.024
Water	......	......	0.061
Ashes	......	0.106	......
AIR, 13.867 tons containing:			
Oxygen	......	......	3.200
Nitrogen	......	......	10.590
Moisture *	......	......	0.077

If in this blast furnace the total carbon of the charcoal was burned to its maximum of oxidation, that is, to carbonic acid, the amount of heat developed would be 8,484 *ton units of heat*, or in other words, its calorific power would be enough to raise the temperature of 8,484 tons of water from 0° to 1° centigrade. But such calorific power cannot be realized in blast furnace practice, for the gases escaping from the tunnel head of any furnace are mixtures of carbonic acid and carbonic oxide, and indicate, therefore, a more or less perfect combustion, so far as the heat production is concerned, according to the preponderance of the one or the other. Hence, the ratio of these two escaping gases when determined, gives an index to the working of the furnace, considered in respect to its fuel consumption. This ratio of CO (carbonic oxide) to CO_2 (carbonic acid), Gruner† fixes as 1 to 1.217 in an ideally perfect working coke furnace; it is always less in charcoal furnaces, for reasons noted hereafter, and in the great furnaces of the Cleveland ore district, when in good condition, it is generally between 0.5 and 0.7, and for bad working 0.35 to 0.40.

If the final product were carbonic oxide alone, the number of heat units produced by an unit of carbon would be 2,473, so that the calorific power produced in the furnace must stand somewhere between this number and 8,080—the number produced by the complete combustion of carbon into carbonic acid—and will vary accordingly as there is more or less of the one or of the other of these two compounds of carbon and oxygen produced. The reason that neither the maximum nor the minimum amount of heat is not obtained will be obvious

---

* Moisture, estimated at 75 per cent. of saturation, and saturated air contains 0.75 of water.

† *Blast Furnace Studies*, p. 15. Translation.

from an examination of the reactions necessary in the furnace to produce the iron.

"In every furnace of this type," says Gruner,* there are two contrary currents in motion, and reacting the one upon the other—a *gaseous current ascending*, the temperature of which is at first very high, and decreases very gradually till it quits the furnace at the tunnel head or top; and a *solid descending current* composed of the ores, the fluxes and the fuel, the temperature of which goes on increasing always under the action of the gaseous current in the opposite direction. Of these two currents, the one is slow, the other very rapid. The solid materials of the charges rarely descend with a greater speed than 20 inches per hour, whilst the gases pass upwards with a velocity of 20 inches per second. * * * The air blown into the furnace at the tuyeres is almost instantly transformed into carbonic oxide,† and this gas, in its passage up the body of the furnace, acts more or less directly in reducing ores—that is to say with or without the aid of solid carbon. The reduction of the oxide of iron may take place in three ways:

1. $3\,CO + Fe_2\,O_3 = 3\,CO_2 + 2\,Fe.$
2. $3\,C + 2\,Fe_2\,O_3 = 3\,CO_2 + 4\,Fe.$
3. $3\,C + Fe_2\,O_3 = 3\,CO + 2\,Fe.$

It may at once be remarked that the first two ways of reduction are not in fact realizable, if we adopt the proportions given in the formulas. In these conditions the metallic iron would be partially reoxidized by the carbonic acid. We know by the experiments of M. Debray, confirmed by Mr. Lowthian Bell, that in presence of equal volumes of carbonic oxide (CO) and carbonic acid (CO_2) peroxide of iron ($Fe_2\,O_3$) and metallic iron (Fe) are both brought to the state of protoxide ($Fe\,O$). But these two first modes of reduction are impossible taken singly; they generally help, with the third mode, in producing the final result, and in fact, the gases taken at the furnace top are always composed of a mixture of CO and CO_2. According as the one or the other mode of reduction has the greater share in the final

---

* *Ib.* pp. *et seq.*

† I may add here that the excellent authority of Tunner is against the assumption that carbonic oxide alone is present in the lower part of the furnace, or that, in fact there is any part of the furnace which does not contain both carbonic acid and carbonic oxide in varying proportions, dependent on various conditions. Consult in this connection Prime's translation of Akerman's *Researches on the Consumption of Heat in the Blast-furnace Process. Transactions Am. Inst. Mining Engineers*, Vol. I, pp. 426, *et. seq.*

result, the proportion of CO or CO_2 in the gases taken at the furnace top is the greater. But it is very easy to show that these three modes of reduction require very different quantities of caloric; and in this point of view—that is, in reference to the consumption of fuel—it is not a matter of indifference which of these reactions takes place in blast furnaces."

The calorific effect of each of these reactions is made up of the amount of heat absorbed and the amount given off. The first is constant for all those reactions, being exactly the amount of heat which an unit of oxygen would produce in burning iron to Fe_2O_3, but working in the opposite direction that is being *absorbed* by reduction instead of *liberated* by oxidation. It has never been determined rigorously, but is generally estimated to be about 4,500 heat units. The second, that is the amount of heat given off, is variable, being for the first reaction 4,205 units, for the second 3,030 units, and for the third 1,855 units for each unit of oxygen removed by the carbonic oxide or the carbon. Comparing these variables with the constant, it will be seen that in the first reaction the heat liberated by the combustion of the carbonic oxide into carbonic acid by the oxygen of the Fe_2O_3 is almost equal to that absorbed by the reduction of the Fe_2O_3 to 2 Fe, while in the other reactions in which the reduction is effected by solid carbon, (in the one case oxidized by the change to CO_2, and in the other, to CO), the heat absorbed is greater by about 1,500 and 2,600 units, respectively. In other words, to effect a reduction by solid carbon, the amount of the latter must be increased over the amount required in the form of carbonic oxide, and the amount of increase is relatively greater as carbonic oxide or carbonic acid is the final product. Representing the amount of carbon required to effect the reduction by carbonic oxide by unity, the solid carbon burned to carbonic acid (reaction 2) becomes 1.55, and to be burned to carbonic oxide, (reaction 3), it must be increased to 2.33.

The ideally perfect furnace considers the reduction by carbonic oxide only, and regards the final products as carbonic oxide and carbonic acid in the proportion of one weight of the first to 1.217 of the latter. For one unit of pig iron produced there will be consumed 0.880 of carbon, producing 1.408 of carbonic acid and 1.157 of carbonic oxide. If the relation between the two forms of oxidation of the carbon in the gases escaping from the tunnel head be the index of the effectiveness of the fuel in the furnace, analysis of these gases will give the required knowledge with all necessary exactness. But the performance of this analytical work is certainly not easy, nor is its execution at iron establishments generally provided for. In by far the larger number of cases the determination of the ratio of the furnaces, gases can only be arrived at by calculations, which being based on so many assumed positions, necessarily lead only to approximations, interesting in a theoretical view, but without the element of certainty to give them any great practical significance. Nevertheless, these calculations are at present the only guide that we have to knowledge of how far the furnaces in this State are proportioned and governed in order to secure economy in the consumption of fuel.

The distribution of materials given on a preceding page, in connection with certain data, established by Bell, Akermann, Gruner and others, enable us to reach the following approximation to the amount of ton-units of heat necessary and required to produce one ton of iron in the Maramec furnace:

Reduction of iron from oxide	[0.960 ton	×1780 units a]	1,708	ton H-U
Carbon impregnation	[.030	×2467 b]	74	"
Expulsion of carbonic acid from limestone	[0.339	× 370 c]	125	"
Decomposition of carbonic acid from limestone	[0.154	×3200 d]	493	"
Decomposition of water from blast and charcoal	[0.015	×29638 e]	444	"
Reduction of silicon from silica	[0.010	×7830 f]	78	"
Fusion of pig iron	[1.000	× 330 g]	330	"
Fusion of slag	[0.858	× 550 h]	472	"
Heat transmitted through furnace walls k			113	"
Heat carried off in the tuyere water l			14	"
Heat carried off by gases from tunnel head m			553	"
Total units of heat per ton of iron			4,404	"

To produce this amount of heat there were consumed 1.052 tons of carbon, burning in such a manner 31 per cent. should form carbonic acid and 69 per cent. should be converted into carbonic oxide, and the resulting gases will be $\frac{CO_2}{CO}=0.712$; while the calorific power, represented by the heat consumption, will be 76 per cent. of that possible for the same amount of carbon if burned as in the ideal furnace. It may be well to mention in this connection, that the activity of furnaces varies greatly with the character of the fuel and the ore. Reference will be made to some of the peculiarities of charcoal furnaces in comparison with those using coke in treating of some of the hot blast charcoal furnaces of this State.

a—Bell's number; b—Bell; c—Bell; d—Bell; e—Akermann; 34,462 is the usual number given for calorific power of hydrogen, but this factor includes the deduction for heat absorbed in converting 9 units of water (formed from one unite of hydrogen) into steam [34,462—(9x536)=29,638; f—Akermann; g—Bell; h—Bell; k—Akermann; l—Akermann; m—Akermann.

Bloomery at the Maramec Works: The bloomery has eight fires, each running eight charges per day, of from 12 to 14 hours; one three-ton trip-hammer serves all the fires. The tuyere enters the fire about 1½ inches; its diameter is 1¼ inches; the bottom plates are not cooled, but rest on the ground, and last from one to three months.

A charge is 280 pounds of pig iron, 2-5 being white or mottled iron and 3-5 gray iron. A charge gives a loop of 246 pounds, which is hammered into two blooms (120 lbs. each) at each heat, without any reheating. Eighty bushels of charcoal are required per ton (2,464 pounds) of bloom produced. Each fire requires two hands, and makes only one turn per day.

The power for the blowing engine is furnished by the spring. There are two vertical cylinders, 6 feet 2 inches in diameter; length of stroke, 4½ feet; number of revolutions, 14 per minute.

The total production averages one hundred tons of blooms per month, in winter time, and somewhat less in summer.

OZARK FURNACE.

[Sec. 21, T. 37, R. 9; Phelps County.]

This furnace was built in 1874, but owing to the depression of the iron industry throughout the country, was blown out in the spring of 1875.* (For form and dimensions see plate 2.) While running it used a mixture of equal weights of ores from the Beaver. † (S. ½ Sec. 33, T. 37, R. 8 W., Phelps county), the Hancock, (Sec. 14, T. 38, R. 12 W., Miller county), and the St. James banks. The composition of these ores is indicated by the annexed results of analyses by Mr. Hare:

	Beaver Ore.	St. James Ore.	Hancock Ore.
Ferric oxide	91.612	73.604	79.437
Alumina	3.800	2.608	5.229
Manganous oxide	....	....	0.081
Lime	0.384	12.402	0.174
Magnesia	0.221	4.803	0.290
Silica	3.853	6.491	11.941
Phosphoric acid	0.156	0.230	0.074
Sulphur	0.151	0.214	0.073
Water	....	....	not est.
	100.352	100.177	97.298
Metallic iron	64 128	51.523	55.606
Phosphorus	0.0671	0.1004	0.0323

---

* The Ozark furnace resumed work during May, 1877.

† For description (Schmidt) see report, 1872, 148.

According to estimates made at the time the furnace was in blast, 1,502 tons of ore made one ton of iron. To this were added 0.175 tons of magnesian limestone for flux, (= 12 per cent. of the weight of the ore) which contained the following constituents, centessimally expressed:

Calcic carbonate	55.541
Magnesic carbonate	41.168
Ferric oxide and alumina	0.734
Insoluble matters	3.795
Ferrous di-sulphide	0.091
	99.329

Special examination for phosphoric acid, showed 0.004 per cent. (= 0.0017 phosphorus.) The sulphur is 0.0485 per cent.

The charcoal was used in the proportion of 0.762 tons per ton of iron made. The blast was heated to the temperature of melting lead (325°C.) and entered the furnace by two tuyeres.

The iron contained:

Carbon	2.6588 per cent.
Silicon	0.4199 per cent.
Manganese	0.5713 per cent.
Phosphorus	0.0066 per cent.
Sulphur	0.0440 per cent.
Iron, by difference	96.2994 per cent.
	100.0000 per cent.

The distribution of the materials show that the slag-forming substances amounted to 0.528 ton per ton of Iron. The mean of two analyses gives the following composition to the slag:

Ozark Slag.

Silica	48.928
Alumina	16.617
Ferrous oxide	2.573
Lime	27.895
Magnesia	0.855
Potassa and soda	2.202
Calcium sulphide	0.456
Phosphoric acid	0.078
	99.625
Sulphur	0.2005
Phosphorus	0.0340

The oxygen ratio of silica to the bases is about 1½ to 1 (more exactly 26 to 17), showing the slag to consist of a bi-silicate in mixture with a singulo-silicate.

The difficulties attendant upon bringing the opening work of a furnace into a fair or satisfactory condition had hardly been overcome in the instance of the Ozark furnace before it was put out of blast. No statement of its working effect could therefore be just, and comparisons based upon the fuel consumption and produce of iron would result only in erroneous conclusions. The furnace is owned by the Ozark Iron Company, and was built and operated under the general management of William James, Esq., of St. James. The character of the work upon the furnace and upon the other improvements of the estate is most substantial.

SCOTIA IRON WORKS.

(N. E. ¼, SEC. 1, T. 38, R. 3 W., CRAWFORD COUNTY.)

The furnace is located eight miles southeast from Leasburg, its station and depot on the Atlantic & Pacific Railroad. It was erected in 1849, and has made a most creditable record in respect to its production and the quality of its iron.

It is of the open top form with two tuyeres (4 inches in diameter), the blast having a pressure of from two and a half to three pounds, and a temperature of from 500° to 600° Fah. (say 325° Cent., or about the melting point of lead). The hearth is of fine sandstone quarried near Leasburg, standing a campaign of about six months duration.

The blast is furnished by a steam blowing-engine with a steam cylinder of twenty-nine inches diameter. The blast cylinders are two in number, each four feet diameter, and with six feet stroke. Eighteen to twenty revolutions are made per minute. The daily work is therefore 4,127,000 cubic feet of air. The blast is heated by passing thirty syphon pipes, in two rows of fifteen each, entering by eight pipes and returning by seven.

The ores are calcined in heaps of fifteen feet height, and twenty by sixty feet base, holding about one thousand tons. The heaps burn three weeks. A sample each of raw and roasted ores gave as follows (each mean of two analyses):

	Raw Ore.	*Calcined.*
Ferric oxide	93.051	83.305
Ferrous oxide	1.262	10.513
Alumina	0.545	0.557
Manganous oxide	0.262	0.235
Lime	trace	trace.
Magnesia	0.018	0.974
Silica	5.361	5.082
Sulphur	0.046	0.031
Phosphoric acid	0.075	0.174
	100.415	100.871

The metallic iron in the samples analyzed is 66.61 per cent. for the raw, and 66.491 for the calcined, and the amounts of phosphorus are 0.0327 and 0.0759 respectively.

["The ore frequently melts in the interior of heaps and runs or drops down, forming beautiful stalactites, often with a velvet-like crystalline, dark surface." *Schmidt's notes*, 1872.]

Flux: The limestone used in the furnace, is from the third magnesian formation, and is quarried on a level with the tunnel head, a few yards back of the furnace. It is less magnesian or dolomitic than other belts of the same formation examined, and contains:

Calcic carbonate	84.078 per cent.
Magnesic carbonate	12.654 per cent.
Ferric oxide	0.423 per cent.
Alumina	0.564 per cent.
Insoluble matters	2.565 per cent.
	100.284

Besides 0.046 per cent. of sulphur, possibly as pyrites, and a faint trace of phosphoric acid. To every 850 pounds of ore in a charge, 80 pounds of limestone are added, the flux being upwards of nine per cent. of the ore.

Fuel: The wood for charring cuts at an average rate of twenty-five cords from the acre; the cost of chopping is sixty cents per cord, the latter yielding in the heaps from thirty-five to forty bushels of charcoal. Five cents per bushel is paid for charring, deducting, however, the cost of cutting the wood. A heap contains from forty to fifty cords; burns from eight to ten days, and cools five or six days.

The wood is mostly oak and hickory; the charcoal has four per cent. ash and about the same amount of hygroscopic water. Twenty bushels (say, 360 pounds*) are added with a charge, or, with an average daily production of 20.2 tons of pig, the consumption of coal is 0.9115 tons per ton of iron made.

Products: The four qualities of *pig* made are No. 1 (73 per cent. of the total production), No. 2 (16 per cent.), No. 3 (8 per cent.), and No. 1 (3 per cent.). Most of the iron is used in St. Louis; the No. 2 is sold for car-wheel foundry purposes, and the white is utilized at the bloomeries of South St. Louis. The general grades show the following amounts of foreign matters:

	I.	Mottled.	3	White.
Silicon	0.8260	0.541	0.768	1.180
Manganese	0.2419	0.374	0.273	0.266
Sulphur	0.0256	0.043	......	0.047
Phosphorus	trace.	trace.	trace.	trace.

---

*The coal at the Scotia property weighs more than eighteen pounds per bushel, as there is less admixture of poplar woods used; the weight is still used in this for sake of uniformity.

In making one ton of pig 0.716 ton of *slag* is produced, the constituents of which, according to our analyses, are:

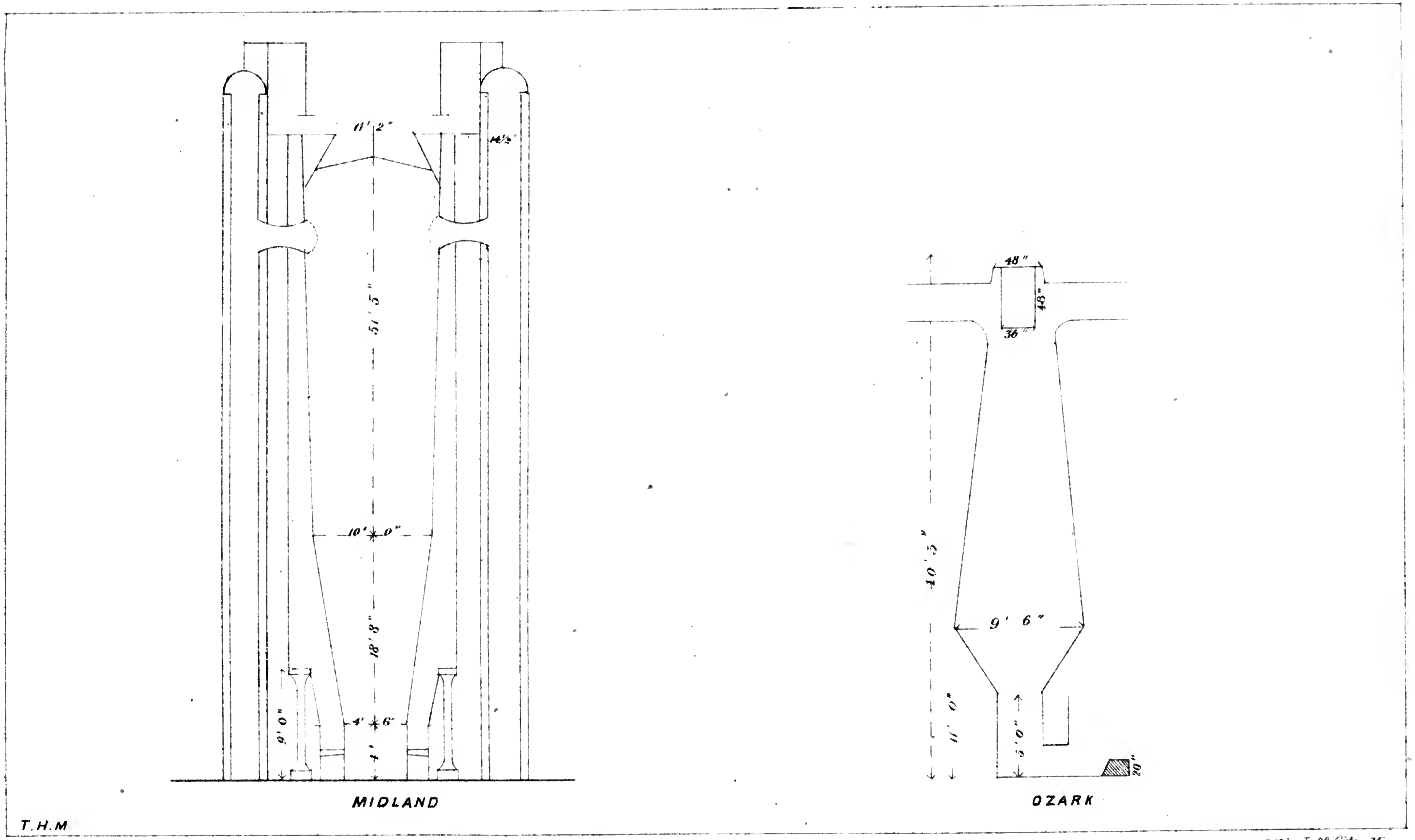
Silica	51.676 per cent.
Alumina	6.437 per cent.
Ferrous oxide	0.871 per cent.
Lime	22,065 per cent.
Magnesia	16.399 per cent.
Soda	1.003 per cent.
Potassa	0.652 per cent.
Calcium sulphide	0.159 per cent.
Phosphoric acid	0.311 per cent.
	99.673
Phosphorus	0.136
Sulphur	0.076

The oxygen of the bases is 16.44; that of the silica is 27.56, giving ratio of 1:1.7 nearly, and showing as a mixture of bi- and singulo-silicates.

The foregoing figures give the following distribution of the raw material for the production of one ton of pig, in the Scotia furnace:

Ore 1,855 tons, containing	Iron.	Slag.	Gas.
Iron	0.954	......	......
Oxygen	......	......	0.309
Ferrous oxide	......	0.590	......
Oxygen	......	......	0.007
Silicon	0.011	......	......
Oxygen	......	......	.011
Silica, alumina, magnesia, etc	......	0.349	......
Flux, 0.175 tons, containing:			
Silica	......	0.005	......
Lime	......	0.083	......
Magnesia	......	0.011	......
Carbonic acid	......	......	0.076
Charcoal. 0.9115 tons, containing:			
Carbon	0.035	.....	0.753
Ash	......	0.036	......
Moisture	......	......	0.045
Air, 7.604 tons, containing:			
Oxygen	......	......	1.750
Nitrogen	......	......	5.811
Moisture	......	......	0.043

By using the same factors employed for the calculation of the Maramec, with the new variables given in the above, the number of



Regan & Carter Lith. Jeff City Mo.

SECTION'S OF "MIDLAND" AND "OZARK" FURNACES.

PLATE II.

ton units of heat necessary in the Scotia furnace may be approximated closely enough for purposes of comparison:

Ton units of heat necessary:	Units.
For reduction of the iron	1697
" carbon impregnation	87
Expulsion of carbonic acid from limestone	65
Decomposition " " "	243
Decomposition of water from blast and charcoal	293
" " silica	86
Fusion of the pig	330
" " slag	355
Transmission through the furnace walls	100
Carried off in the tuyere water	14
" " " " gases	313
	3582

In this case the hot blast furnishes:

7.604 tons × 325° × 0.239 =	586
Leaving	2993

to be furnished by the fuel in the furnace. If 24 per cent. of the carbon of the fuel were burned to carbonic acid, and 76 per cent. burned to carbonic oxide, this calorific power would be produced, while if the same quantity of carbon were consumed in the propotions given by Gruner for the carbon burned to carbonic acid and oxide, there would be produced 3.184 ton units. The necessary heat is therefore 94 per cent. of the ideally possible.

The record of the results reached by this furnace are confirmatory of the view that the fuel consumption must approach very closely to the possible minimum. (For dimensions, etc., see plate 1.)

MIDLAND FURNACE.

(SEC. 2, T. 37, R. 4 W., CRAWFORD COUNTY.)

The Midland furnace was put in blast April 10, 1875. From the accompanying section [Plate 2] it will be seen that in its construction there are features differing from those ordinarily obtaining in the charcoal furnaces of this State; but the campaign was too short to determine how far these novelties were advantageous.*

The *ore* used was mined at the Ferguson bank (SW¼ Sec. 18, T. 34, R. 5 W.) near Salem, Dent county. The mean of two analyses (after

---

* This furnace has again gone into operation after changing form to that of the Scotia. Work was resumed in April of the present year (1877).

calculating the small amount of ferrous to ferric oxide) gave the following composition for the ore:

Ferric oxide	81.942	per cent.
Alumina	8.760	per cent.
Silica	7.661	per cent.
Lime	0.956	per cent.
Magnesia	0.964	per cent.
Sulphur	0.038	per cent.
Phosphoric acid	0 173	per cent.
	100 494	
Metallic Iron	57 36	per cent.
Phosphorus	0.056	per cent.

The *flux* is a magnesian limestone containing:

Calcic carbonate	51.931
Magnesia carbonate	38.871
Ferric oxide and alumina	0.785
Insoluble matters	7.478

besides 0.028 of sulphur and 0.050 of phosphoric acid (equivalent to 0.022 of phosphorus).

The *Charcoal* contained 4.97 per cent. moisture, and 8.65 of ash matters. The charge consisted of 48 bushels coal (say 864 pounds), 1,500 pounds ore and 450 pounds limestone (=30 per cent. of the ore). The average run of pig was twenty-two tons daily, and the consumption of raw materials per ton of produced iron was about 1.737 tons ore, 0.522 ton flux, 1.171 tons charcoal and 9.560 tons air. The blowing cylinder is five feet in diameter, with a four feet stroke; the number of revolutions per minute is 25. The blast left the tuyeres (four in number) with a pressure of 2½ pounds; hot blast was used, the temperature being, probably, about 325° C.

In a specimen of this pig we found:

Graphitic carbon	2.780
Combined carbon	0.360
Silicon	0.566
Manganese	0.673
Phosphorus	trace.
Sulphur	0.028
Iron, by difference	95.593
	100.000

A specimen of the slag, taken at the same time, gave a per centage composition as under:

Silica	41.675
Alumina	13.439

Ferrous oxide	2.813
Lime	30.203
Magnesia	9.343
Potassa	0.537
Soda	0.446
Calcium sulphide	0.322
Phosphoric acid	0.306
	99.184
Phosphorous	0.133
Sulphur	0.143

The silica contains 22.221 of oxygen, and the aggregate of oxygen in the bases is 18.556; giving a ratio of 1 1-5 to 1, and showing the slag to be composed of singulo and bisilicates in the proportion of two of the former to one of the latter [or $2\,(2\ \mathrm{RO\ SiO_2}) + \mathrm{RO\ SiO_2}$]. The production of slag per ton of iron would be by calculation 0.674 ton.

We have fewer figures for arriving at the heat requirements of the Midland furnace than for the others, but from the data above given the following may be calculated and will be approximately correct:

DISTRIBUTION OF MATERIALS IN THE MIDLAND FURNACE.

(Per ton of pig iron made.)

	Iron.	Slag.	Gas.
Ore, 1.737 tons, containing:			
Iron	0.960	......	......
Oxygen	......	......	0.411
Ferrous oxide	......	0.046	.....
Oxygen	......	......	0.005
Silicon	0.006	......	......
Oxygen	......	......	0.006
Silica Alumina Lime Magnesia	......	0.303	......
Flux, 0.522 tons, containing:			
Silica Lime Magnesia	......	0.288	......
Carbonic acid	......	......	0.220
Charcoal, 1.171 tons, containing:			
Carbon	0.030	......	0.932
Water	......	......	0.580
Ash	......	0.050	......
Blast, 9.560 tons (water, 0.554)	......	......	9.560

Corresponding to this distribution of materials, and with the constants used before, we find:

Ton units of heat necessary	4,260
From which deduct units furnished by blast	743
Leaving to be furnished by 0.982 ton carbon of fuel	3,517

Of the given amount of carbon, 19 per cent. must be burned to carbonic acid, and 81 per cent. to carbonic oxide to produce the above named heat-units to be furnished by the fuel. If the carbon were burned as in the ideally perfect consumption, 4818 heat units would be produced, and the amount actually necessary is 73 per cent. of this ideal.

MOSELLE FURNACE.

(Sec. 14, T. 42, R. 1 E., Franklin County.)

This furnace was built in 1857, and is owned by Messrs. I. H. Brown & Co., of Youngstown, Ohio. It is located within three-fourths of a mile of Moselle Station, on the line of the Atlantic & Pacific Railroad. [For dimensions, etc., see plate I.]

The ore used was mostly from the Benton Creek Bank, in Crawford county, (Sec. 32, T. 36, R. 5 W.), on the line of the Salem & Little Rock Railroad, (Description of this deposit is to be found in Report of 1872, pp. 134, *et seq.*) Some limonite ore from a deposit in the vicinity of and belonging to the furnace company, is also used. The following are the results of analyses of these ores:

	Hematite.		Limonite.
Ferric oxide	92.486		73.766
Ferrous oxide	3.598		2.551
Alumina	0.916		5 379
Lime	trace.		1.116
Magnesia	0.639		0 813
Silica	3.132		5.703
Sulphur	0.042	Sulphuric acid	0.106
Phosphoric acid	0.007		....
Water	....		10 290
	100.820		99.902
Metallic iron	67.530		53.630
Sulphur	0.042		0.113
Phosphorus	0.003		None.

The *flux* is a magnesian limestone, composed as under:

Calcic carbonate	53.061
Magnesic carbonate	37.122
Alumina and ferric oxide	1.531
Insoluble matters	8.419
	100.133

It contains also, by special examination, 0.061 of sulphur and 0.015 of phosphorus (0.034 phosphoric acid). To six hundred pounds of ore in one charge, 150 pounds of flux (or 25 per cent.) and 22 bushels of charcoal are added. The charcoal contains 4.03 per cent. moisture and 2.24 of ash. From one hundred to one hundred and ten charges are made every twenty-four hours. The yield is from seventeen to twenty of pig iron daily, and the materials consumed per ton of iron are 1.651 tons of ore, 0.421 ton of flux, 1.094 tons of charcoal and 6.025 tons of air.

The results of analyses of the several grades of pig iron are given below:

	No. 1.	No. 2.	No. 3.	Mottled.	White.
Carbon, graphitic	1.860	2.608	1.012	0.812	0.346
Carbon, combined	1.683	0.632	2.604	2.021	3.338
Silicon	not est.	0.459	0.634	0.413	1.556
Manganese	0.570	trace	0.154	0.185	0.095
Phosphorus	trace	trace	trace	trace	0.039
Sulphur	0.150	0.099	0.065	0.177	0.112
Iron	....	96.202	95.531	96.392	94.514
		100.000	100.000	100.000	100.000

The slag produced is, by calculation, 0.404 ton to the ton of iron, and gives on analysis:

Silica	48.819
Alumina	4.093
Ferrous oxide	2.424
Lime	27.596
Magnesia	14.737
Soda	0.781
Potassa	0.650
Calcium sulphide	0.622
Phosphoric acid	0.277
	99.999

Sulphur	0.276 per cent.
Phosphorus	0.120 per cent.
Oxygen ratio of bases to acid	1: 1 3-5 nearly.

DISTRIBUTION OF MATERIALS.

	Iron.	Slag.	Gas.
Ore, 1.651 tons composed of:			
Iron	0.952		
Oxygen	......	......	0.308
Ferrous oxide	......	0.062	......
Oxygen	......	......	0.007
Silicon	0.015	......	......
Oxygen	......	......	0.015
Silica, alumina, lime, magnesia	......	0.077	......
Water	......	......	0.075
Flux, 0.421 ton, containing:			
Silica	.....	0.035	......
Lime	......	0.125	......
Magnesia	......	0.075	......
Carbonic acid	......	......	0.183
Charcoal, 1.094 tons containing:			
Carbon	0.030	......	0 998
Ash	......	0.023	......
Moisture	......	......	0.044
Air, 6.025 tons containing:			
Oxygen	......	......	1.386
Nitrogen	......	......	4.623
Water	......	......	0.016

The total number of heat units required per ton of iron is found to be 3,805, of which the blast furnishes 464, leaving 3,341 to be produced by the fuel. If 84½ per cent. of the carbon were burned to cabonic oxide, and 15½ per cent. to carbonic acid this calorific power would result; if, however, the carbon were consumed in the proportions theoretically adopted, 4,912 heat units would be produced; hence, the number of units required is equal to 67 per cent. of the theoretically possible amount.

BOULLINGER CREEK FURNACE.

(SEC. 4. T. 39, R. 18 W; CAMDEN COUNTY.)

[For dimensions, etc., see plate III; and for character of deposits furnishing the ores, see Schmidt, in report of 1872, 183 *et seq.*] The furnace used the limonite and specular ores of the Osage river district with an average furnace yield of sixty per cent, the ores being previously calcined. A charge consisted of 600 pounds of roasted ore, 20 bushels (360 pounds) of charcoal and from 30 to 50 pounds of lime-

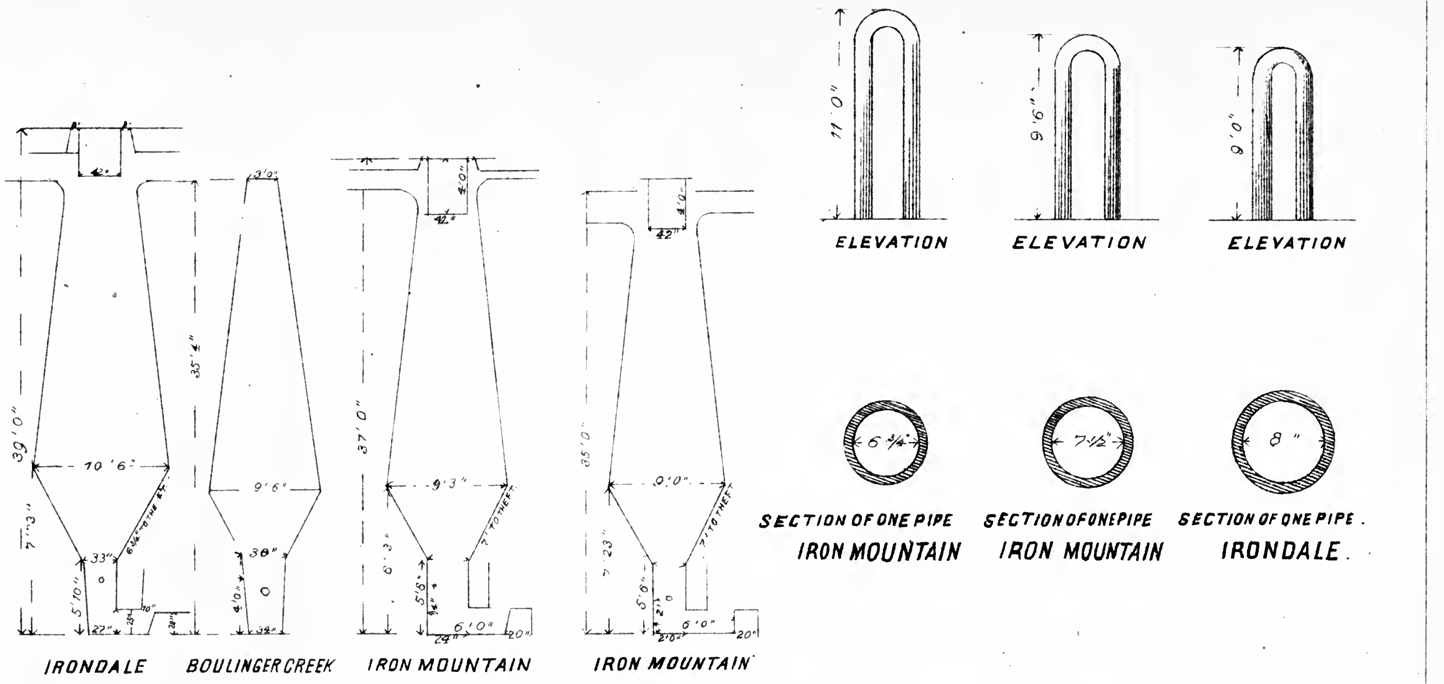


PLATE III.

S.E. CHARCOAL FURNACES.
FROM NOTES BY SCHMIDT.

Regan & Carter Lith. Jeff. City, Mo.

stone. The production was from 15 to 20 tons per twenty-four hours, with a consumption of 150 bushels (1.205 tons) per ton of pig.

PILOT KNOB FURNACE.

(IRON COUNTY.)

[Plates III and IV.]

[Notes of Dr. Adolph Schmidt.] Roasting the ores is done with the charcoal braze (or fine charcoal from the charring) in heaps of thirty feet width, fifty-feet length and twenty feet height, containing 3,200 tons. The layers of ore, eighteen inches thickness, alternate with layers of the braze of from seven to eight inches thick; the heap burning from two to three months. A charge is made up of 22 bushels of charcoal, 760 pounds roasted ore, broken to nut size, and consisting of three-fourths Pilot Knob and one-fourth Shepherd Mountain and 17 pounds of raw limestone per hundred of ore (= 129 pounds per charge.) In a campaign of the furnace, lasting sixteen months and twenty-three days, there were produced 8,267 tons of pig, with the consumption of 910,000 bushels of charcoal, or 134 bushels per ton produced. In that time the average daily production was 16.4 tons. The iron is very grey, looking almost like coke iron. Pilot Knob ores do not melt as those from Shepherd Mountain, but the latter do not make enough cinder; the best results are obtained with the mixed ores. The pressure of the blast is from 2½ to 3 pounds; the temperature is between 700° and 800° Fah., and the air enters the furnace through three three-inch tuyeres. The heating of the boilers and of the blast is done by the furnace gases; the hot blast apparatus consisting of six rows of double pipes—one ton weight each and cast in one piece. The blowing apparatus consists of two upright blowing cylinders, four feet diameter and six feet long, the pistons connected by gearing with one horizontal steam cylinder, two feet diameter and five feet long. From six to eight revolutions per minute are made when the pipes do not leak. (For dimensions of furnace, etc., see plate IV.]

IRON MOUNTAIN FURNACES.

(IRON COUNTY.)

[Plate III.]

[Schmidt's Notes.] The principal object of roasting is to facilitate the breaking of the ore, and surface ore, which is partly small and generally easier to break, is often not roasted at all. There seems to be a further advantage in the roasting, for it is claimed that a higher daily production can be reached by the use of such ores, which may be accounted for by the splitting up and softening by roasting, so that less time is required for reduction and carburretting. The roasting is done in a similar manner to that practiced at Pilot Knob.

There are two old furnaces, the western, No. 3, and the eastern, No. 2; the first holding from one hundred and fifty to two hundred bushels more than the other. Each has two tuyeres of 3 inches diameter. The blowing engine for No. 3 furnace has two cylinders, two and a half feet diameter each, and five feet stroke, with from 18 to 22 revolutions per minute. For No. 2 there are three cylinders, two and three-fourth feet in diameter and five feet stroke, and with from 12 to 15 revolutions per minute. The pressure of the blast for the latter is 4 to 5 inches of mercury, (2 to 2½ pounds,) and for the first, 3 to 4 inches of mercury (1½ to 2 pounds.)

The furnaces are now smelting only surface ores, which work harder and make a somewhat harder iron. The charge is made up of 700 pounds of raw ore, 70 pounds limestone and 20 bushels of charcoal, and runs gray iron. From 55 to 58 charges are made daily, giving from 18 to 20 tons pig iron. The latter is increased from 1½ to 2 tons when roasted ores are used. Furnace No. 2 requires 124 bushels of coal per ton of iron, and No. 3,119 bushels. Both furnaces run very irregularly, making frequently mottled and white iron; this is especially true of No. 2 furnace.

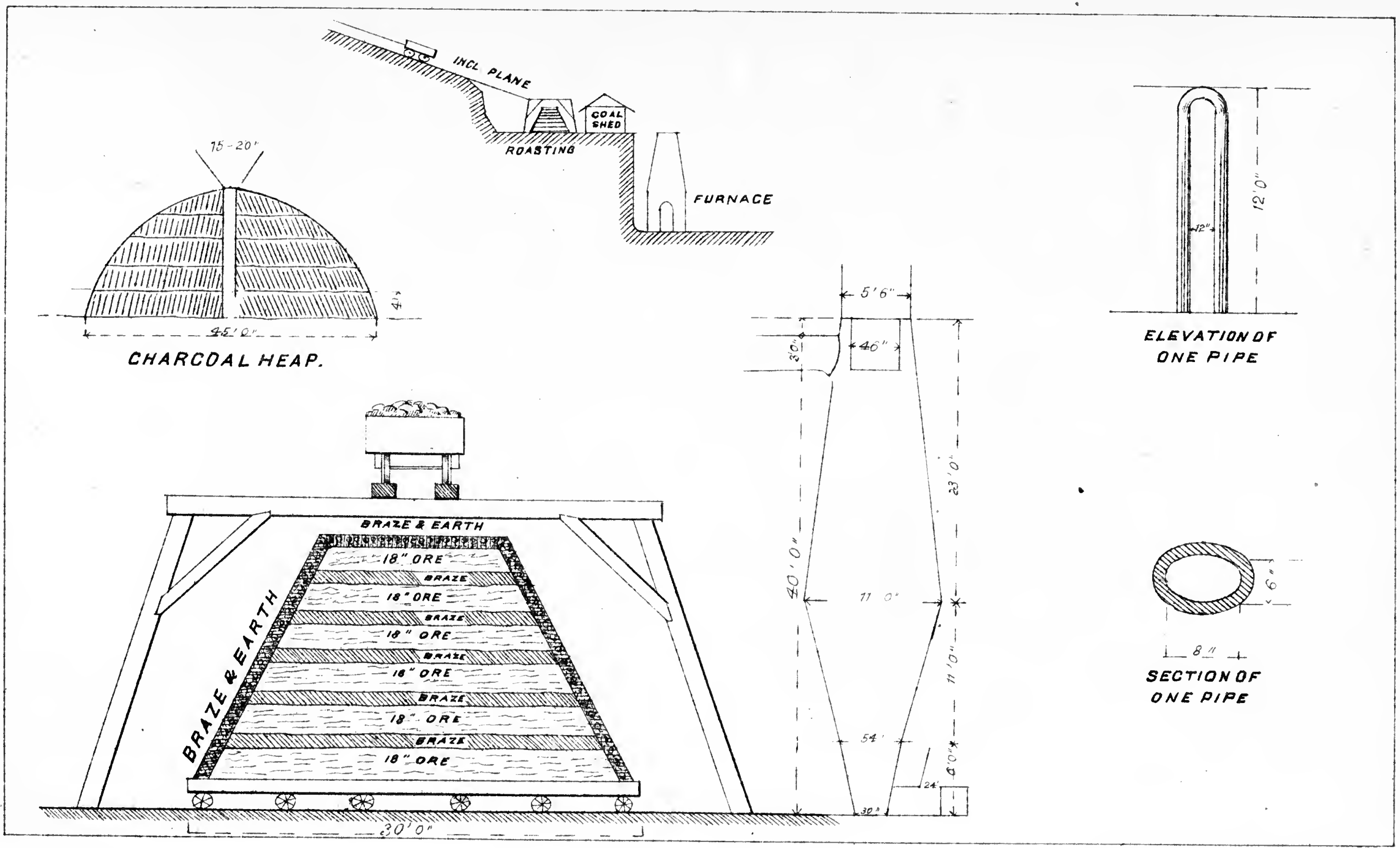
The Iron Mountain ores require a strong blast; they are apt to run dry, making too little cinder. The limestone used at the furnaces, though being magnesian and of the same general character as that used at Pilot Knob, looks considerably purer, containing less green clay and less red silicious matter. This may partly account for the fact that much less of it is used in the Iron Mountain charges; besides there are the further facts that the ores are richer and less attention is paid to the regular production of gray iron.

IRONDALE FURNACE.

(WASHINGTON COUNTY.)

[From Dr. Schmidt's notes]—This furnace runs on Iron Mountain ores, in mixture with limonites from the Marquand or the Cornwall banks in Madison county, or from Marble Hill, Stoddard county, or from several localities within two and a half miles of the works. The charcoal braze, of which from eight to ten per cent. are produced in the burning, is used for roasting the ores. A furnace charge is composed of 650 pounds roasted Iron Mountain ores, 70 pounds roasted

PLATE IV.



Regan & Carter Lith. Jeff. City, Mo

PILOT KNOB FURNACE, ETC.

FROM NOTES BY A. SCHMIDT PH.D.

limonites from the various localities, and 50 pounds limestone and 21 bushels of charcoal. One hundred and eight charges are made per day.

The limestone is quarried close to the furnace. It is magnesian and is rather cleaner than that used at the Pilot Knob furnace, but not so clean as the Iron Mountain flux.

The charring is done by contract, and at distances ranging from two to ten miles from the furnace. A pit contains usually about forty cords and burns from twelve to sixteen days. A cord of wood—chiefly oak and hickory—yields from forty to fifty bushels of charcoal.

The blowing engine has a steam cylinder twenty-two inches, and one blast cylinder of five feet diameter, with six feet stroke. From ten to twelve revolutions are made per minute. The pressure of the blast is 4¼ inches of mercury in the engine room, and 3 inches at the furnace (1½ pounds). The heating apparatus, (which, like the boilers is heated with the furnace gases) consists of two double ovens, containing each a double row of ten pipes, or forty pipes in the full apparatus. Each pipe weighs about nineteen hundred pounds; the pipes are cast in St. Louis. The temperature is about the melting point of lead. (For form of pipes, dimensions and outline of furnace, see plate III.)

The furnace has two tuyeres (3¼ inches diameter). A campaign lasts from nine to fourteen months, and ends with the smelting of the hearth stone, which is brought from the Cedar Creek quarries about six miles distant. Four taps are made in twenty-four hours. There is no stoppage at tapping, and the hearth is closed at once. The product is 22 tons daily, and the consumption of fuel is 125 bushels of charcoal per ton of iron made.

Attention is called to the fact that the production of iron is usually larger in reality than the furnace records show. This arises from the practice, in weighing each day, of noting only the thousands, and disregarding the hundreds that are over. There is also a wide range of values of the standards of weight, tending to complications and somewhat confusing in calculating ratios of products to raw materials. In furnace practice, one ton of limestone is 2,268 lbs., and the same weight holds good for the ore and cinder; a ton of iron is 2,240 pounds, and one ton of fuel equals 2,000 pounds. In the calculations on the preceding pages, the raw materials have been reduced to the standard ton for pig iron (2,240 lbs). There are also great differences in the weights of charcoal; at Iron Mountain a bushel of the coal is estimated as weighing from 18 to 25 pounds; at Irondale from 22 to 28 pounds; at Pilot Knob from 22 to 25 pounds, while at other localities the weights are from between 15 and 22 pounds. In such, almost inextricable confusion, it was deemed best to select an uniform weight of eighteen pounds per bushel, for making comparisons of the fuel—consumption at the various furnaces, and reaching thereby results which, while they may be somewhat too low for some of the furnaces, probably are not widely different from the actual expenditures of charcoal necessary to the production of one ton of pig iron at the Missouri furnaces.

There are also differences in the readiness with which ores from different localities give up their oxygen to carbonic oxide. Thus the ores of the Atlantic and Pacific region are generally more easily reduced than those of the Iron Mountain district, the latter allying themselves more closely with the "strengflüssig," and the former coming near to the "leichtflüssig" ores of the German writers.* While this difference certainly does make itself felt in actual furnace experience and practice with Missouri ores, it can hardly be regarded as sufficiently well understood or established, nor as being of sufficient magnitude to enter as a factor in comparing the results of iron smelting in this State. Ignoring this little understood, and in reality, triffling difference, the differences in the results of the various furnaces as far as regards the charcoal necessary to the production of iron must be sought for elsewhere.

It certainly seems that the *time* involved in those various reactions which result in the production of iron, must be only variable where furnaces, working under similar conditions respecting ores and fuel differ in height and other dimensions. Enlarging or diminishing the volume of the furnace, and thereby lengthening or shortening the period of working beyond normal limits, not yet defined, may cause less of calorific power by making the reactions between CO and the ore, and CO_2 and the carbon, less or more rapid, and thus destroy that balance, between the finally escaping carbonic oxide and carbonic acid, which has been hypothetically taken to represent the most advantageous combustion of carbon possible in the blast furnace. If the proper velocity of descent could be determined with accuracy and rigor, the proper proportioning of the height and volume of the furnace to the work to be done would become easier ; at present it is known that with similar ores it must be less with coke than with charcoal, and hence, smaller sized charcoal furnaces have relatively larger

---

*On this point Bell writes (*Chemical Phenomena of Iron Smelting*, p. 422): "If these words are to be taken in their literal sense of comparative susceptibility to fusion, their use, in my opinion, may lead to error. * * * The actual cause of lesser consumption of combustible in small furnaces, I have conceived and described as being due to difference in susceptibility of reduction, and not of fusion."

production than the larger apparatus necessary for coke or raw coal. Hence, also, the reactions take place less rapidly in the latter than in the former, and a smaller weight of charcoal is necessarily consumed to the unit of iron than is practicable with the other fuels, while it also follows that, the necessary heat remaining about constant for the ton of iron however produced, more of the small weight of carbon in the charcoal must be burnt to carbonic acid (with its 8,080 heat units) to produce that heat effect than will be necessary with the larger weight of carbon in the coke. This last is corroborated by the results of analyses of gases from charcoal furnaces, always showing a much larger per centage of carbonic acid than do those of gases from coke furnaces.

These points connected with the rapidity of descent in the furnaces will, perhaps, be more evident to a non-professional reader by an examination of the following table, which contains in the last column the number of cubic feet of furnace room concerned in the production of one ton of iron in twenty-four hours, calculated from the data in the fourth and fifth columns :

FURNACE	FUEL.	BLAST.	Volume of furnace—cubic feet........	Production per day—tons.......	Fuel per ton of Iron—tons......	Cubic feet per ton of Iron per day.
Maramec	Charcoal	Cold	1,045	13.8	1.221	75
Boulinger	Charcoal	Cold	1,030	17.5	1.205	60
Scotia	Charcoal	Hot	1,489	20.2	0.912	74
Midland	Charcoal	Hot	3,435	22	1.171	156
Moselle	Charcoal	Hot	1,191	17.5	1.094	64
Hamilton	Charcoal	Hot	....	......	....	......
Pilot Knob	Charcoal	Hot	1,889	16.5	1.079	118
Irondale	Charcoal	Hot	1,563	22	1.911	71
Iron Mt. No. 3	Charcoal	Hot	1,142	19	0.955	60
Iron Mt. No. 2	Charcoal	Hot	1,023	19.3	0.997	59
Mo. Furnace Co	Coke and raw coal	Hot	6,461	?	2.375	......
South St. Louis	Coke and raw coal	Hot	5,514	35	2.457	155
Vulcan	Coke and raw coal	Hot	6,328	33.7	2.615	187
Grand Tower	Coke and raw coal	Hot	8,630	37.5	2.380	266
Big Muddy	Coke and raw coal	Hot	8,100	33.2	2.279	243

The last two are Illinois furnaces working on Missouri ores. The figures for them and for all the coke furnaces are taken from the notes of Dr. Schmidt (1872,) which were collected for the Geological Survey but never made public.* Even at this late date these notes possess so much value as to make them worthy of a more permanent and more accessible form. They are therefore produced here with but slight changes in the phraseology. The notes of the Illinois furnaces are omitted.

[*Dr. Schmidt's Notes on the St. Louis Industrial District.*]

SOUTH ST. LOUIS IRON COMPANY.

(CARONDELET.)

[Plates V and VI.]

Two blast furnaces situated a few hundred yards south of Station 2, Carondelet.

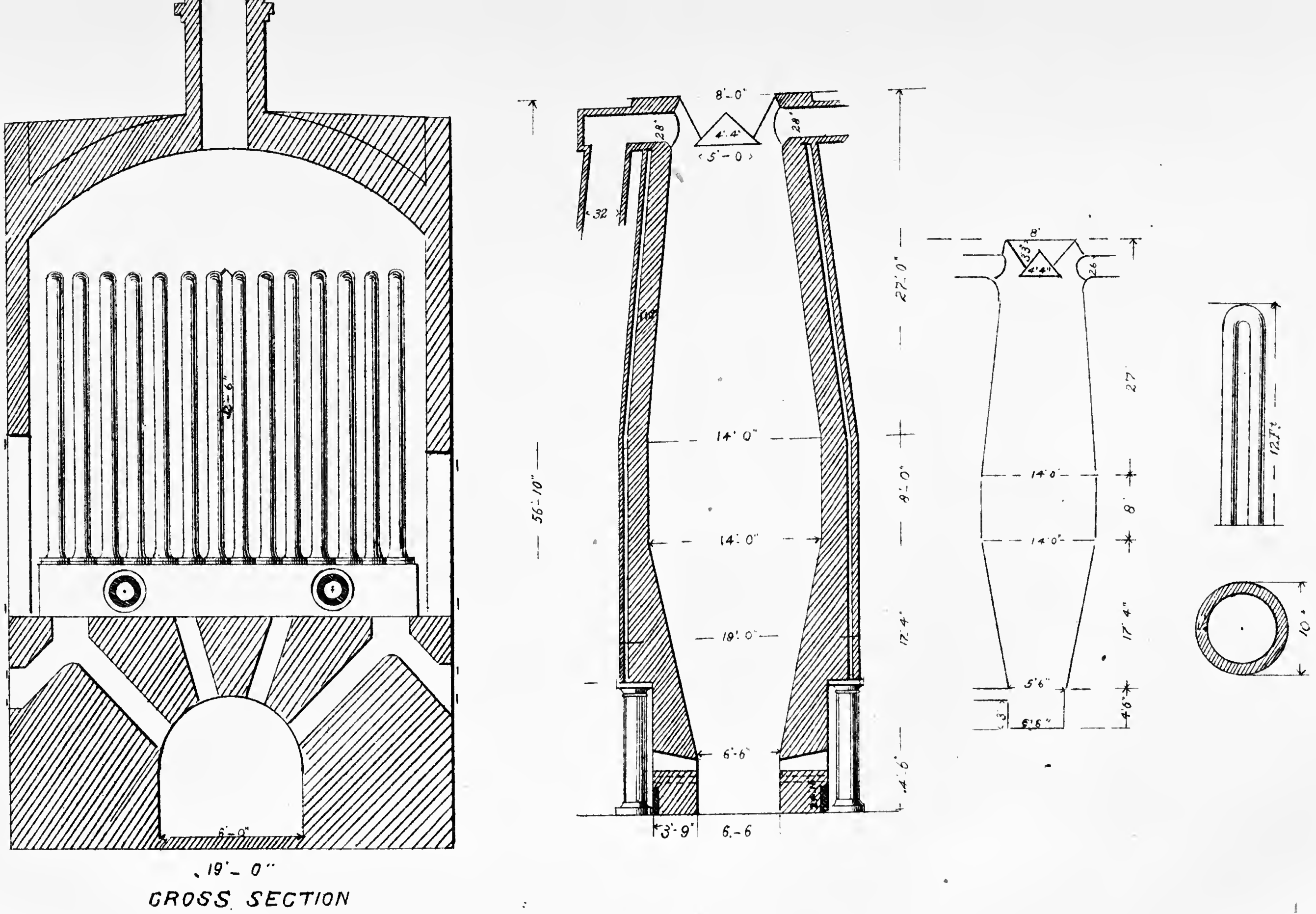
Ores : Iron Mountain specular ore, principally; surface ore and bank ore mixed, usually about one-third of the former, but not always in regular proportion. No difference has been noticed between these two kinds of ore, although the surface ore has never been run alone, while the vein ore has. Iron Mountain ore alone works too dry, and is therefore mixed with some brown hematite. About a year and a half ago, Pilot Knob ore was tried, but complaints were made of the lack of strength in the iron for foundry and for nail works. Brown hematite was not used at that time; the mixture contained twenty-five per cent. Pilot Knob ore. There was no considerable difference in the yield noticed.

The brown hematites come from Southeast Missouri, from the vicinity of Fredericktown and Marquand, on the Iron Mountain Railroad. The soft, red hematites from the line of the Atlantic and Pacific Railroad are sometimes used to the amount of twenty to twenty-five per cent. of the charge. It is preferred for softer and purer irons. The same amount of flux is used for the red or the brown hematites; the latter are used only when the red is not to be had.

Coal : Brazil, Indiana coal, reaches the works by the Indianapolis and St. Louis Railroad. Big Muddy coal, from Grand Tower, Illinois, is brought by boat one hundred and twenty miles up the river. When coke is to be had, it is mixed with the coal and improves the running of the furnace on grey iron. The coke comes from Connels-

---

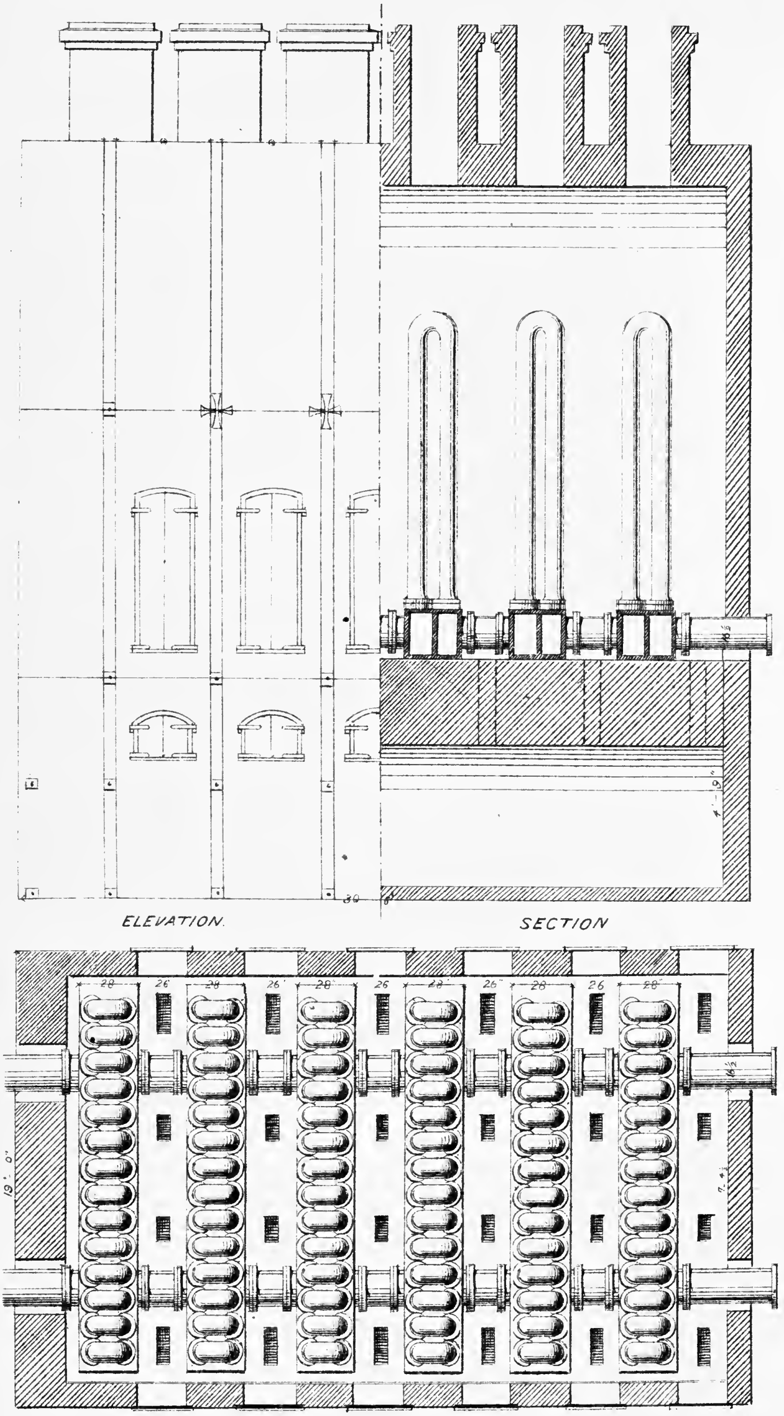
*It was the intention of the writer to have emended these notes, bringing them down to a later period, and to have illustrated the furnaces work by analyses of the materials and products. On account of the unfavorable condition of the iron industry this work was so long delayed that other laboratory work took precedence, and time did not allow its being completed to date.



Begar & Carter Lith. Jeff. City Mo.

SOUTH SAINT LOUIS FURNACE WITH CROSS SECTION OF HOT BLAST APPARATUS.

PLATE V.



ELEVATION.

SECTION

HORIZONTAL SECTION.

HOT BLAST APPARATUS AT S^TH ST LOUIS

(SCHMIDT.)

PLATE VI.

Regan & Carter Lith. Jeff. City, Mo.

ville, and the necessity of transportation by river renders the supply uncertain. Coke from East St. Louis is now being tried.

Blowing Engine—Old: One steam cylinder 34 inches diameter and 7 feet stroke; the blowing cylinders, 16 inches diameter and 7 feet stroke; two fly-wheels, 13 tons each, 18 feet diameter. Temperature of blast 800° to 900° Fah.; pressure, 4¾ pounds; pressure of steam 80 to 85 pounds; revolutions, 16. *New Engines:* Two, horizontal; steam cylinders 29 inches each; two blowing cylinders, 72 inches, each; length of stroke, 6 feet; one fly wheel, 12 tons.

Hot Blast Apparatus: No. 1, 16 feet wide, 25 feet long, with 65 syphon pipes, 12 feet high, and 5 bed pipes. There are therefore, five systems of thirteen pipes each. No. 2, 28 feet 4 inches long, 17 feet, 8 inches wide, with 78 syphon (12 feet) and 6 bed pipes.

Results for the year ending May 31, 1872.

Number of charges made		11,065
Fuel used:		
Coal	18,904 tons.	
Coke	3,639 tons.	
		22,543 tons.
Ores used:		
Iron Mountain	13,113 tons.	
Tenn. brown hematite	241 tons.	
Hermitage, red hematite (A. & P. R. R)	342 tons.	
		13,696 tons.
Puddle cinder		739 tons.
Flux		3,345 tons.
Product:		
Foundry, No. 1	2,710 tons.	
Foundry, No. 2	1,227 tons.	
Foundry, No. 3		
Mill iron, No. 1	4,441 tons.	
Mill iron, No. 2	776 tons.	
Castings	25 tons.	
Total	9,179 tons.	

[From the above the following are calculated: Yield of pig from the ore, 67 per cent.; fuel consumed per ton of ore smelted, 1.6 tons; fuel, per ton iron produced, 2.4 tons; flux per ton of ore, 0.25 ton; flux, per ton of iron, 0.36 ton.—C. P. W.]

[Dr. Schmidt's notes on markets, freights, etc., are, for obvious reasons, omitted.—C. P. W.]

Missouri Furnace Company, (Plate VII,) July 5, 1872. Two blast furnaces situated near Station 2, Carondelet, a few hundred feet north of the station. One furnace running, the other in repairs. *Ores:* Iron Mountain, arriving by I. M. R. R.; the cars are pushed into the shed, and in unloading the large pieces are separated by hand from the fine. The coarse ore is hoisted on an incline to a Blake breaker, and crushed to a fist size. No roasting of the ore; surface ore works equally well with the other. Pilot Knob ores have never been worked. Ten per cent. brown hematite, used, uncrushed.

Coal: Usually run with Big Muddy coal with one-fourth coke. Indiana coal is used only in case of necessity, but does not make so good quality of iron. Coal alone does not affect the quality of the iron, but the furnace is liable to hang and does not run so fast as with the mixed coal and coke, the latter giving from 30 to 45 tons of iron against 30 to 35 tons with the coal alone. Coarse coal works almost as fast as coke.

Connellsville or Pittsburg coke is used, reaching the works generally by barges down the Ohio. Connelsville coke weighs about 38 pounds per bushel, and the cost by river transportation is 18 cents against 20 cents per bushel for the same by rail. The difference in the price is therefore $1.20 per ton , estimating 60 bushels to the ton.

Blowing Engine, etc.—One engine, two horizontal cylinders about 6 feet by 55 feet 3 inches, directly connected with steam cylinder (29 inches [?]); 6 feet stroke; 25 revolutions per minute for one furnace, or 35 for two. Steam pressure, 70 pounds, but only 40 pounds are needed in the cylinders, the reduction being effected through the steam valve. The boilers are six in number, each of 40 feet length.

Hot Blast Apparatus: One system has four rows of eight or thirty-two syphon pipes; the other, six rows of eight or forty-eight pipes. Temperature of blast=900° Fah.; pressure 2½ to 3 pounds.

Furnaces: Have eight tuyeres of four inches diameter. The two furnaces have the same dimensions, No. 1 has, however, a closed top.

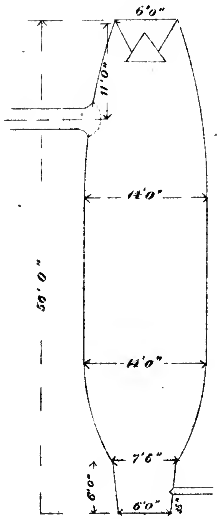
Charge: For *grey* iron:

Iron Mountain ore (uncrushed)	2.500—3,000 pounds.
Cornwall (raw) “) “)	2,000—3,000 pounds.
Limestone	20 per cent.

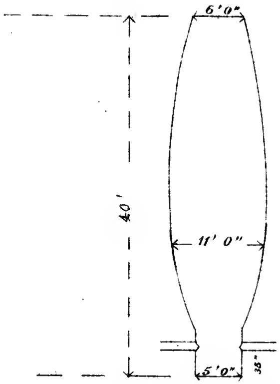
Coal and coke, 6 barrows, at 10 bushels (of 64 lbs each for coal or 35 lbs for coke.)

The yield is 66 per cent. of iron from the ore, and the consumption of fuel is from 2¼ to 2½ tons of mixed coal and coke per ton of iron produced.

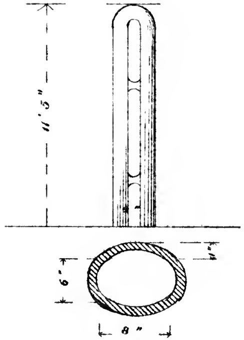
Carondelet Iron Works, near station No. 2, Carondelet: This furnace is under the same management and uses similar materials to those used at the furnaces of the Missouri Company. It was put up in 1852, as an experimental furnace, to test the smelting of Iron Mountain ores with raw coal. The furnace has a closed front, open top,



MO. FURNACE CO.



OLD CARONDELET.



ELEVATION
AND
SECTION
OF
ONE PIPE

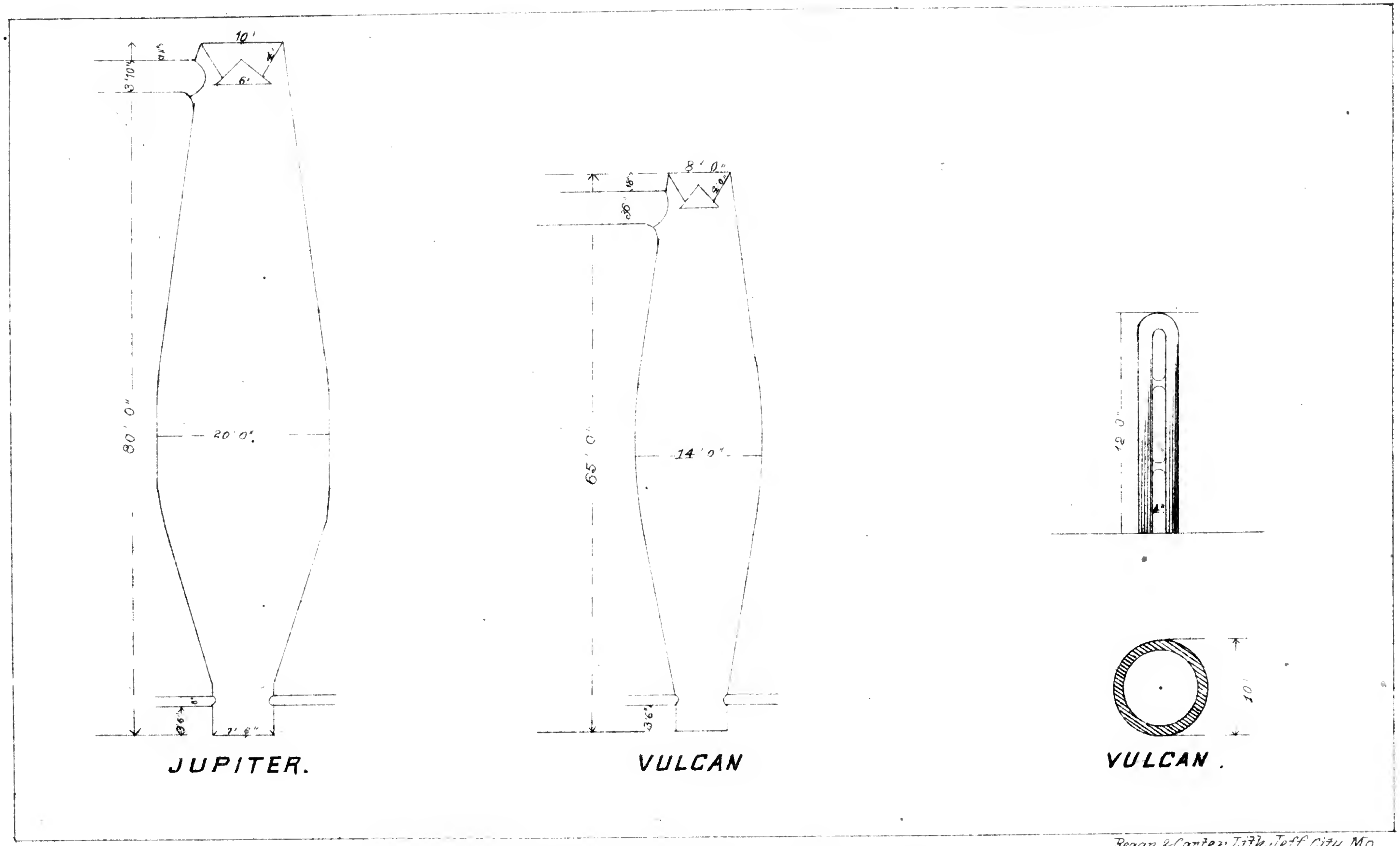
MO. FURNACE CO.

PLATE VII.

SOUTH ST. LOUIS FURNACE'S.

FROM NOTES BY SCHMIDT

Regan & Carter Lith. Jeff City, Mo.



Began & Carter Lith. Jeff. City, Mo.

PLATE VIII.

"JUPITER" AND "VULCAN" AT SOUTH ST. LOUIS and has five tuyeres, none above the tymp. Its production is from eighteen to twenty tons daily.

The hot blast apparatus has two double rows, or about forty pipes (Scotch pistol-pattern). Temperature of the blast 600°–700°; pressure 2½ pounds. Practice and results, the same as at the Missouri Company's furnace.

Vulcan Iron Works: The two furnaces of this company are run chiefly with Iron Mountain ore with ten per cent. puddling cinder. Sometimes mix Pilot Knob ores and sometimes brown hematites from Iron Mountain R. R., or Atlantic and Pacific R. R., or from Osage river (pipe ore) or bog ore from Southern Missouri or Arkansas. The proportion of Pilot Knob ore used is from one-sixth to one-fourth the weight of the Iron Mountain ore with which it is mixed.

The fuel is mixed Big Muddy and Indiana coal and Pittsburg coke—one third of each.

Both of the existing furnaces [July 10, 1872,] are alike, and have the dimensions indicated in the sections [plate VIII]. Each has eight tuyeres. One larger furnace, eighty feet high and twenty feet bosh is being built, especially for Bessemer iron, to run with coke alone, or with coke mixed with a little raw coal.

Diameter of tuyeres, 4 inches; temperature of blast 800°; pressure, 2½ to 3½ pounds.

A charge is 1,500 pounds coal, 1,700 pounds coke, 3,300 pounds crushed ore (Iron Mountain) and cinder, and 1,000 pounds (= 30 per cent.) limestone. Forty charges are made in twenty-four hours to each furnace. The actual production is from 70 to 80 tons of iron from the two furnaces.

Puddling Mill: Eighteen double furnaces; charge 1,050 pounds, with five heats in eleven hours; labor, five men to a furnace; fuel, forty bushels of Belleville, Illinois, coal to the ton of puddle bars.

For head plates for rails, the following mixture of irons is used: 1-6 Tennessee charcoal, (Wayne furnace); 1-6 Wisconsin charcoal iron; ⅓ Iron Mountain charcoal, and ⅓ Vulcan Iron.

Rail Mill, has ten heating furnaces and twenty two inch trains with a capacity of 90 tons of finished rails in twelve hours. The engine is 350 horse power.

The consumption of fuel for puddling, heating and all purposes is eighty bushels to the ton of finished rails.

The analytical results relating to the charcoal furnaces are collected on the following two pages, for convenience of reference. They include the analyses of the fluxes employed, of the slags produced, with the oxygen ratios, and of the several grades of pig iron manufactured.

Recapitulation of Results of Analyses of the Fluxes used, and the Slags and Irons produced at some Missouri Furnaces.

FLUXES OF CHARCOAL FURNACES.

Furnace.	Lime.	Magnesia.	Carbonic Acid.	Oxide of Iron and Alumina.	Insoluble Matter.	Sulphur.	Phosphorus.
Maramec	29.462	20.197	44.367	1 330	4 201	0.036	0.0022
Ozark	31.102	19.603	56.994	0.734	3.795	0.0485	0.0017
Scotia	47.083	6.025	43.624	0 987	2 565	0.046	Trace
Midland	29.081	18.510	43.210	0.785	7.478	0.028	0 022
Moselle	29.714	17.678	42.791	1.531	8.419	0.061	0 015

SLAGS FROM CHARCOAL FURNACES.

Furnace.	Silica.	Alumina.	Ferrous Oxide.	Lime.	Magnesia.	Soda.	Potassa.	Phosphorus.	Sulphur,	Total.	Oxygen ratio.
Maramec	64.746	7.729	2.282	16.973	5.349	1.812	0.702	0.249	0.059	100.299	1 : 3 (nearly.)
Ozark	48.928	16.617	2.573	27.895	0.855	2 202		0.034	0.2005	99.625	1 : 1½
Scotia	51.676	6.437	0.871	22.065	16.399	1.003	0.652	0.136	0.076	99.673	1 : 1.7
Midland	41.675	13.439	2.813	30.203	9.343	0.446	0.537	0.133	0.143	99.184	1 : 1.2
Moselle	48.819	4.093	2.424	27.596	14.737	0.781	0.650	0.120	0.276	99.999	1 : 1.6

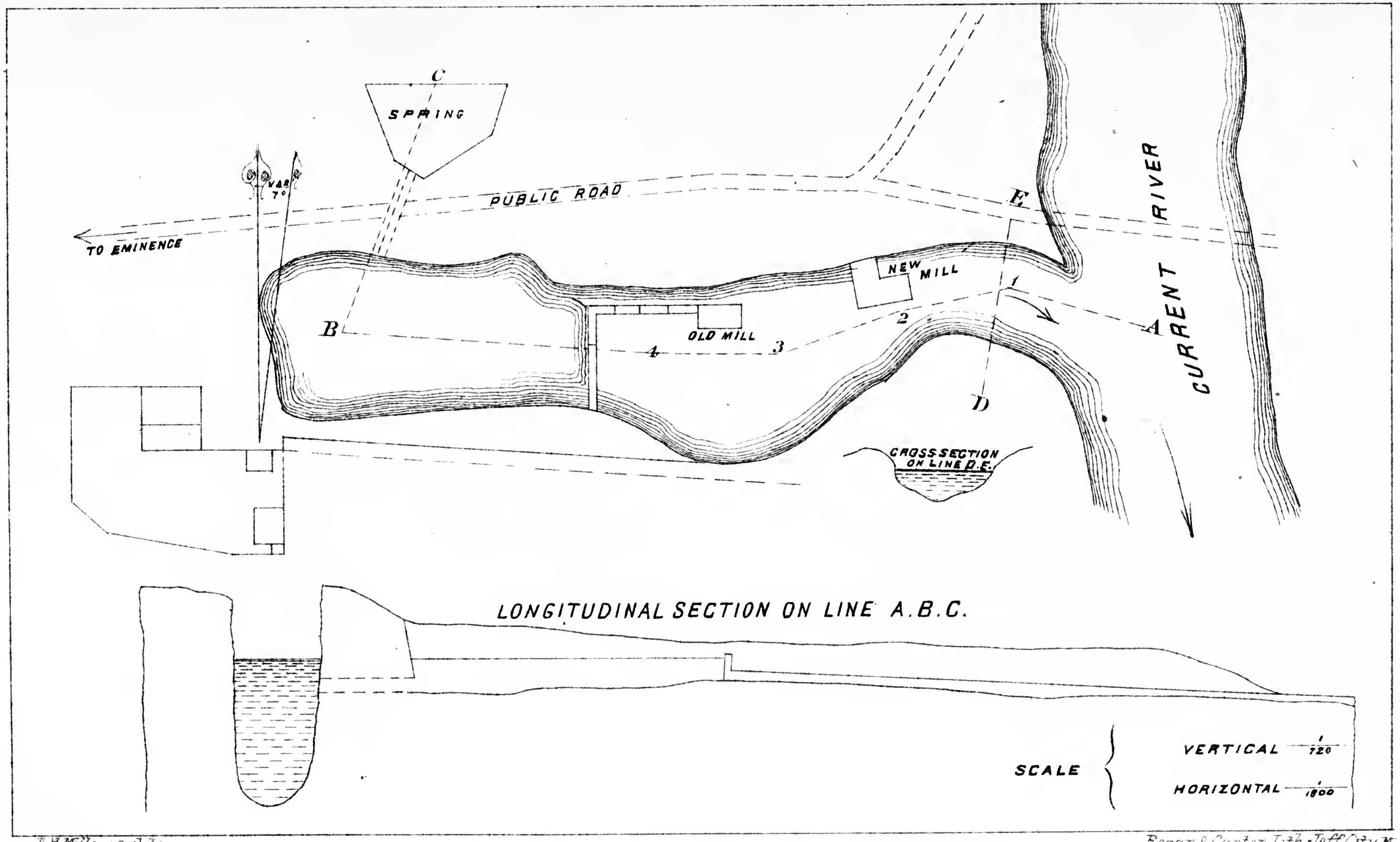
IRONS FROM CHARCOAL FURNACES.

Furnace.	Graphitic Carbon.	Combined Carbon.	Silicon.	Phosphorus.	Sulphur.	Manganese.	Iron.
Maramec No. 1	2.024	2.276	0.378	Trace	0.048	0.904	94.370
Maramec No. 2	2.000	0.887	1.246	Trace	0.136	0.717	95.010
Maramec No. 3	2.662	0.288	1.337	Trace	0.191	0.852	94.770
Ozark	2.6588		0.4199	0 0066	0.044	0.5713	96.2994
Scotia No. 1	Not est.	Not est.	0.826	Trace	0.0256	0.2419	....
Scotia mottled			0.541	Trace	0.043	0.374	....
Scotia No. 3			0,768	Trace	Not est.	0.273	....
Scotia white			1.180	Trace	0.047	0.266	....
Midland	2.780	0.360	0.566	Trace	0.028	0.673	95.593
Moselle No. 1	1.860	1.683	Not est.	Trace	0.150	0.570	....
Moselle No. 2	2.608	0.632	0.459	Trace	0.099	Trace	96.202
Moselle No. 3	1.012	2.604	0.634	Trace	0.065	0.154	95.531
Moselle mottled	0.812	2.021	0.413	Trace	0.177	0.185	96.392
Moselle white	0.346	3.338	1.556	0.039	0.112	0.095	94.514

SHANNON COUNTY

AND ITS

COPPER DEPOSITS.



T. H. Millsaps del.

Regan & Carter Lith Jeff. City Mo.

PLAN AND SECTIONS OF BLUE SPRINGS,

N. W. ¼ of Section 20, Town 30 Range 4 West — SHANNON COUNTY.

Velocity at D. E. = 100 Feet in 29.75 Seconds

PLATE IX.

SHANNON COUNTY.

GEOGRAPHY, TOPOGRAPHY AND HYDROGRAPHY.

Shannon county, situated in the southeastern part of the State, is bounded by the counties of Reynolds and Carter on the east, by Oregon county on the south, by Howell and Texas counties on the west, and by Dent on the north. Its northern limit is township line 32; its southern, the northern line of the lower tier of sections in township 26; its western boundary is in range line 7, while easterly its outline is irregular, but its limits are chiefly to the west of range 3. It includes twenty-six townships, and thirty-four sections, or an area of 980 square miles.

The surface of the county is broken and hilly, the hills rising abruptly or precipitously, and rarely with an approximation to a gentle slope from the streams. On Sec. 28, T. 30, R. 4 W., barometric observations gave the summit a height of 350 feet above the level of Current river, distant less than one-half mile; in sections 9, T. 29, R. 4 W., the elevation is 400 feet above the Current.

The largest stream is the Current river, entering the county in Sec. 5, T. 31, R. 6, and flowing with a very tortuous course, but in a general southeast direction, enters Carter county in Sec. 16, T. 28, R. 1 west. It is a rapid-flowing and clear stream, with a narrow fertile valley, limited by abrupt hills, often with steep mural faces. Its chief confluents are Barren Creek and Jack's Fork of the Current. Barren Creek enters Shannon county from Dent in Sec. 6, T. 31, R. 4 W., and flowing a general southerly course, discharges into the Current on Sec. 7 in the next township south, after receiving, in Sec. 33, T. 31, R. 4, the waters of Sinking Creek. Jack's Fork enters from Texas county (on Sec. 31, T. 28, R. 6,) and with a general northeasterly flow, joins the Current in the S. E. ¼ of Sec. 9, T. 29, R. 3, after receiving the waters of the Delaware and Big Shawnee creeks, both flowing from the South.

Besides the main water courses enumerated, the county is abundantly supplied with smaller streams and brooks, along the narrow bottoms of which excellent farming land is found. All these are quickly flowing streams, and many of them furnish fine power, which must eventually be utilized for the manufacture of lumber from the heavy growths of yellow pine and other woods with which the hills and creek bottoms are clothed. Two of these small streams are of sufficient interest to warrant a special description.

Blue (or Round) Spring, on the northwest quarter of Sec. 20, T. 30, R. 4 west, rises in a circular sink or opening in the limestone rock. The spring has a diameter of 84 feet, and careful soundings reached a maximum depth of 48 feet. The water is of a pale azure blue—possibly due to the polarization of light by the finely divided limestone or clay in suspension—and has a temperature of 58°5 Fah., that of the air being 80½°. It has a subterranean outlet through 72 feet of limestone, after which the waters discharge themselves into Current river, at a distance of 660 feet measured on the course of the stream with a fall of 17 feet. The volume of water discharged by this interesting spring we may estimate from the accompanying plan and sections (from a careful survey in August, 1875,) to be not less than 425 cubic feet per second, or an available horse power of about 800. The origin of this spring must be looked for in some of the subterranean water courses which are known or believed to exist in the vicinity.

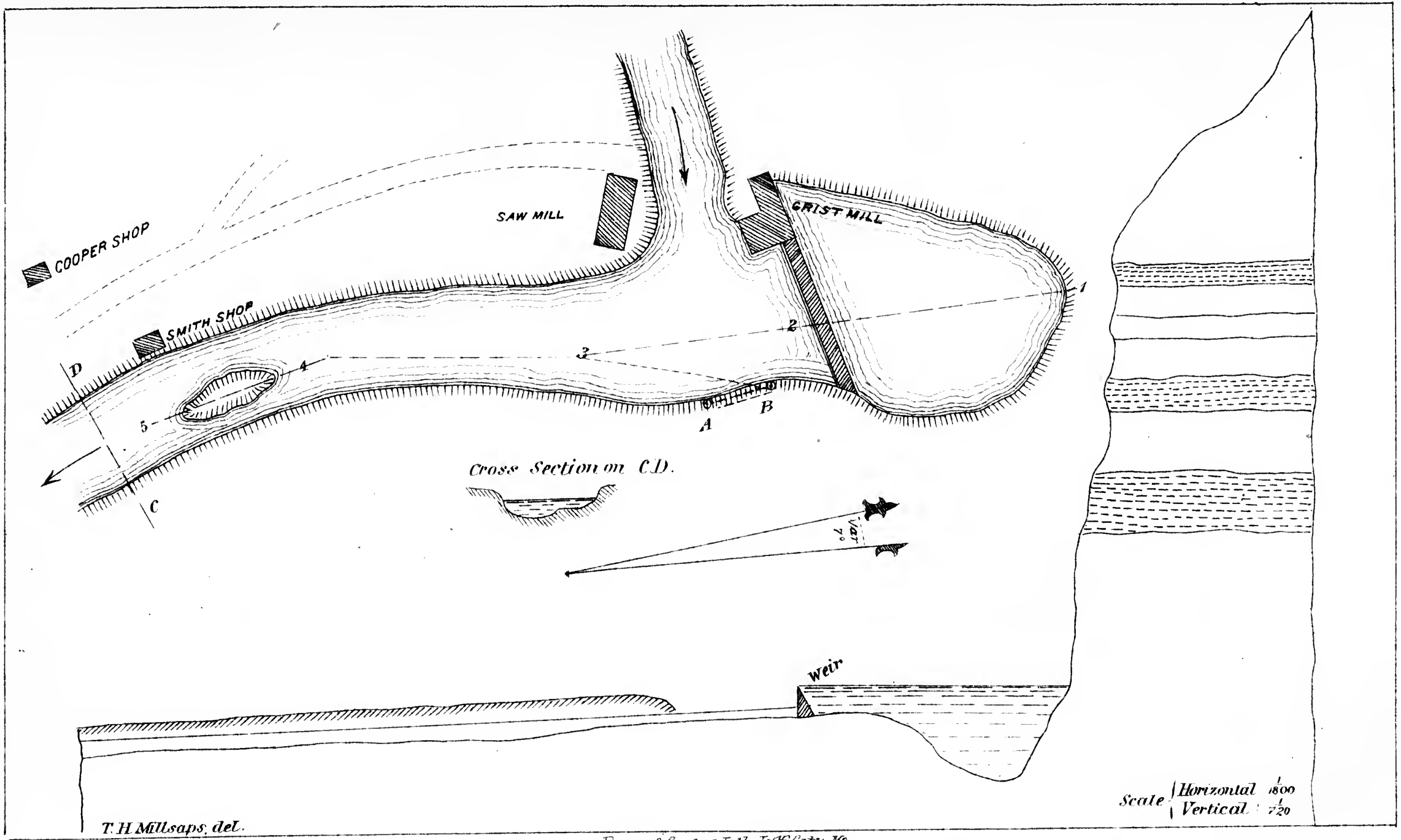
Mammoth Spring (west ½ S. E. ¼, Sec. 25, T. 29, R. 5 W.) appears near the base of an almost perpendicular bluff which has a height of 159 feet (by the barometer) above the present surface of the water in the dam. The bluff rises by a gentle slope, 95 feet more to its summit. The water has a temperature of 57° Fah., that of the air noted at the same time being 80°. It has the blue tint, but not so well marked as that of Blue Spring. A line of soundings from the base of the bluff to the dam gave the following depths: 31 feet, 30 feet, 31 feet, 27 feet, 19 feet, 15 feet and 10 feet. The surface velocity of the stream, determined by the time required for a float to pass an accurately measured distance of one hundred feet, near the line of cross section of the stream (see plan and section of Mammoth Spring, Plate X), was 5.38 feet per second.* The volume of water discharged must be at least 588 cubic feet per second, giving ——— horse power.

GENERAL GEOLOGY.

Archean: This age is represented in the limits of Shannon county by the so-called poryphyry rocks of various textures, gener-

---

* The observations were 18″, 19″, 19″, 19″ and 18″ for the one hundred feet; mean, 18.″6. The observations and the survey of this and the Blue Spring were made chiefly by Mr. John McGuire, a most faithful and competent assistant, who was with me in Shannon and other counties.



T. H. Millsaps, del.

Regan & Carter Lith. Jeff City, Mo.

PLATE X. PLAN AND SECTION OF MAMMOTH SPRING. S. E.¼ OF SEC. 25, TOWN. 29, RANGE 5, W. SHANNON COUNTY.

ally compact, however. They contain little or no free or visible quartz are highly feldspathic and generally micro-crystalline in structure, corresponding in these particulars to the rocks classified as *porphyrite* in the nomenclature proposed by Naumann, the crystals not being large enough or sufficiently well-defined to place these rocks among the true porphyrys, as the word is understood. In fact, the term is applicable only to rocks of certain structural peculiarities, not confined strictly to any special mineralogical composition, and is therefore apt to cause error when used to denote the latter instead of the structural character. However, as the name porphyry has been used to describe certain rocks in Iron county, which are closely related, in physical features at least, to those of Shannon county, and has therefore become somewhat specific in application, it is deemed best to retain it in this connection.

The color of these rocks ranges between a reddish brown and a dark pink; masses of a flesh-tint are occasionally met, and others with a dark liver-color are still more common. They break usually with a smooth fracture, inclining at times to choncoidal.

The absence of distinctly crystalized feldspars in these rocks makes the determination of their mineralogical composition somewhat difficult. A specimen with a dark brown color, showing a few patches or specks of lighter color scattered throughout, but of very hard and compact texture, from N. E. qr., Sec. 36, T. 29, R. 4 W., on being treated with hydrochloric acid yielded material amounting to 12.216 per. cent. of the rock (including 0.651 of loss by ignition, which belonged to this portion,) which, on analysis, was found to consist of:

Silicic acid	37.27 per cent.
Ferric oxide	42,19 per cent.
Alumina	4 29 per cent.
Lime	1.74 per cent.
Magnesia	4.80 per cent.
Soda	1.19 per cent.
Potassa	3.12 per cent.
Water	5.38 per cent.
	99 98

Giving an oxygen ratio for the bases, silica and the water of 4 : 4.5 : 1. It is probably a variety of the uncertain species, pinite, and a product of alteration; but is interesting from the relatively large amounts of ferric oxide and small amount of alumina. Part of the iron may exist, however, as a mechanical mixture, and the decomposition of which this material is the result, may have been aided by the sulphides existing in these rocks.

The portion of the rock remaining unaffected by boiling hydrochloric acid, was of a paler color, with a tint somewhat inclining to purple, and after drying at steam heat, constituted 88.226 per cent. of the original rock. It contained no water, and yielded by analysis:

Silicic acid	73.11 per cent.
Alumina	14 26 per cent.
Ferric oxide	1.08 per cent.
Lime	0.31 per cent.
Magnesia	trace.
Soda	trace.
Potassa	11.08 per cent.
	99.84

The oxygen of the sesquioxides (alumina and ferric oxide) stands to that of the protoxides (potassa and lime) in the ratio of 3.4 to 1, while the ratio in the feldspars is 3 to 1. The mineral is an orthoclase felsite, a portion of the potash having been removed by decomposition. The analytical results point to the view that the original rock was a mixture of albite and orthoclase felsites, the latter preponderating. The decomposition has been most active in the soda feldspar, this being more readily decomposed than the potash or orthoclase feldspar.

The bedding of this rock is obscure, and the differences in lithological features and in texture are so very slight that there are great difficulties in the way of arriving at a knowledge of the structure of the region in question. However, on section 36, there are three parallel intercalations of a white quartzite rock, having a strike of about N. 37° E—S. 37° W., and dipping S. E., 56°—determined in the shafts and other openings on the copper deposits occurring in connection with these quartzite beds. Commencing with the most southeasterly of these, we find it to be about twelve feet thick, and to rest on 18 feet of a decomposed porphyry of a dark brown color, under which is the second belt of white quartzite, resting on 130 feet of porphyry somewhat decomposed (sample for analysis above given was taken from this bed). Below this is the third quartzite bed resting on porphyry, which is of a paler color, but with a purple tint, and is of compact and massive texture. If bedded at all, the bedding is on a large scale.

The white quartzite from bed II, is enfilmed in patches with green stains, probably of malachite, and yields, by analysis, 0.058 per cent. copper (equivalent to 0.104 per cent. malachite). It, and the material from the other quartzite beds, contains, especially in the seams, a white clayey material decomposable by acids, which amounts in the case of No. II, to 15.8 per cent. of the weight of the material, and contains silica, alumina, a trace of ferric oxide, lime, magnesia, soda, carbonic acid and water. Of the remaining 84.2 per cent. of the rock, 96.9 per cent. is silica, the three remaining per cent. being made up of alumina, ferric oxide and lime.

The copper deposits exploited on Sec. 36 are beneath the belts of white quartzite, resting on felsite.

Silurian. The rocks of the calciferous epoch make up the larger part of the county. They rest unconformably on the porphyry, which forms the substructure of the region, covering the latter, excepting at few points, which were islands or peaks above the surface of the waters of the silurian sea. The most recent member is the second sandstone, showing on Secs. 1 and 2, T. 28, R. 5 W., with a thickness of somewhat less than fifty feet; on Sec. 9, T. 29, R. 5, and on the road from Blue Spring to Eminence, on Sec. 9, T. 29, R. 4, with about the same thickness. At the last named locality its height above the Current river is 400 feet. On section 18, T. 30, R. 4, the third magnesian limestone rock forms the entire hill from its base on the Current river to its summit, 350 feet above the river.

The heavy belts of this third limestone approach very closely to a true dolomite in the relations of the calcic and magnesic carbonates, as shown by the following analysis:

	A. per cent.	B. per cent.	Mean. per cent.
Calcic carbonate	55.313	55.216	55.264
Magnesic carbonate	42.532	42.374	42.453
Ferric oxide and alumina	0.964	0.916	0.940
Insoluble matters	1.384	1.330	1.357
	100.193	99.836	100.014

Deducting the insoluble matters and the ferric oxide and alumina (= 2.297 per cent.) and calculating then the percentages of calcic and magnesic carbonates, they will be found as follows:

Calcic carbonate	56.56 per cent.
Magnesic carbonate	43.44 per cent.

whilst the molecular proportions of true dolomite require:

Calcic carbonate	54.35 per cent.
Magnesic carbonate	45.65 per cent.

The magnesic carbonate in the specimen examined is slightly excessive.

ECONOMICAL GEOLOGY.

Copper. The occurrence of deposits of copper ores in the porphyry, and their position on Sec. 36 has already been noted. The existence of these deposits at this locality has been known since the year 1830, when Joseph Slater began mining operations, which resulted in the production of fifteen hundred tons of copper ores, it is said, which were shipped to Swansea or smelted on the location. A period of litigation seems to have followed, lasting until the year 1841, when the Aberdeen (Scotland) Mining Company came in possession and resumed work. Through what cause this company ceased operations I am not advised, but the property came into the possession of the Current River Mining Company, and by them transferred within the last three years to the present owners, the Consolidated Land Company of Missouri, whose estate comprises about nine thousand acres in Shannon county.

About the year 1841 this section of country was visited by Mr. James T. Hodge, and his views were published in Silliman's American Journal [XLIII, 65.] According to Whitney,* who bases his opinion on Hodge's Report, the ores appear to be contact deposits between lower silurian rocks and a reddish quartzose porphyry of azoic age. At the time of my examination (1875), the older works were not in condition to admit of making such an examination as would warrant a positive opinion as to the nature of these deposits, but from the access to them, which was permitted by the cleaning up work of the present company, they seemed to be rather of the character of bedded veins, coincident in strike and dip with the bedding of the porphyry, and associated with the white quartzite above noted, as the hanging wall. It is possible, however, that they may be classed as vertical seggregations or vein masses (*stehende stoecke*, Cotta†) between the felsites proper and the white quartzites. More extended working is necessary before the true character of the deposits can be fixed.

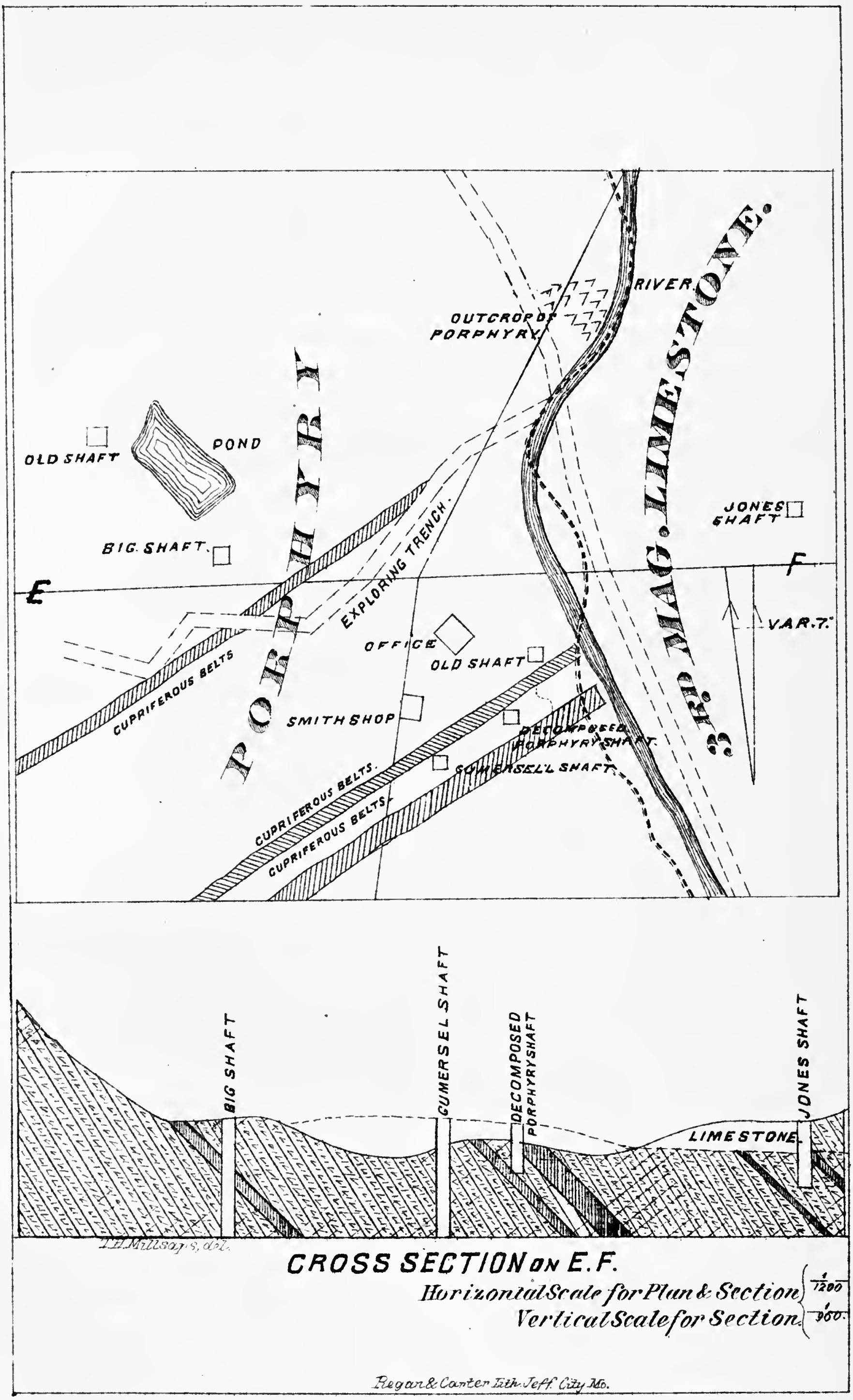
The accompanying diagram (Plate XI) and section of the part of section 36, on which the works have been concentrated, will show the positions of the deposits and the principal shafts which have been carried down to develop them.

The ores in the upper parts of the deposits are almost exclusively oxidized ores, or secondary products from the sulphides. The anogene metamorphoses which have resulted in the production of such considerable amounts of the carbonates and oxides from the sulphur combina-

---

* Metallic Wealth of the United States, 311.

† *Die Lehre von den Erzlagerstaetten,* pp. 191, *et. seq.*



T.H. Millsaps, del.

Regan & Carter Lith. Jeff. City Mo.

PLATE XI. **PLAN AND SECTION OF COPPER PROPERTY.**

N.W.¼ of N.E.¼ Sec. 36. T. 29 R. 4. W.

SHANNON COUNTY.

tions, and in the simultaneous liberation of so much sulphuric acid, must have played no minor part in the alteration of the felsites which has been described. It may be that the beds of white quartz themselves, possibly owe their existence to such action, they being final products from the felsites after the elimination of the basic constituents therefrom. If these hypothetical considerations are well founded, the geognostic history of the copper deposits would be made up of the following periods: (1) formation of a fissure along a line of bedding in the porphyry; (2) repletion of this fissure by sulphides of copper and iron—these being cathogen operations—and, (3) oxidation of the sulphides and formation of the carbonates and oxides, with the simultaneous production of the beds of white quartz and partial decomposition of other beds of felsite.

The gangues are almost entirely silicious; at times small amounts of calcite and dolomite are found, and limonite occurs intermixed with the carbonates of copper, and coating larger lumps of the yellow sulphide.

The following analyses show the composition of the "carbonate ore" from this property:

	I.	II.	III.
Cupric oxide	70.800	70.641	70 598
Ferric oxide	3.463	4.073	3.749
Lime	1.035	0.845	0.804
Magnesia	0.650	0.192	0 232
Carbonic acid	16.640	16.523	16.331
Water	7.050	7.037	7.126
Sulphur	0.876	not est	1 052
Silica	0.363	0.382	0.468
	100.876	....	100.360
Less oxygen equivalent to sulphur	437		526
	100.439		99.834
Metallic copper	56.61	56.52	56.42

The analyses were of a sample of several large pieces. No. I is the mean of two analyses by Ohmann-Dumesnil; II, mean of two by James Pack, and III, single analysis by Gavens—all in this laboratory.

The sulphides at present produced are chiefly from the cleaning up of the old works, no deep or systematic mining having been begun at the time of our visit. The larger lumps are composed of chalcopyrite(=copper pyrites) intermixed with brown iron ore (limonite), a small amount of malachite and more black oxide or melaconite. Two samples were analyzed at my request by Mr. John D. Greason with results, as follows:

	A.	B.
Cupric oxide	28.207	69.001 per cent.
Ferric oxide	45.660	4.049 per cent.
Alumina	nil.	trace.
Lime	0.430	1.881 per cent.
Magnesia	tracc.	0.198 per cent.
Arsenic	nil.	trace.
Antimony	0.070	nil.
Sulphur	13.669	3.908 per cent.
Carbonic acid	1.562	11.906 per cent.
Water	5.462	13.409 per cent.
Silicious matter	5.539	0.259 per cent.
Metallic copper	22.52	55.16 per cent.

The mineralogical composition of A may be calculated from the above analysis to be probably, as follows:

Chalcopyrite	39.165 per cent.
Malachite	2.769 per cent.
Melaconite	12.487 per cent.
Limonite	33.414 per cent.
Gangue	5.969 per cent.

Sample B shows less chalcopyrite, and more malachite.

The first shipment of ore made under the present organization amounted, I am informed, to 36,213 pounds. It was obtained in the work of cleaning up and exploration, and included 546 pounds of furnace bottoms and regulus from the old works. Of the remainder, about one-half was lump or cobbed ore, and the other, sand or wash ore, yielding respectively, 32 per cent. and 12 per cent. It was treated at Wheatleys' works, near Phœnixville, Pennsylvania, the route of shipment being by teaming to Mill Spring station, on the Iron Mountain Railroad, (at a cost of sixty cents per hundred), thence to St. Louis and eastward. These figures are furnished me by Chas. T. Biser, the superintendent of the works, who claims that the Iron Mountain Railroad can be reached, at Newport station, at a cost not exceeding three dollars per ton by flat-boating down Jack's Fork and the Current river.

The limestone near the line of junction with the porphyry shows small amounts of copper, chiefly in the form of malachite stains. These occurences are probably of the class of impregnation zones, and can hardly be regarded as of any practical significance. The Biser shaft (on S. E. qr. of S. E. qr., S. 25, T. 29, R. 4), was carried down to a depth of 70 feet, through limestone, without reaching the porphyry. Coarsely crystallized calcite and dolomite were met, and occasional specks or stains of malachite.

Iron: Deposits of hematite of the variety known as blue specular ore, are found in the sandstone. A specimen from Sec. 2, T. 28, R. 5 W., gave 96.205 per cent. ferric oxide, (=67.34 per cent. metallic iron); 0.893 per cent. alumina; 0.105 per cent. lime; traces of magnesia and phosphoric acid; 0.018 per cent. sulphur, and 2.080 per cent. silica=100.101. Material freed from hygroscopic water, by drying at 102°C. before analysis.

The variety of hematite known as micaceous ore is found in small seams in the porphyry. It is said to be titaniferous, but I have seen no analysis, nor have I deemed the matter of sufficient practical importance to make the examination myself. The little streams running over the porphyry show black iron sand.

Lead: Explorations for lead have been made at several localities, in the magnesian limestone, and in the vicinity of the Blue Springs, on the opposite side of the Current. Some little galenite, largely intermixed with pyrite, was found. Galenite is reported as having been found at other localities in the county, but in no large amount, nor have any deposits, as yet, been opened. The occurrence of the third magnesian limestone—a notable lead-bearing formation—is some foundation for the expectation that lead may be found, but it should be borne in mind that the discovery of the coarsely crystallized calc spar or "glass tiff" is in itself not sufficient incentive to explorations for lead, even in the third magnesian limestone. This mineral is found largely in the limestones of southeastern and southern Missouri, and has been the cause of considerable outlays of time and money, made in confident expectation that it was invariably associated with lead ores.

APPENDIX A.

LEAD AND ZINC STATISTICS.

LEAD STATISTICS OF MISSOURI.

(ST. LOUIS MERCHANTS' EXCHANGE REPORTS.)

Lead Importations to St. Louis for the Twenty Years from 1856 to 1876.

YEAR.	TOTAL POUNDS.	YEAR.	TOTAL POUNDS.
1856	16,372,840	1866	11,966,720
1857	13,004,400	1867	11,564,400
1858	25,311,760	1868	14,865,840
1859	21,150,400	1869	18,264,240
1860	19,059,280	1870	18,963,120
1861	9,220,720	1871	18,369,880
1862	7,664,000	1872	22,861,520
1863	6,385,840	1873	28,482,960
1864	7,442,800	1874	38,357,840
1865	9,330,880	1875	46,336,160
Total			364,975,600

Distribution for Two Years.

Home consumption in 1874	21,536,000 pounds.
Export in 1874	16,821,840 pounds.
Home consumption in 1875	29,600,000 pounds.
Export in 1874	17,736,160 pounds.

Lead received in St. Louis from 1865 to 1876 by following routes:

YEAR.	Upper Mississippi—pounds	Lower Mississippi—pounds	Missouri River—pounds	Iron Mountain R. R.—pounds	Atlantic and Pacific R. R.—pounds	Missouri Pacific R. R.—pounds	St. Louis, Kansas City and Northern—p'ds	Illinois R. R.—pounds	Missouri, Kansas and Texas R. R.—pounds	Other sources—pounds	Total pounds
1865	2,610,000	729,920	....	3,630,560	....	418,560	....	1,596,880	....	....	8,985,020
1866	861,600	1,711,200	....	4,282,640	....	1,758,640	....	2,560,400	....	....	11,174,480
1867	2,643,200	1,417,440	....	4,467,120	....	2,266,640	....	770,000	....	....	11,564,400
1868	2,260,880	2,760,000	....	5,084,560	....	4,194,320	....	1,063,680	....	....	15,360,440
1869	3,349,120	2,236,960	....	8,038,000	....	3,756,320	....	561,360	....	....	17,941,760
1870	3,352,320	340,400	6,320	11,264,808	2,460,880	692,400	556,800	80,320	....	....	18,754,248
1871	2,806,960	124,480	53,440	9,126,160	3,569,680	1,127,600	1,001,920	626,640	....	751,120	19,187,900
1872	1,596,640	....	57,920	8,270,135	4,463,360	6,316,080	1,181,680	16,480	16,480	1,606,640	20,526,415
1873	499,600	....	13,760	9,851,360	7,326,400		4,722,960	....	5,965,680	13,200	28,372,960
1874	132,400	....	102,648	11,734,720	20,076,560		2,602,880	....	....	2,747,600	37,390,808
1875	537,600	....	423,720	16,398,800	27,622,320		1,278,400	....	50,720	2,320	46,313,920
Total	....	....	....	....	....	....	....	....	....	....	235,571,446

Partial Report of Mineral and Pig Lead—1875.

COMPANY.	COUNTY.	MINERAL—POUNDS.	LEAD—POUNDS.
Davis & Murphy	Jasper	2,250,000	3,440,000
Pichar & Bro	Jasper	3,328,960	40,000
Joplin M. and S. Co	Jasper	3,820,066	240,000
Lone Elm M. and S. Co	Jasper	7,142,836	5,980,000
Porter & Dorsey	Jasper	3,464,582	1,500,000
Granby M. and S. Co	Newton	35,606,400	9,900,000
	Dade	500,000	297,797
	Christian	175,000	
	Webster	150,090	
	Greene	50,000	
	Lawrence	4,000	
Palmer	Washington	1,748,430	1,262,222
Long & Bugg	Washington		1,250,000
John Evens	Washington		320,000
Other sources	Washington		293,720
St. Joe	St. Francois		4,960,000
Osage Company	Miller	380,000	374,580
Total			32,954,519

Prices of Lead at St. Louis, for 20 years, from 1856.

[Compiled from various sources.]

1856	$6 22	1866	$10 00
1857	6 00¼	1867	9 00
1858	5 20	1868	9 00
1859	5 25	1869	8 75
1860	5 25	1870	7 25
1861	5 25	1871	7 00
1862	6 50	1872	6 87½
1863	8 62½	1873	6 87½
1864	12 80	1874	....
1865	10 00	1875	6 66⅔

Prices per 100 pounds of Soft Missouri Leads, by Months, for 1875.

January	$6 62½	July	$6 50
February	6 62½ to 6 75	August	6 75
March	6 00 to 6 75	September	6 75 to 6 87½
April	6 25 to 6 50	October	7 00
May	6 25 to 6 37½	November	7 00
June	6 37½ to 6 50	December	7 00 to 7 25

Prices of Hard Lead, for last Quarter of 1875.

October	$6 10 per hundred.
November	6 25 per hundred.
December	6 20 per hundred.

ZINC STATISTICS.

Production of Zinc Ores for 1875.

Newton county	8,500 tons.
Jasper county	4,000 tons.
Dade county	6,500 tons.
Greene county	50 tons.
Washington and other counties	4,500 tons.
Total	23,500 tons.

Consumption of Missouri Zinc Ores for 1875.

Martindale Zinc Company	7,800 tons.
Missouri Zinc Company	4,200 tons.
Carondelet Zinc Company	3,000 tons.
For Oxide production in Missouri	800 tons.
Other furnaces outside the State	7,750 tons.

Production of Spelter in Missouri for 1875.

Martindale Zinc Company	2,200 tons.
Missouri Zinc Company	1,500 tons.
Carondelet Zinc Company	950 tons.
Total	4,650 tons.

Production of Zinc in the United States for 1875.

Lehigh Zinc Company, Pennsylvania	1,505 tons.
Passaic Zinc Works, New Jersey	600 tons.
New Jersey Zinc Company, New Jersey	700 tons.
Bergen Point Zinc Company, New Jersey	500 tons.
Matthiessen & Hegler Zinc Company, Illinois	3,500 tons.
Illinois Zinc Company, Illinois	1,518 tons.
LaSalle Zinc Company, Illinois	1,329 tons.
Robert Langen & Co Zinc Company, Illinois	331 tons.
Martindale Zinc Company, Missouri	2,200 tons.
Missouri Zinc Company, Missouri	1,500 tons.
Carondelet Zinc Company, Missouri	950 tons.
Chicago Zinc Company, Kansas	600 tons.
Reworked Zinc	600 tons.
Total	15,833 tons.

Production of Zinc in 1875, by States.

Pennsylvania	1,505 tons.
New Jersey	1,800 tons.
Illinois	6,678 tons.
Missouri	4,650 tons.
Kansas	600 tons.
Total	15,233 tons.
Reworked	600 tons.
Stock on hand January 1, 1875	135 tons.
Importations during 1875	540 tons.
	16,508 tons.
Stock on hand January 1, 1876	145 tons.
Total consumption for 1875	16,363 tons.

Gold prices per hundred pounds of Common Silesian in New York, on December 31st, of following years:

1866	$6 50	1871	$6 25
1867	6 37½	1872	6 37½
1868	6 25	1873	7 00
1869	6 18¾	1874	6 87½
1870	5 87½	1875	7 18¾

APPENDIX B.

NOTE ON THE OCCURRENCE OF GOLD IN NORTH MISSOURI.

APPENDIX B.

NOTE ON THE OCCURRENCE OF GOLD IN NORTH MISSOURI.

The summer of 1875 witnessed somewhat of an excitement in some of the counties of North Missouri, concerning the occurrence of gold-bearing deposits. The wide-spread notices of the reported discoveries in the neighborhood mentioned were considered of sufficient importance to warrant a reconnoissance of the reputed auriferous region, and accordingly at the request of Governor C. H. Hardin, such was made in October 1874.

The occurrence of gold in the drift of North Missouri in small quantities has been noted by Broadhead,* and the distribution and the character of this drift has been described by the same authority and by other members of the surveys. The wide distribution of the precious metal in small amounts has been long known to chemists, and has become a matter of scientific interest rather than one of any practical significance. Messrs. Dubois and Eckfeldt,† assayers at the U. S. Mint, have described this natural dissemination of gold, and have collected some curious facts, among which may be cited that the clay underneath the city of Philadelphia contains one part by weight of gold in every 1,224,000 or $\frac{7}{10}$ grains (=$0.03) per cubic foot. In citing these results, Genth‡ adds: "Still more astonishing—but unfortunately equally devoid of practical value—would have been the results, if the gravel which *underlies* this auriferous clay, and which is always richer than the clay above it, had been examined for gold."

The occurrence of gold in the materials of drift origin being therefore already admitted, and the distribution of the drift having been described by the geologists of the survey, there remained only the necessity to establish the amount of the gold in the drift at the localities where, among some, the expectations were high that the precious metal might be profitably extracted. The following addressed to his Excellency, Governor C. H. Hardin, under date of November 17, 1875, has already been made public:

The gold is contained in a gravel or drift material foreign to the locality, the underlying rock formation being carboniferous, and unal-

---

* Report, 1873-1874, page 47.

† Proc. Am. *Phil. Soc.* viii, 273.

‡ Preliminary Report Mineralogy of Pennsylvania—4.

tered. This precludes the possibility of gold veins. The gravel is distributed over a wide area, attaining a thickness in the county named [Macon] of upwards of sixty-four feet and consists of sands, marly or calcareous clays, and rounded pebbles or boulders, chiefly of quartzite, granite, porphyry and greenstone, the origin of which was in some locality which in all probability was auriferous. The fact in connection with the wide distribution of the gravel dismisses at once the idea of artificial agencies in connection with the occurrence of gold in the locality.

The large amount of this gold-bearing material, and the circumstances of its origin being understood, it seemed to me that all that remained to be determined were the probabilities of its being economically or profitably worked. To reach this knowledge, steps were taken to establish the quantitative relations of the gold to the containing gravel.

Samples representing seven localities at Murray's Gulch, Sec., T. 60, R. 16, and aggregating seventy-three and one-quarter pounds were collected, and treated carefully by amalgamation. The following are the results:

Amount taken in experiment, 25,600 cubic centimetres weighing 73.25 pounds avoirdupois. Sieved with a sieve of eight holes to the linear inch, giving 8,000 cubic centimetres of coarse pebbles, weighing 32.5 pounds or 44.36 per cent. of the weight of the orignal drift. A sample of the pebbles gave no trace of gold.

The material passing through the sieve was amalgamated and gave .0046 grammes of bullion.

The tailings collected weighed 25.65 pounds (dry) equivalent to 35.02 per cent. by weight. Three assays gave respectively 4-10, 45-100 and 45-100 ounces of bullion per ton of tailings.

A cubic foot of the moist drift as received at the laboratory weighs 97½ pounds.

The bullion obtained was 711—1,000ths fine.

The above data gives the following for the value of one ton of 2,000 pounds of earth:

Gold	58.13 grains or	$2 51
Silver	9.57 grains or	0 03
Total	67.70	$0 03

or at the rate of $3.25 per cubic yard.

Under circumstances where large quantities of earth can be readily handled, with abundance of water and great fall of the water, an amount considerably below this can be profitably worked. Neither of these conditions obtain in the region in question.

The distance of the drift from its place of origin would lead us to expect that the gold would be of the character of small particles of the shape and weight best suited for long suspension in water, a mode of occurrence which would greatly increase the difficulties of extracting by mechanical processes, even were the topographical conditions favorable for such. The results of panning the drift of Macon county make it evident that the gold exists in fine particles, and almost entirely in the form of "scale gold," which it would be difficult, if not impossible, to save by the process of washing.

This is substantiated by a second experiment in which 1.76 cubic feet or 176½ pounds from six localities in Lilly's Gulch (about six miles northwesterly from the locality of the material of the first experiment,) were passed through a "rocker" by an experienced hand, and concentrated up to 12⅓ ounces of fine material. This amount was divided into three unequal parcels, (for convenience in assaying) and an assay gave as follows:

First lot from rocker	0.0018 gramme.
Second lot from rocker	0.0092 gramme.
Third lot from rocker	0.0025 gramme.
Total gold in 12⅓ ounces	0.0135

By calculation this will be found to be equal to 2 $\frac{34}{100}$ grains of bullion per ton of 2,000 pounds of original earth with a value of somewhat less than ten cents, or at the rate of about 13 cents per cubic yard.

If the samples drawn for and used in the assays, were representative of the gravel—and every effort was made to have them such—then the results are not encouraging for profitable extraction of the gold. Under the economic conditions surrounding the deposits, the yield of bullion would not be adequate to cover the expenses of handling and washing or amalgamating, even on the supposition that the bullion can be saved. This latter condition, I believe, as before stated, to be impossible from the form in which the gold occurs in the gravel; for the second experiment (with the material from Lilly's gulch) seems to indicate that in the process of washing by far the largest part of the gold is carried off, even by the slight force of the current of water in an ordinary prospecting rocker.

To the above may be added a few words on the pyrites—numerous samples of which have been brought or sent to the laboratory, on the belief either that the mass of them was gold or that they were gold-bearing to some extent. Of the samples, thirty-eight have been assayed at the request of parties bringing or sending them, and in three only were even traces of gold discovered. The mass of evidence is entirely against the occurrence of auriferous pyrites in Missouri.

INDEX.

ERRATA.

Page 14, line 20, for "analysis" read "analyses"
Page 19, line 23, for "lust" read "lustre."
Page 22, line 6, omit parenthesis after "limonite."
Page 26, line 8, for "being" read "are."
Page 41, line 8, for "*Krumofen*" read "*Krummofen.*"
Page 67, line 6, after "furnace" insert "capacity."
Page 68, line 9, for "Rogers'" read "Rozier."
Page 91, line 2, for "three fining" read "the refining."
Page 93, line 8, after "‡" insert "attributes it."

The last paragraph on page 93, and the tables on pages 94 and 95 should precede the last paragraph on page 92.

Page 109, line 23, for "*Williamite*" read "*Willemite.*"
Page 112, line 34, for "speltre" read "spelter."
Page 152, line 22, after "effected" insert "*by wire drawing.*"
Page 160, line 36, after "giving" insert "660."
Page 160, line 38, for "*Archean*" read "Archæan."
Page 165, line 35, for "Gavens" read "Garvens."