PRODUCTION AND PROPERTIES OF ZINC

A treatise on the occurrence and distribution of zinc ore, the commercial and technical conditions affecting the production of spelter, its chemical and physical properties and uses in the arts, together with a historical and statistical review of the industry

By

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PREFACE.

This book was undertaken originally as the introductory portion of a treatise on the metallurgy of zinc. Upon the completion of the latter, however, it appeared advisable for various reasons to limit it to a consideration of the purely technical questions involved in the extraction of zinc from its ores, or rather the concentrated products which are delivered by the miners for smelting. Although a knowledge of the physical and chemical properties of the metal, the occurrence of its ores, and the conditions which affect the markets is of importance to the smelter, it is equally of importance to the miner and trader. It was thought therefore that a collection of the general commercial and technical data of the metal in a special treatise devoted solely to it would be useful to those who are engaged in the industry in the various ways, and with that in view the present book was prepared upon a considerable enlargement of the original plan.

It has been the aim in this book to present a comprehensive review of the world's resources of zinc ore, the conditions under which it is produced, the concentration of the crude ore into a product suitable for smelting, and a summary of the statistics of the industry from the earliest records down to the present time. The extent into which the subjects of ore supply and mechanical concentration of the crude ore have been entered into is due to their especial interest in this industry. There is perhaps no other important branch of metallurgy in which the exigencies of the smelting process impose such narrow limits as to the grade and character of the ore and make the preliminary preparation of the ore a matter of such great concern as in the metallurgy of zinc.

In concluding this preface, I desire to make acknowledgment to all who have aided me in the preparation of the book, and especially to Mr. E. C. Moxham for notes as to zinc mining in Virginia, and Professor H. O. Hofman for permission to reproduce illustrations from his valuable treatise on the Metallurgy of Lead; also to the American Metal Co. for its statistical publications, and the Robins Conveying Belt Co., the Wetherill Separating Co. and Mr. W. P. Cleveland for illustrations supplied by them.

WALTER RENTON INGALLS.

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I.

HISTORY OF THE ZINC INDUSTRY.

Of the seven common metals zinc was the last to come into use in the arts, and its production on a commercial scale is, indeed, of comparatively recent date. Nevertheless there is evidence, contrary to the general belief, that it was known by the ancients, since bracelets made of it have been found in the ruins of Cameros, which was destroyed 500 B. C.[1] However, this knowledge, which may not have been more than local, seems to have been lost subsequently, for though brass was in use at that time, or a little later, no one appears to have been aware of its exact character.[2] Aristotle in the fourth century B. C. mentions this alloy under the name of Mossinœcian copper, which he describes as having been produced by melting copper with a peculiar earth found on the shores of the Black Sea.[3] This ore was called *καδμεια,* or cadmia, by Dioscorides and Pliny, the latter using the same word to designate the crusts, consisting of impure oxide of zinc, which collected in the brass founders' furnaces.[4] The alloy made with this material was also known as aurichalcum. The alchemists of the middle ages were aware of the effect of this earth upon copper, but came to no more exact understanding concerning it than was possessed by the Romans.

The word zinc appears to have been first used by Basil Valentine, occurring in his *Currus Triumphalis Antimonii* and also in his *Last Testament,* but he did not refer to it especially as a metal. That it was known as such was first mentioned by Paracelsus (1493-1541), who said in his treatise on minerals: "There is another metal called the zinken, which is unknown to

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[1] Raoul Jagnaux, Traité de Chimie Générale (1887), II, 385.

[2] Moses refers to brass in Numbers xxxi, 22, and mention is made of it elsewhere in sacred writings. The manufacture of brass and bronzes seems to have been engaged in by the Phœnicians and Assyrians from a very early period.

[3] Percy quotes the following analysis by Tookey of an undoubtedly genuine Greek coin of Trajan struck in Caria, A. D. 110: Cu, 77·590%; Zn, 20·700%; Sn, 0·386%: Fe, 0·273%, total, 98·949%.

[4] Beyond doubt metallic zinc was often made in the brass furnaces as an accidental product. The φευδάευεος of Strabo is thought to signify drop-zinc. probably formed in that manner.

the fraternity, and is a metal of a very singular kind. It can be melted, for it consists of three fluid principles, but it is not malleable. In its color it is unlike all others and does not grow in the same manner, but with its *ultima materia* I am as yet unacquainted, for it is almost as strange in its properties as *argentum vinum.* It admits no mixture, will not bear the *fabricationes* of other metals, but keeps itself entirely to itself."[1] Consequently, it is unknown to whom is due the honor of its isolation as a metal, but it is probable that the discovery was first made in the East. In the sixteenth century zinc was brought to Europe from China and the East Indies under the name of tutanego (whence the English term tutenegue), and it is likely that knowledge of it was obtained from that source at an earlier date. Agricola recognized in 1550 a metal accidentally produced in the furnaces at Goslar, which he called zink, or conterfey, but he did not know that it came from calamine.

Libavius (1595) was the first to investigate the properties of zinc with any exactness,[2] but he was not aware that it could be derived from the ore known as calamine, which was then used extensively in brass making. The specimens which he examined came through Holland from the East Indies, and he spoke of it as a peculiar kind of tin found there and called calaëm. He described minutely its appearance and general properties, comparing them with those of other metals, and noting especially that when it was heated in the air it took fire and burned. The fact that Libavius was ignorant of the connection between calamine and zinc is at least negative evidence as to the eastern origin of the latter. However, this connection was soon established, since Glauber (1603-1668) mentioned calamine as an ore of zinc, though the exact nature of the metal and its ores continued doubtful throughout the seventeenth century. As late as 1675 Lemery believed that zinc was identical with bismuth, and Boyle often employed indiscriminately the names bismuth, zinc and spiauter, the last being apparently of eastern origin and the first form of the English word spelter.

During this time the manufacture of brass by the cementation process was actively carried on at numerous places, where the industry is so ancient that it is impossible to trace its beginning. It is said that the calamine deposits of Moresnet, the neutral territory between Belgium and Germany, were exploited at intervals from the beginning of the middle ages.[3] In 1435 the

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[1] According to some authorities zinc was first discovered and mentioned by Albertus Magnus, a Dominican monk who lived in the thirteenth century. He called it *marchasita aurea.* The sample of zinc examined by Paracelsus is said to have come from Carinthia, a duchy of Austria.

[2] Roscoe and Schorlemmer, Treatise on Chemistry (New York, 1889), II, i, 251.

[3] According to M. St. Paul de Sinçay (Engineering and Mining Journal, XXXVI, 95) there are ancient documents in Belgium relating that the calamine near Moresnet was worked at the beginning of the seventh

concession of a mine by the Duke of Limbourg is mentioned, and in 1439 there is reference to the mountain of calamine which was worked by the men of Aix. This mine, which was then abandoned, had been worked in former years, but for how long it is unknown. However, the antiquity of the openings was such that the place had received the name of the Vieille Montagne (Altenberg), or the "Old Mountain." In 1454 a grant of the mine was made by Philippe le Bon to Arnold van Zevel, and operations were resumed. The ore was calcined on the spot with charcoal from the Hertogenwald, and was then sent to the brass makers of Aachen (Aix-la-Chapelle), Stolberg, and Cornelius-Munster, afterward to Dinant, Bouvignes, and Oignies, in the neighborhood of Namur, and to Liège and elsewhere in Belgium, the rivers Meuse and Sambre being utilized for its transportation. The process of brass making then employed consisted in imbedding pieces of copper in a mixture of calamine and charcoal in crucibles, which were subjected to a high heat. The zinc was reduced by the carbon, and vaporizing combined with the copper to form brass. The process was slow and wasteful, and it was impossible to produce an alloy with a high percentage of zinc. The manufacture of brass by this method was carried on at Goslar, in the Lower Harz, in the sixteenth century, while the vast deposits of ore in the vicinity of Beuthen, in Upper Silesia, were worked for the same purpose at an equally early date. In England brass works were established in Surrey in the middle of the seventeenth century,[1] and in 1721 as many as 30,000 persons are said to have been engaged in the industry there.

In 1721 Henckel published his discovery that zinc could be obtained from calamine, and he is named by Beckmann as the first who intentionally carried out the process. However, the production of zinc on an industrial scale was first begun in England; it is said that the method applied was Chinese, having been introduced by Doctor Isaac Lawson, who went to China expressly to study it.[2] Be that as it may, a patent for a process of distillation downward was granted to John Champion in 1739, and in 1740 he erected works at Bristol and actually began the manufacture of spelter, but the production was small, and the more part of that used continued to come from India and China.[3] In 1742 Van Swab produced zinc, at Westerwick

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century, but they are of doubtful authenticity, and the best evidence is that it was first utilized during the twelfth century.

[1] A patent for the use of calamine in brass making was granted during the reign of Queen Elizabeth.

[2] L. Knab, Traité de Métallurgie des Métaux autres que le fer (1891), p. 428.

[3] According to Bishop Watson, zinc works were first established at Bristol about 1743. "In about the year 1766 Watson visited Mr. Champion's works near Bristol and saw the process of making zinc, which at that time was kept rigidly secret. Many years afterward he published an accurate description of the process, which is the same as that hereafter described as the English process" (Percy, Metallurgy, I, 521).

in Dalecarlia, by reducing zinc oxide (obtained from blende) with coal, but the process was too costly and was therefore abandoned.[1] In 1746 Margraaf made further attempts, and finally Cronstedt and Rinmann perfected the process of distillation *per ascensum.* None of these early experiments, however, seems to have been of much importance; certainly none of them led to the establishment of a permanent industry, which did not begin until more than 80 years after Henckel's discovery.

The principle upon which the modern process of zinc smelting is based, or rather the method of carrying out that principle in practice, was discovered in Silesia in 1799, or about that time. Contemporaneously a practical method of zinc smelting was devised and put in operation in Carinthia. A similar discovery was made accidentally and independently in Belgium in 1805 by the Abbé Dony, who does not appear to have been acquainted with the contemporary work of others in the same direction. The two processes of zinc smelting, the Silesian and Belgian, in use at the present time have been developed from those beginnings. Some features of the former have been introduced in the latter, and vice versa, but there have been no revolutionary improvements in either, as in the metallurgy of copper and lead, and each remains essentially unchanged. Numerous attempts have been made by zinc metallurgists, notably by Müller, Lencauchez, Clerc, Thum, Köhler, and Hempel, to reduce the cost of producing zinc by distilling in shaft furnaces instead of in the small retorts that have been used for 100 years, but none of those experiments has led to the desired end, because it has been impossible to prevent oxidation of the zinc and the formation of zinc powder in comparatively large quantities, thereby giving products which must be reworked in small retorts and lead to higher metallurgical losses.

Belgium.—Under the Dukes of Limbourg and of Burgundy, as well as under the Spanish dominion, the Vieille Montagne mines were sometimes leased to private parties—never for more than twelve years, however—and sometimes were worked on government account, the last system being followed particularly under the Archduke Albert and later in the time of Philippe IV, King of Spain. After the annexation of the Belgian provinces to France, in 1795, the Republican Government worked the Vieille Montagne for the good of the nation, but under this régime the success of the exploitation fell off, and the Imperial Government renounced it, ceding the mine in 1806 to the Abbé Daniel Dony, a chemist of Liège, "with the obligation of making experiments, which could be recognized as useful, to de-

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[1] Phillips, Elements of Metallurgy (1891), p. 551.

monstrate his ability to reduce, by the aid of proper furnaces, calamine to the metallic state."

The Société de la Vieille Montagne, in a pamphlet entitled *L'Industrie du Zinc,* presented at the Exposition Universelle de 1889, at Paris, from which the foregoing details concerning the history of the zinc deposits of Moresnet have been taken, states that the Emperor thus ordered Dony, in a manner, to discover a method of winning zinc, and Dony obeyed and discovered it, although such discoveries are rarely made by decree. The probability is, however, that Dony, who had been experimenting on the subject since 1780, expressed to the government his ability to produce zinc, and asked for the concession of the mines, which was granted on the condition that he should prove his statements.

Dony's success was finally achieved accidentally, after a long series of disappointing results. The story of the actual discovery is classical. When endeavoring to extract zinc from calamine by fusion in a reverberatory furnace it occurred to him that the heat applied was not sufficiently high, and he added coal dust to the mineral to attain a greater temperature. In order to observe what took place in the interior of his furnace, he built an ordinary flower-pot into one side to serve as a peep-hole through which he could look. The furnace having been charged and the firing begun, drops of zinc were observed to condense on the inside of the flower-pot, which was cooler than the furnace itself, and the method of extracting the metal by distillation was thus discovered.

Dony's experiments were carried out in a little workshop in the Faubourg St. Léonard at Liège, where he subsequently built furnaces and commenced the production of zinc on a commercial scale. There was no use then, however, for the new metal, with which the public was unfamiliar, and it was necessary to raise it to the rank of the needful commodities. This task was beyond the power of one man alone, and Dony, having ruined himself, died without having accomplished it.

Dominique Mosselman, a man of vast energy, took up Dony's work in 1818, devoting his life to perfecting the process for making zinc and to establishing a market for it, but nevertheless at his death, in 1837, the industry could scarcely be said to exist in the west of Europe except in expectation.[1] In order to settle his estate, Mosselman's children founded the Société de la Vieille Montagne, with a capital of 7,000,000 fr., divided into 7,000 shares of 1,000 fr. each, and since that time (1837) said company has been the leading zinc producer of Europe.

[1] These facts are matters of record, but the situation seems strange in view of the development which the zinc industry in Silesia had already attained at that time.

The property of the Vieille Montagne Company, at the time of its organization, consisted of the calamine mine in Moresnet, the St. Léonard zinc works at Liége, two small rolling mills at Hom and Houx, in France, and a new smelting works at Angleur, on the river Ourthe, a branch of the Meuse, which was then in course of construction. A few months later the company bought the rolling mills at Bray, in France, and at Tilff, in Belgium. For nearly 10 years the company had a hard struggle, but after 1846, in which year the direction of its business was intrusted to M. St. Paul de Sinçay, who continued to manage it until 1890,[1] there was a rapid advance in its affairs. The Société Anonyme des Mines et Fonderies de Zinc et de Plomb de la Nouvelle Montagne established works at Engis, in 1845, and other companies engaged in the industry at about the same time.[2]

CARINTHIA.—Zinc smelting was begun at an early date in this duchy of Austria, and by a method of distinct character, the inception of the process and its application being due to Bergrath Dillinger, of Klagenfurt. His first furnace was built in 1799 at Döllach, near Gross Kirchheim, in the Möllthale; according to some authorities this was the earliest furnace for the distillation of zinc in use on the Continent of Europe.[3] In 1800 Dillinger was appointed "director of all the zinc works of Austria," and in 1801 he erected a second plant, at Delach in the Drauthale. Dillinger's furnace, which was wood-fired, comprised 135 vertical retorts, 4·5 in. in diameter at the bottom and 3·5 in. at the top (inside measurements) and 40 in. long. Only 84 of the retorts were charged with ore, each one receiving from 5 to 6 lb.

Metallurgical literature contains but few references to these early Carinthian zinc works and their history is uncertain. The plant at Delach was described by Hollunder, in his *Tagebuch einer metallurgischen Reise,* published at Nürnberg in 1824; a full abstract of his description is to be found in Percy's *Metallurgy.* The distillation in the small vertical retorts was very expensive and the loss of metal was high. Operations at Döllach and Delach appear to have been discontinued in the early part of the nineteenth century. The employment of a modified form of the Carinthian furnace was proposed about 1880 by various metallurgists, among others by Binon and Grandfils and by Chenhall, but their projects did not advance beyond the experimental stage, and subsequent improvements in the execu-

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[1] He was then succeeded by his son, M. Gaston St. Paul de Sinçay, who is the present administrator-general of the company.

[2] The Valentin-Cocq and Flône works of the Société Anonyme de la Vieille Montagne were built in 1851 and 1855 respectively. The dates of the establishment of some of the other zinc smelteries of Belgium are as follows: Ougrée, 1850; Prayon, 1853; Corphalie, 1842; Ampsin, 1841; Antheit, 1842; Seilles, 1857; Boom, 1890; Overpelt, 1893.

[3] V. Spirek, Oest. Zts. (1881), XXIX, 323.

tion of the Belgian process enabled the distillation of ores high in lead, which was the chief advantage the use of vertical retorts was conceived to present.

GREAT BRITAIN.—The first zinc smelting works in England were built in 1740[1] by John Champion, to whom reference has previously been made. He employed the large pots, characteristic of the so-called English process (which survived until about 1860), wherein the vapor is conducted downward through a pipe extending from the bottom of the pot. The first works at Swansea, which is now the chief center of the British zinc industry, was established by the Vivians in 1835.[2] The early English furnaces fell gradually into disuse, being replaced by furnaces of the Silesian type, which have in late years been to a large extent displaced in turn by Belgian furnaces.

SILESIA.—The history of the Silesian zinc industry is similar to that of the Belgian. It began a few years earlier, however, and once having been started grew more rapidly and more regularly, so that it seems strange that the Belgians did not borrow more from the experience of the Silesian metallurgists. The existence of the great deposits of calamine in the neighborhood of Beuthen, in Upper Silesia, was known in the sixteenth century, and even at that time the ore was used for the manufacture of brass by cementation. The zinc mining industry began to assume important proportions in this Province in the eighteenth century, when Georg von Giesche, a merchant of Breslau, received from the Emperor Leopold (Nov. 22, 1704) the privilege of exploiting the zinc deposits of the Beuthen domain and the right of selling the ore outside of Silesia, which grant was prolonged until 1802 by agreement with the Counts Henckel, the great landlords of the district. Toward the end of the century the industry increased largely in importance, the production averaging 10,000 centners of calcined calamine per year between 1780 and 1792 and rising to 18,000 centners in 1792.[3] The cost of production was very small, and the selling price of the mineral was about four marks per centner. The calamine was at first calcined in open heaps, but subsequently calcining furnaces, designed after the form of baking ovens and fired with stone-coal, were introduced.

The zinc industry of Upper Silesia was revolutionized at the beginning of the nineteenth century, when the manufacture of spelter was begun. This was due to the Kammerassessor Johann Ruhberg of Pless, who learned the art in England (where zinc smelting was then being carried on by the

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[1] The authorities differ as to the date.

[2] George Borgnet, Revue Universelle des Mines, Vol. II, November and December, 1877. This paper contains complete data of the economic and technical conditions of the Welsh zinc industry at that time.

[3] The centners referred to in this section are the German centners of 100 lb. =50 kg.

English process of distillation downward) and brought the secret to Upper Silesia. It is incomprehensible indeed that 10 years later there should have been no knowledge in Belgium of what was being done in this branch of metallurgy in England, the two countries being separated by only a narrow strip of water, while the news had previously penetrated eastward to the Polish frontier. Ruhberg constructed the first zinc furnace in Upper Silesia, in the neighborhood of Wessola, utilizing for this purpose the pots in a wood-fired glass-furnace, and it is noteworthy that the Silesian zinc furnace, in contradistinction to the Belgian, preserves to this day traces of the glass-furnaces from which it originated.[1]

After Ruhberg's first experiments others were taken up at the Koeniglichen Friedrichs-Bleihütte and later at the Koenigs-Eisenhütte. In those experiments the round form of the Wessola glass-furnace was still preserved, but more practical retorts for reduction of the ore were adopted, long half cylinders being substituted for the original glass-pots. The experiments with the improved retorts gave such favorable results that the erection of larger zinc works at Koenigshütte was soon undertaken. These works, the Lydogniahütte, came into operation in 1809, and on December 16 of the same year the Bergwerksgesellschaft Georg von Giesche's Erben put in operation a zinc furnace with four muffles at its calcining works at Scharley, the success of which led to the establishment of the Siegismundhütte, with 10 furnaces, at the same place in the following year. In 1813 the Konkordiahütte, also situated at Scharley, was built by the same company. The production of the two works of Giesche's Erben from 1811 to 1814, in spite of the war which was then raging throughout Europe, amounted to 6,279 centners of spelter. Several other small zinc works had also been established in Upper Silesia in the meanwhile, and the increasing production caused a heavy fall in the price of the metal. The consumption of coal in 1815 was from four to five tons per ton of calamine, and the success of the zinc works situated near the zinc mines and remote from the coal, or otherwise unfavorably located, was checked by the cost and difficulty of procuring fuel. This led to the establishment of new zinc works near the collieries, the Giesche's Erben first building the Georgshütte, near the Fanny coal mine at Michalkowitz, in 1818. These works had eight furnaces of eight muffles each, and produced 2,638 centners of zinc in 1818 at a cost of 13·5 marks per centner. The

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[1] The precise date of Ruhberg's first furnace at Wessola is uncertain; some authorities say 1798; others 1799; and others 1800. It is doubtful therefore whether the credit of establishing the first zinc smeltery on the Continent belongs to Ruhberg or to Dillinger. The probability appears to be, however, that Dillinger was the earlier.

two works at Scharley were then closed down. In 1822 the Georgshütte was increased by the erection of eight new furnaces, each with 10 muffles, while two muffles per furnace were added to those previously in use. The production of these works then rose to 8,571 centners and the cost of production was reduced to eight marks per centner.[1]

The number of zinc works in Upper Silesia increased very rapidly after 1815; besides the Georgshütte, the Hugo and Liebehoffnung works were built at Neudorf in 1818 and 1820 respectively, while the Clarahütte was put in operation in 1822 and the construction of the Davidhütte at Chropaczow, with five double furnaces of 20 muffles each, was begun in 1825. The Davidhütte was the forerunner of the present great works at Lipine. The total production of spelter in Upper Silesia in 1825 amounted to more than 200,000 centners and largely exceeded the consumption at that time. Consequently the price of zinc fell to 9@12 marks per centner, and many of the zinc works, whose supply of ore was not assured, were closed down.[2]

The normal Silesian furnace as installed at the Lydogniahütte, Georgshütte, Davidhütte and Liebehoffnungshütte previous to 1820 retained the dome-shape arch of the glass-furnaces and had a plane-grate upward of 6 ft. in length, extending in its longer direction, which divided it into two equal parts. On each side of the grate there were arranged five muffles, about 4 ft. in length and 12 to 15 in. in width and height, inside measurements. The walls of the muffles were much thicker than nowadays, and because of that and the great width of the muffles a fair extraction of zinc was attained only by means of an immoderate consumption of fuel. The first important improvement was made in the '20's, at the Georgshütte, by the Hüttenmeister Knaut, who reduced the width of the muffles more than half and set two in each opening in the sides of the furnace. This innovation was found so advantageous that it was quickly adopted by all of the other smelters. At each corner of the furnace there was a low chimney, through which the gases of combustion were directly discharged. In the Wilhelminehütte, built in 1834, furnaces with 20 muffles were installed and the number of muffles was gradually increased, but the type of furnace remained essentially the same until the '60s, when the furnaces designed by

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[1] In 1874 the cost of smelting (not including general expense) at the Pauls- and Wilhelminehütten was 7·91 marks per centner of 50 kg. in the case of 24-muffle "unterwindöfen" and 5·97 marks in the case of 56-muffle Siemens regenerative furnaces (Max Georgi, Berg-u. Hüttenm. Ztg., March 9, 1877, et seq.). These results were obtained with ore yielding 11·17% and 11·39% Zn respectively. The ore cost per centner of spelter was 10·19 and 10·45 marks respectively, making the total cost of production 18·10 and 16·42 marks.

[2] These data concerning the development of the zinc industry in Upper Silesia are drawn mostly from the excellent book by Doctor H. Voltz, Die Bergwerks- und Hüttenverwaltungen des Oberschlesischen Industrie-Bezirks.

Thometzek and others, having a central chimney and down draught method of heating (imitating the type of furnace installed at Valentin-Cocq, Belgium) came into use. The form of the grate was modified so as to permit of the satisfactory combustion of the small sizes of coal, and the method of firing with an undergrate blast (Unterwindöfen) was introduced. The first Siemens furnace was erected at the Wilhelminehütte in 1869. The old Silesian furnaces were provided with the knee-form, or drop-zinc condensers; these remained in use at the Bobrekhütte as late as 1877. The metallurgical progress in Upper Silesia previous to 1860-1870 was apparently very slow indeed.

Of the great zinc producers of Silesia at the present time the Giesche's Erben engaged in the industry in 1809, as above described; the Counts Henckel von Donnersmarck in 1818; the Schlesische Actiengesellschaft für Bergbau und Zinkhüttenbetrieb in 1853, taking over works previously in operation, of which the Davidhütte, built in 1825, was the first establishment; while the great works of the Herzog von Ujest, the Hohenlohehütte, were not built until 1871.

The technical development of the zinc industry in Upper Silesia is outlined in the history of the Wilhelmine works of the Bergwerksgesellschaft Georg von Giesche's Erben. Those works were built at Schoppinitz in 1834. According to the old records of the company the calamine which was smelted in the early years of their operation assayed from 40 to 45% Zn and yielded about 30% of metal; i.e., out of 400 to 450 kg. of zinc in the ore about 300 kg. were got as spelter, which is approximately as good metallurgical work as is done with modern furnaces, fitted with much better retorts than were formerly obtainable and heated to a much higher temperature.

The explanation of this apparently good work is that in former times the miners were accustomed to give legitimately an overweight of 10% and the smelters not infrequently obtained illegitimately an overweight of as much as 20%. If allowance be made for such errors in the data it would appear that the old Silesian smelters used to experience a loss of 33% of the zinc contents of the ore which they smelted. Considering the conditions under which they worked even such a loss was not an excessive one and the fair extraction of metal was due no doubt to the high grade and docile character of the ore which was available to them. The calamines of that day probably gave an actual yield of about 27% Zn. The calamine which is smelted now assays about 13% Zn and gives a yield of about 10%.[1]

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[1] This refers to calamine alone. The grade of the ore commonly smelted nowadays is raised by admixture of roasted blende.

The value of the calamine produced in Upper Silesia was always more or less dependent upon the price of spelter, but formerly it was not made to correspond therewith so closely as now, and the difference between the value of zinc in the ore and zinc in pigs was generally greater. The ore which is smelted at present was formerly cast aside as worthless, because of its low grade.

In the period about 1835 the consumption of coal in smelting amounted to 250-270 kg. per 100 kg. of ore, according to the records of that date, but taking into consideration the 20% overweight in ore referred to above, the actual consumption of coal was probably about 2·2 times the weight of the ore. This ratio of coal consumption was not reduced to speak of until the introduction of the Siemens furnace. The coal employed in 1835 consisted of about 80% lump coal and 20% slack; the latter was sifted before use and the dust thrown away. The lump coal cost 17·5 pfennigs per centner; the slack cost six pfennigs. Notwithstanding the low cost of each of those kinds *per se* at that time, the average cost was higher than that of the coal used at the present time. On the other hand the cost of the cinder employed as reduction material is now higher than formerly. It was estimated in 1883 that the average cost of heating and reduction coal was five pfennigs per centner higher than in 1835. In 1835, in smelting 100 centners of calamine about 225 centners of coal costing 41 marks were used; in 1883 only 130 centners of coal and cinder, costing 23 marks. An equally striking economy in the consumption of coal per centner of zinc produced has not been attained, on account of the increased poverty of the ore, as a result of which eight centners of coal and cinder, costing 1·6 marks, were required per centner of zinc in 1883, whereas in 1835 the consumption of coal per centner of zinc was only seven centners, costing 1·3 marks.

The cost of labor in smelting in Upper Silesia has also increased. In 1850 wages amounted to 19 marks per 100 centners of calamine; in 1883 they had risen to 26 marks, and that in spite of the introduction of labor-saving improvements. A smelter who was formerly paid one mark per shift got from 3·2 to 3·5 marks in 1883; stokers who received formerly 75 pfennigs were paid 50 years later from 2·25 to 2·5 marks; while the wages of helpers increased from 50 pfennigs to 1 or 1·5 marks. In 1835 the cost of labor per centner of zinc was but little over 0·60 mark; in 1883 it was nearly three times as much. The cost of refractory material increased as well; and also the cost of furnace repairs, partly because of the greater cost of refractory material and partly because of the greater complexity of the improved furnaces. On the other hand the lower cost of ironwork was to some extent an offset in that item. Taking into consideration the cost of material and labor and the consumption thereof, the expense of smelting 100 centners of calamine was about 11% higher in 1883 than in 1835, while the cost per centner of zinc was nearly double. The conditions which existed in Upper Silesia in 1883 were substantially the same as at the present time, but if anything, the cost of coal is now dearer than it was then, and labor also is somewhat higher.

The price of zinc since the metal first became of industrial importance has been subject to curious fluctuations. Unlike most of the other metals of which the value has gradually decreased with improved methods of winning, that of zinc is now but little below the rate of previous years, and is actually higher than the general market price during the first two decades of the commercial history of the metal. This remarkable circumstance is due to the slight demand at first for the new and but little known metal, and the fact that the cost of production in Silesia 80 years ago was less than at present, the increase in the wages of labor and the cost of ore and coal since 1814 having more than kept pace with the saving effected by new and improved mining and metallurgical methods. Previous to 1814 the price of zinc in Upper Silesia appears to have been generally over 30 marks[1] per 50 kg. ($150 per ton), but upon the rapid development of the industry in that year there was a falling off, so that in 1814 the price was 21 marks per centner ($105 per ton), 20 marks ($100) in 1815, and in 1817 and 1818 only 16 marks ($80). In 1820 the price fell further to 10·5 marks ($52·50), which was lower than the average cost of production at that time, wherefore arose the first crisis in the zinc industry, many of the newly established works being obliged to close.

Until 1820 the chief market for Silesian zinc was in Asia, whither it was shipped through Russia; in 1821 exportations to British India were begun, and soon grew to such an extent that the Chinese zinc was driven out of that market. This circumstance, together with the restricted production and the erection of the first rolling mills at Malapane, Friedrichshütte, and Rybnik, thus affording a new home market for spelter, sent the price for the metal up to 32 marks ($160) at the beginning of 1823; but this led to such a great increase in the make that the demand was quickly outstripped, and a second crisis in the industry resulted, which lasted from 1826 to 1830. At times it was impossible to sell zinc at all, and only the most favorably situated works could keep in operation, the production of the Province fall-

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[1] Zinc is quoted in Breslau customarily in German centners (50 kg.), and in recounting the course of Silesian zinc that form will be used, with the equivalent price in dollars per metric ton in parentheses.

In 1809, when the Lydognlahütte was built, the value of zinc was 60 marks per centner.

ing off more than one half its former maximum. The lowest point was reached in 1829, when spelter was quoted at nine marks ($45) at Breslau.

In 1830 the Silesian zinc industry began to develop on a sounder basis. The price for the metal remained low during the next 10 years, but sales became more regular and gradually the demand began again to exceed the production, in consequence of which prices ruled high from 1840 to 1848, varying from 16 to 26 marks ($80@$130). The political disturbances of 1848-1850 then gave the industry a setback, which lasted until 1852, prices ranging from 11 to 13·50 marks ($54@$67·50), but from that time until 1878 there was a steady period of prosperity, which was only interrupted by the financial crises of 1858 and 1873, and the wars of 1866 and 1870-1871. The exhaustion of some of the important deposits of calamine (especially the Scharley and Marie mines) in 1870 led to a decrease in production, and an increase in price owing to the higher cost of ore; but by the end of the decade all the works were equipped with roasting plants for the treatment of the blende, which had first begun to be exploited in 1870. The supply of the sulphide ore being large, the value of zinc in the five years following 1878 averaged five marks, or 25%, below that of the 10 years preceding 1878, and in 1883 was lower than it had been for 30 years. The average price of Silesian zinc per centner and per ton at Breslau by decades since 1830 was as follows: 1830-1840, 13·04 marks ($65·20); 1840-1850, 18·17 marks ($90·85); 1850-1860, 18·42 marks ($92·10); 1860-1870, 18·32 marks ($91·60); 1870-1880, 20·15 marks ($100·75).[1] The history of spelter since 1880 is summarized in Chapter IV.

United States.—Zinc was first made in the United States about 1838, at the Government Arsenal in Washington, from the red zinc ore of New Jersey, for the brass designs of the standard weights and measures ordered by Congress. The process was so expensive, however, as to preclude any idea of producing zinc commercially in the same manner. The regular manufacture of zinc was first undertaken in 1850 at Newark, N. J., by Richard Jones, the ore being charged into Belgian retorts just as it came from the mine. The experiment proved a failure owing to excessive breakage of the retorts due to the high tenor of the ore in iron and manganese. Attention was then directed to recovery of the zinc as oxide, and a furnace constructed of fire brick with a large clay muffle was designed, which withstood the corrosion better than the Belgian retorts. A row of these was erected in connection with a muslin bag apparatus (invented by Samuel T.

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[1] These data concerning the fluctuations in the price of Silesian zinc are taken from a pamphlet published by the Bergwerks Gesellschaft Georg von Giesche's Erben, entitled Denkschrift zur Feier des fünfzigjährigen Bestehens der Wilhelminezinkhütte.

Jones) to collect the fume, and the regular manufacture of zinc oxide was begun. In 1851 Samuel Wetherill, one of the officers of the New Jersey Zinc Co., invented the process, since known as the Wetherill process, by which the extraneously heated muffle was done away with. The ore mixed with anthracite coal was thrown in a layer 3 to 4 in. thick upon a hearth composed of perforated cast-iron plates, 1 in. thick; the door was closed and cold air blown under the grate, which passing through the charge raised the temperature to such a point that the ore in contact with the carbon was reduced to metallic zinc, vaporized and oxidized, passing off as a white smoke to the collecting apparatus, where the products of combustion strained through the muslin, leaving the oxide inside the bags. This process proved so successful that it was introduced immediately and has remained in use without essential change up to the present time. A description of the first works and mines was given in Whitney's *Metallic Wealth of the United States,* published in 1854.

Attempts to produce spelter were not given up, however, and in 1856 experiments with a Silesian furnace were made at Bethlehem, Penn., by the Lehigh Zinc Co. They proved unsuccessful, neither the anthracite fuel nor the retort clay being apparently adapted to the purpose. In 1857 Messrs. Matthiessen and Hegeler, who had just come to the United States from the school of mines at Freiberg, Saxony, obtained permission from the company to experiment on their own account at the abandoned plant. They did it on a small scale, using one muffle placed in a kiln altered for the purpose. They demonstrated that anthracite, as well as New Jersey clay, could be used, and made some spelter in this experimental way, but failed to come to an agreement with the owners of the property for building works, largely on account of the financial crisis prevailing at that time. They then turned their attention to the West, where they studied the zinc deposits of Wisconsin, and late in 1858 began the erection of the present works at Lasalle. Lasalle was selected as the point where the Illinois coal field approached nearest to the Wisconsin zinc mines. The Joplin mines were unknown at that time.[1]

In the meanwhile experiments were still carried on in New Jersey and Pennsylvania by Samuel Wetherill, Joseph Wharton and others, who invented furnaces of various types, but the several undertakings failed, and after all the Lehigh Zinc Co. returned to the Belgian furnace and in 1860 erected works at Bethlehem, Penn. After that date works were built at Newark, Jersey City, and Bergen Point, N. J., and at Friedensville, Penn.

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[1] The first zinc works in the West are said to have been built in the '50s, in Wisconsin, by Georgi, an old Silesian smelter; his undertaking was unsuccessful.

In Missouri zinc was not made until 1867, when small works were erected at Potosi, in Washington County. The Carondelet works were built in 1869. These works were supplied with calamine mined in Southeastern Missouri. The mines of the Joplin district became productive in 1873, their ores being shipped first to the Illinois Zinc Co. at Peru, Ill. A little later zinc works were built at Weir and Pittsburg, Kan., Robert Lanyon, who had been engaged previously in zinc smelting at Lasalle, being one of the pioneers in Kansas.

The zinc industry in the United States has been characterized by rapid development in respect to production. The technical and economical development has been slow on the other hand, and up to a few years ago the same type of distillation furnace was in use as at the time of the inception of the industry in this country. The Wetherill process for the manufacture of zinc oxide directly from ores is the most important contribution that has been made by American engineers to the metallurgy of zinc. Apart from that the most distinctive feature in American practice has been the successful application of mechanical roasting furnaces to the desulphurization of blende. About 1895 the discovery of natural gas at Iola, Kan., and the location of zinc smelteries at that point began a change in the American zinc industry, which has become a radical one. In New Jersey also the successful development of the Wetherill process for the magnetic separation of franklinite and willemite has been of great importance.

II.

PRESENT ECONOMIC CONDITIONS.

The largest zinc-producing countries are Germany, Belgium and the United States, which in 1899 ranked in importance in the order named, each producing upward of 100,000 metric tons of spelter. In Germany and the United States there are in each case two quite distinct zinc-producing districts; in Germany, the eastern, or Upper Silesia, and the western, or Rheinland and Westphalia; in the United States, the eastern and southern, or New Jersey and Virginia, and the western, or Kansas and Missouri, including the works of Illinois and Indiana, which derive their ores from the same source. Considered by districts, the rank in 1900, with the production of each, was as follows: 1, Belgium, 119,317 metric tons; 2, Kansas and Missouri, 104,303;[1] 3, Upper Silesia, 102,093; 4, Rheinland and Westphalia, 53,000; and 5, New Jersey and Virginia, 7,491. Considered more broadly, the production of Upper Silesia ought to be increased by the output of Poland, since the works of that Kingdom reduce ores mined from the same deposits which lie at the boundary between Russia and Germany and extend into each of those empires. Consequently the quantity of Silesian spelter produced is really but little inferior to the output of the works of Belgium.

Looking at the production of zinc ore the largest producer is Germany, which is followed in the order named by the United States, Italy, France, Spain, Sweden, Russia, Algeria, Greece, New South Wales, Austria, Great Britain, Belgium, Tunis, Turkey and Canada. These countries, together with the neutral territory of Moresnet, supply the entire consumption of zinc in the world. Their ores may be divided into two classes: (1) High grade, assaying over 40% Zn before roasting or calcination; and (2) low grade, assaying less than 40% Zn. All the ore of Poland (Russia) and Upper Silesia (Germany) falls into the latter class, the average grade of the output in each of those countries being less than 30% Zn. The ore of Rheinland and West-

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[1] Including a comparatively small quantity of spelter produced from ore mined in Wisconsin, Colorado, Arkansas and Tennessee. The output of the Joplin district alone is still somewhat less than that of Upper Silesia.

phalia (Germany) and the remainder of the countries named is mostly high grade, or rather it is dressed so as to be high grade, ranging from 40% to 63% Zn, though there is very little of it which is higher than 50% Zn outside of the United States. These figures refer to the raw ores before calcination, if they be calamines, or before roasting if they be blendes. With respect to the character of the ore, the larger part of the production of Upper Silesia Austria, Italy, Greece, Spain and France is calamine; the larger part of that of Belgium, Great Britain, Rheinland, Westphalia, Sweden and the United States is blende; that of Poland is entirely calamine; of Canada, blende; while the entire output of New South Wales is mixed blende and galena from the Broken Hill mines. Statistics of the production of zinc and zinc ore are given in Chapter IV.

List of European Zinc Smelters.—Following is a list of all the zinc smelters of Europe who have operated during the last five years, together with the location of their works:

AUSTRIA.

	Smelters.	Location of Works.
1	Trifailer Kohlenwerksgesellschaft	Sagor.
2	Aerischehütte	Cilli.
3	Gräfliche Potockische Berg- u. Hütten Verwaltung	Siersza, near Trzebinia.
4	Verwaltung der Hugo von Loebbeckeschen Zinkhütten	Niedzieliska, near Szczakowa.
5	Erste Böhmische Zinkhütten u. Bergbau Gesellschaft	Merklin, near Pilsen.
6		Trzebinia.

BELGIUM.

	Company.	Location of Works.
1	Société Anon. de la Vieille Montagne	Valentin-Cocq, Hollogne-aux Pierres.
2	" " " " "	Angleur (Chenée).
3	" " " " "	Flône, Hermalle-sous-Huy.
4	G. Dumont et Frères	Sart-de-Seilles, Seilles.
5	Soc. Anon. de la Nouvelle Montagne	Engis.
6	Soc. Anon. Austro-Belge	Corphalie lez Huy.
7	Soc. Anon. metallurgique de Prayon	Prayon, à Forêt.
8	Soc. Anon. metallurgique de Boom	Boom.
9	L. de Laminne	Antheit.
10	Soc. Anon. d'Escombrera-Bleyberg	Bleyberg, à Montzen.
11	Soc. Anon. des metaux d'Overpelt	Overpelt, near Neerpelt.
12	Soc. Anon. des fonderies de Biache St. Waast	Ougrée.

The Société Anonyme de la Vieille Montagne has also a blende roasting plant and sulphuric acid works at Baelen-Wezel. L. de Laminne has a blende roasting plant at Ampsin. Lead smelteries are connected with the zinc works at Bleyberg, Overpelt and Seilles.

FRANCE.

	Company.	Location of Works.
1	Société Anon. de la Vieille Montagne..........	Viviez (Aveyron).
2	Compagnie Royale Asturienne des Mines........	Auby (Nord).
3	Usine à Zinc de St. Amand.....................	St. Amand lez Eaux (Nord).
4	Société des Mines de Malfidano	Noyelles-Godault.
5	M. Dubouchet.....................................	St. Jean de Losne (Côte d'Or).

GERMANY—UPPER SILESIA.

	Company.	Name and Location of Works.
1	Graf Hugo Henckel von Donnersmarck	(Hugohütte) Antonienhütte.
2	" " " " "	(Liebehoffnungshütte) Antonienhütte.
3	Graf Guido Henckel von Donnersmarck	(Lazyhütte) Radzionkau.
4	" " " " "	(Guidottohütte) Chropaczow.
5	Herzog von Ujest.....................	(Hohenlohehütte) Hohenlohehütte.
6	" " "	(Fanny Franzhütte) Bogutschütz.
7	" " "	(Carlshütte) Ruda.
8	" " "	(Theresiahütte) Michalkowitz II.
9	Gräfin Schaffgotsch.....................	(Godullahütte) Morgenroth.
10	Bergwerksgesell. G. v. Giesche's Erben	(Wilhelminehütte) Schoppinitz.
11	" " "	(Bernhardihütte) Rosdzin.
12	" " "	(Normahütte) Normahütte.
13	" " "	(Paulshütte) Klein Dombrowka.
14	O. S. Eisenbahn-Bedarfs. Akt. Gesell..	(Beuthenerhütte) Friedenshütte.
15	" " " " "	(Florahütte) Bobrek.
16	" " " " "	(Rosamundehütte) Morgenroth.
17	H. Roth..................................	(Kunigundehütte) Zawodzie.
18	"	(Clarahütte) Schwarzwald.
19	"	(Franzhütte) Bykowine.
20	Schles. Akt. Gesell. f. Bergbau u. Zinkhüttenbetrieb..................	(Silesiahütte II) Lipine.
21	Schles. Akt. Gesell. f. Bergbau u. Zinkhüttenbetrieb..................	(Silesiahütte III) Lipine.
22	Schles. Akt. Gesell. f. Bergbau u. Zinkhüttenbetrieb..................	(Thurzohütte) Bärenhof-Bykowine.
23	Königl. Preuss. Bergfiscus.............	(Friedrichshütte) Friedrichshütte.

In addition to the above smelteries, the Schlesische Akt. Gesellschaft has a roasting and sulphuric acid plant (Silesia IV) and a sulphurous acid plant (Silesia V) at Lipine, besides a sheet zinc rolling mill at the same place; also rolling mills at Jedlitze, near Malapane, at Ohlau, and at Piela, near Rudzinitz. The Duke of Ujest has a large rolling mill at Hohenlohehütte. The Bergwerksgesellschaft G. von Giesche's Erben has blende roasting and acid works in connection with the Bernhardihütte and also a separate plant known as the Reckehütte. There are acid works at the Lazyhütte and Guidottohütte, and a zinc white factory at Antonienhütte.

GERMANY—RHEINLAND AND WESTPHALIA.

	Company.	Location of Works.
1	Société Anon. de la Vieille Montagne	Berge-Borbeck.
2	Rheinisch-Nassauischen Akt. Gesellschaft	Eschweiler.
3	“ “ “ “	Stolberg.
4	Akt. Gesell. f. Bergbau, Blei. u. Zinkhüttenbetrieb zu Stolberg u. in Westphalen	Dortmund.
5		Stolberg.
6	Markisch-Westfälischen Bergwerksverein	Letmathe.
7	Akt. Gesell. f. Zink Industrie vormals W. Grillo	Hamborn-Neumühl.
8	Aktiengesellschaft Berzelius	Bergisch-Gladbach.

The Société Anonyme de la Vieille Montagne has also a blende roasting plant, a sulphuric acid works and a sheet zinc rolling mill at Oberhausen, Westphalia.

GERMANY—SAXONY.

	Smelter.	Location of Works.
1	Königliche Hütte (Muldnerhütte)	Freiberg.

GREAT BRITAIN.

	Company.	Location of Works.
1	Brunner, Mond & Co.[a]	Winnington.
2	Central Metal and Smelting Co., Ltd	Glasgow.
3	Dillwyn & Co	Swansea (Llansamlet).
4	English Crown Spelter Co	Swansea (Port Tennant).
5	Williams. Foster & Co Pascoe Grenfell & Sons	Swansea.
6	Swansea Vale Spelter Co	Swansea (Llansamlet).
7	Villiers Spelter Co	Swansea (Llansamlet).
8	Vivian & Sons	Morriston, Swansea.
9	John Lysaght, Ltd	Netham, near Bristol.
10	Dynevor Spelter Co	Dynevor, near Neath.
11	H. Kenyon & Co.[b]	Warrington.

[a] Make electrolytic zinc. [b] Recover zinc from galvanizers' waste, etc.

ITALY.

	Company.	Location of Works.
1	Società di Monteponi	Monteponi, Sardinia.

NETHERLANDS.

	Company.	Location of Works.
1	Société de la Campine	Budel.
2	Raôour & Co	Maestricht.
3	Zinkmaatschappy in Limburg	Limburg.

POLAND.

	Company.	Location of Works.
1	Sosnowice Co	Bendzin.
2	Derwiz-Szewcow-Pomeranoff Co	Dombrowa.

SPAIN.

	Company.	Location of Works.
1	Compagnie Royale Asturienne des Mines	Arnao (Asturias).

List of American Zinc Smelters.—In the eastern and southern States of the United States there are works in operation at the present time, or works have recently been in operation, belonging to the following concerns:

	Company.	Location of Works.	Kind of Fuel Used.
1	Bertha Zinc and Mineral Co	Pulaski, Va	Bitum. coal.
2	New Jersey Zinc Co	Newark, N. J	Anthracite.
3	" " " "	Jersey City, N. J	"
4	" " " "	So. Bethlehem, Penn	"
5	" " " "	Palmerton, Penn	"
6	Wythe Lead and Zinc Co	Wytheville, Va	Bitum. coal.

In the western States there are the following, many of which, however, have gone out of operation during the last year or two:

	Company.	Location of Works.	Date of Erection.	Kind of Fuel Used.
7	Cherokee-Lanyon Spelter Co	Pittsburg,Kan. *a*	1878	Bitum. coal.
8	“ “ “	“ “ *a*	1890	“ “
9	“ “ “	“ “ *a*	1891	“ “
10	“ “ “	Weir, Kan. *a*	1873	“ “
11	“ “ “	Nevada, Mo. *a*	1887	“ “
12	“ “ “	Cherokee, Kan. *g*	1891	“ “
13	“ “ “	Iola, Kan. *e*	1899	Nat. gas.
14	Cockerill, A. B	“ “ *e i*	1901	“ “
15	Collinsville Zinc Co	Collinsville, Ill. *a*	1886	Bitum. coal.
16	Edgar Zinc Co	St. Louis, Mo. *b*	1869	“ “
17	“ “	Cherryvale, Kan	1899	Nat. gas.
18	Empire Zinc Co	Joplin, Mo. *j*	1881	Bitum. coal.
19	“ “ “	No. Chicago, Ill. *c*	....	“ “
20	Girard Smelting Co	Girard, Kan	1889	“ “
21	“ “ “	“ “	1894	“ “
22	Illinois Zinc Co	Peru, Ill	....	“ “ *d*
23	Ingalls Zinc Co	Ingalls, Ind	....	Nat. gas.
24	Lanyon Bros.’ Spelter Co	Neodesha, Kan. *k*	1902	“ “
25	Lanyon, S. H., & Bro	Pittsburg, Kan. *a*	1880	Bitum. coal.
26	Lanyon Zinc Co	Pittsburg, Kan. *a*	1882	“ “
27	“ “ “	Iola, Kan	1895	Nat. gas.
28	“ “ “	“ “	1897	“ “
29	“ “ “	La Harpe, Kan	1898	“ “
30	Latourette, James	Marion, Ind	1892	“ “
31	Matthiessen & Hegeler Zinc Co	Lasalle, Ill	1860	Bitum.coal *d*
32	Midland Smelting Co	Bruce, Kan	1899	“ “
33	Mineral Point Zinc Co	Mineral Point, Wis.	....	“ “
34	Nicholson, Geo. E	Nevada, Mo. *i*	1888	“ “
35	“ “	Iola, Kan. *i*	1899	Nat. gas.
36	Prime Western Spelter Co	Iola, Kan. *e*	1899	“ “
37	Rich Hill Mining and Smelting Co.	Rich Hill, Mo	....	Bitum. coal.
38	United Zinc and Chemical Co	Iola, Kan *h*	1901	Nat. gas.
39	United States Zinc Co	Pueblo, Colo. *k*	1902	Bitum. coal.
40	Vulcan Spelter Co	Upland, Ind. *f*	1899	Nat. gas.
41	Wenona Zinc Co	Wenona, Ill	1892	Bitum. coal.

a Idle (1901). *b* Works at Carondelet, a district of St. Louis. *c* Works at Waukegan, now known as North Chicago. *d* Bituminous coal burned in gas producers. *e* Works at Gas City post-office, near Iola. *f* These works were built originally by the Indianola Zinc Co. in 1896; were destroyed by fire and rebuilt in 1899. *g* Works owned by the Cherokee-Lanyon Spelter Co., but leased to and operated by the Cherokee Smelting Co. *h* Works built in 1901 by the Standard Acid Co., which consolidated in 1902 with the Southwestern Chemical Co., of Argentine, Kan., under the title of United Zinc and Chemical Co. *i* Purchased by the Prime Western Spelter Co. in 1902. *j* Idle and partially dismantled. *k* Works in course of construction (1902).

Besides the above list there have been works, now dismantled, at Clinton, Tenn. (Edes, Mixter & Heald Zinc Co.); Philadelphia, Penn. (Delaware Metal Refining Co.); Friedensville, Penn. (Friedensville Zinc Co.); Bergen Point, N. J. (Bergen Point Zinc Co.); St. Louis, Mo. (Missouri Zinc Co.); and at Sandoval, Ill. (Swansea Vale Zinc Co.). Various of the works mentioned in the above list have become idle since it was compiled, besides those which are so indicated.

The New Jersey Zinc Co. was a consolidation (in 1896) of the Lehigh Zinc and Iron Co., Passaic Zinc Co. and New Jersey Zinc and Iron Co.; it controls the Empire Zinc Co. of Joplin, Mo., and North Chicago, Ill., and the Mineral Point Zinc Co. of Mineral Point, Wis. The Cherokee-Lanyon Spelter Co. was a consolidation (in 1896) of the Cherokee Smelting and Refining Co. of Cherokee, Kan., Robert Lanyon & Co., Pittsburg, Kan., and Nevada, Mo., Pittsburg & St. Louis Zinc Co., Pittsburg, Kan., Scammon Zinc Co., Scammon, Kan. (works now dismantled), and Cherokee Zinc Co., Pittsburg, Kan., and Weir City, Kan. The company also leased the works of the Girard Zinc Co. and Kansas Zinc Mining and Smelting Co., at Girard, Kan., which were subsequently returned to the owners and sold by them to the Girard Smelting Co.; the works of the Rich Hill Mining and Smelting Co., at Rich Hill, Mo., were also leased. The Lanyon Zinc Co. was a consolidation (in 1899) of the Robert Lanyon Sons Spelter Co. (works at Iola and La Harpe, Kan.) and W. & J. Lanyon (works at Iola and Pittsburg, Kan.). The Prime Western Spelter Co. was taken over in 1902 by the New Jersey Zinc Co., and the plants of A. B. Cockerill and G. E. Nicholson were purchased by it.

Ore Supply.—The zinc smelters of Europe draw their supply of ore from the various countries on the Continent, which have been mentioned above, and to a comparatively small extent from the United States, Canada and New South Wales.[1] In Upper Silesia and Westphalia the zinc smelters obtain a comparatively small quantity of zinc oxide, which is recovered as a by-product in smelting zinkiferous iron ores. The highest grade of ore which reaches the zinc smelters of Europe is the blende concentrate from Missouri and Kansas and the willemite from New Jersey, which have been shipped to them during the last three or four years. The smelters of the United States use only domestic ores, the supply of which is abundant.

Systems of Zinc Smelting.—There are only two systems of zinc smelting now in use, namely, the Belgian and the Silesian, which are both based on the same general principles, but differ in the types of retorts employed, the furnaces designed to receive them, the grade and character of the ore that can be treated, and the manner of manipulation. The primary difference between the two processes is the use of comparatively small cylindrical retorts arranged in several tiers, each with a slight inclination, in the Bel-

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[1] A considerable quantity of the ore shipped from New South Wales during the last two or three years was reduced at Swansea, Wales, and Ellesmere Port, England, by the Fry process, which was abandoned as unprofitable in 1901.

gian process; and comparatively large muffle-shape retorts (commonly called muffles) set in one tier in the Silesian process. A combination of the two processes, known as the Silesian-Belgian, or Rhenish, is used in some places, chiefly in Rheinland and Westphalia, and to some extent in Belgium and elsewhere.[1] The Belgian process as used in Wales is sometimes called the Welsh-Belgian, although it presents few or no distinctive differences. There were other methods formerly in use, the most important among them being the English, in which the zinc was reduced in pots and distilled downward through a pipe in the bottom, and the Carinthian, in which a series of short vertical pipes formed the retorts, but both these were abandoned long ago.[2]

The Silesian process is the only one employed in Upper Silesia and Poland.[3] The smelters of Spain and the United States use only the Belgian process. Some of the English and Welsh smelters employ the Belgian and some the Silesian process; in some instances furnaces of each type are to be found in the same works. The Silesian-Belgian, or Rhenish, furnace is adopted generally by the smelters of Rhenish Prussia and Westphalia. In Belgium, the characteristic furnace of that country is still the most commonly in use. The Silesian process was applied at an early date at two of the works of the Société Anonyme de la Vieille Montagne (Valentin-Cocq and Flône) but was eventually modified into what has been herein designated as the Rhenish type; similar furnaces are used at Viviez, France, but with gas firing, while at Valentin-Cocq and Flône direct firing is still adhered to. During the last few years there has been a marked tendency on the part of the Belgian smelters toward a form of Siemens furnace, which in some of its features approximates to the Rhenish type.

The practice in the metallurgy of zinc has undergone many changes during the 100 years that the industry has been in existence, but the changes

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[1] In consulting the old hand books some confusion will be found as to the classification of zinc-smelting processes. The old Silesian process, which is now everywhere abandoned, was modified by the introduction of certain features of the Belgian process and the combination is referred to as the Belgian-Silesian by some of the older writers (vide Kerl, Grundriss der Metallhüttenkunde). This is the process now commonly employed in Upper Silesia, which I designate simply as the Silesian. On the other hand some writers call the Silesian modification of the Belgian process, used in Rheinland, Belgian-Silesian.

[2] Although the English furnace is generally described in metallurgical text-books there are not many living metallurgists who ever saw one in operation. According to Doctor Percy they were considered rare even in 1859.

[3] An exception to this statement is to be found, perhaps, in the case of the Hugohütte, at which since 1897 all of the old furnaces have been replaced by Siemens regenerative furnaces with several rows of small muffles, made by hydraulic pressure. At this works a large proportion (up to 70%) of roasted blende is used in the charge. The opinion is expressed by some of the Silesian metallurgists that, in view of the diminishing supply of calamine and the increasing proportion of blende that must be distilled, the Rhenish type of furnaces will eventually replace the Silesian in that Province.

have been generally the result of gradual development and never by the introduction of a radical innovation. So gradual has been the evolution that many methods and types of furnaces have remained quite unchanged through long periods of years. Thus, both in Belgium and Kansas there are furnaces in use at the present time which in design date back 30 years. For the successful distillation of zinc ore we appear to be limited to comparatively small retorts; all attempts to effect the distillation in blast furnaces or otherwise on a large scale, save for the production of zinc oxide, have been failures.

The chief changes and improvements which have occurred in the metallurgy of zinc during the last 40 years have been the following:

1. Replacement of calamine by blende as the most important ore of zinc.
2. Successful introduction of mechanically-raked furnaces for blende roasting, this being especially a feature of the improved American practice.
3. Utilization of the sulphurous gas evolved in blende roasting for the manufacture of sulphuric acid, which is done to a large extent in Germany and Belgium, to a less extent in the United States and France, and recently has been undertaken in Great Britain.[1]
4. Introduction of gas firing, with or without heat recuperation, especially in connection with the distillation furnaces, leading to economy of fuel and better extraction of zinc and permitting the construction of larger furnaces.[2]
5. Manufacture of improved retorts by means of hydraulic pressure, leading to an increased extraction of zinc and ability to smelt more corrosive mixtures of ore.
6. Improvement in the sanitary condition of the furnacemen by proper control of the fumes, etc., originating from the furnaces; this is a matter which has received a good deal of attention in Belgium, Germany and Great Britain, and little or none in the United States.
7. Introduction of labor-saving devices for the handling of material.
8. Utilization of natural gas as fuel in the United States.

The most important recent development in the metallurgy of zinc has taken place in America, and has been not the result of technical study and experience, but the taking advantage of the natural-gas resources of Kansas.

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[1] The importance of blende as a source of sulphuric acid is likely to increase.

[2] The tendency toward very large furnaces has manifested itself most strongly in the United States, where certain smelters have built them with as many as 1,008 retorts. At Angleur, Belgium, there are massives with 400 retorts. At other Belgian works and in Rhenish Prussia and Westphalia furnaces with 240 large retorts are now the standard size. Similarly, in Upper Silesia the new Bernhardihütte has been provided with Siemens furnaces comprising 80 large muffles, the largest furnaces previously in use in that district having had only 72.

However, this has led to such a large reduction in the cost of smelting that the United States has been placed in the position where it can frequently export zinc at a profit. Natural gas is also used as fuel at several smelteries in Indiana, but these are comparatively small works which are less advantageously situated as to ore supply. The utilization of natural gas in Kansas and in Indiana has not led to any radical change in furnace type and operation from those fired with artificial gas elsewhere in the United States, but it is more economical inasmuch as the labor in handling coal to the gas producers and cinder away from them is saved, and up to the present time the cost of the natural gas has been comparatively insignificant.

BELGIUM.—The Belgian zinc industry is centered in the vicinity of Liège, where the method of smelting which bears that name originated. In this district the advantages of excellent coal, superior fire clay and cheap, well-trained labor are combined. The coal is mined near at hand and the clay has to be transported only a short distance. Formerly these works obtained their entire supply of ore from domestic mines, of which the most important were situated at and near the German frontier, but for a good many years their output has been insignificant and most of the Belgian spelter is now smelted from foreign ores, which are imported chiefly from Italy, Spain, Sweden, Algeria, Tunis and Greece, being brought by sea to Antwerp, whence they are shipped by rail or canal to the works. The ocean freights are low, and the cost of carriage from Antwerp to Liège, 5 fr. per 2,204·6 lb., is moderate. The largest producer in Belgium, the Société Anonyme de la Vieille Montagne, obtains a large part of its ore from its own mines, which it operates in many different countries. The blende imported into Belgium is to a considerable extent calcined before shipment from the mines. The Société Anonyme de la Vieille Montagne has, however, at Baelen-Wezel a roasting plant capable of desulphurizing 75,000 metric tons of blende per annum, to which in 1899 a sulphuric acid plant was added. Some of the other Belgian smelters also recover sulphuric acid, but that operation in connection with the roasting is commonly conducted in separate works and the ore is generally delivered to the smelteries in desulphurized condition.

The Belgian zinc-smelting industry which was based originally upon the existence of the domestic supply of easily smeltable ore now rests chiefly upon the great knowledge of the art accumulated from an experience of nearly a century, the possession of a large number of thoroughly trained workmen who have been brought up in the business and the abundant supply of excellent coal which exists in the Kingdom. The location of the most important zinc smelteries of Belgium is shown by the accompanying map.

Coal Resources of Belgium.—The best-developed portions of the Franco-

Belgian coal-field occur within the limits of Belgium, the westerly extension into France being entirely covered by a great thickness of newer strata. Commencing at the eastern side, the first basin is that of Liège, which extends from the Prussian frontier near Verviers in a southwesterly direction for about 45 miles, the greatest breadth being about nine miles, near Liège. The principal working points are concentrated on the western edge. The number of seams is 83. The uppermost series of 31 affords fat coals, suitable for coking; the middle series of 21 seams yields semi-dry or flaming coals; while the lowest series of 31 seams consists of lean, or semi-anthracite coal.

The uppermost series of seams, which are the most valued, is found only in a small area near the center of the basin at Ougrée, near Liège. The seams vary from 6 in. to 5·5 ft. in thickness, the average being barely 3 ft. The same order of succession is observed in all the coal districts along this axis. The strata have a comparatively small slope on the northern outcrop, and are sharply contorted, faulted, or broken on the south side of the basins. The next basin, that of the Sambre, extends for about 30 miles from Namur to Charleroi, the greatest exposed breadth being about 9·5 miles. At Montceau, near Charleroi, there are 73 seams. The most important development of the coal measures in Belgium is in the basin of Mons, which extends from Mons to Thulin, a distance of about 14 miles, with a breadth of about seven

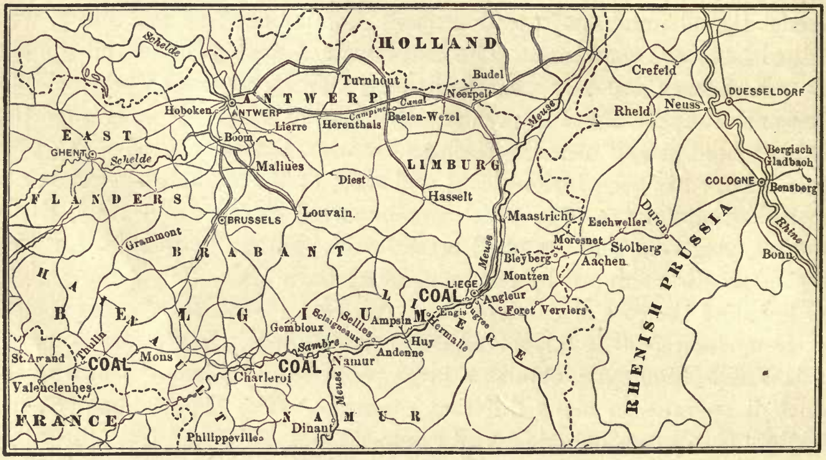

FIG. 1.—MAP OF PORTIONS OF BELGIUM, FRANCE, HOLLAND AND RHENISH PRUSSIA.

Scale: 1 in. = 40 miles.

or eight miles. The number of known coal seams is 157, of which from 117 to 122 are workable, their thickness varying generally between 10 and 28 in., only a few exceeding 3 ft. They are classified into the following groups:

1. Upper series *(charbon flenu)*, 47 seams, which occur chiefly in the neighborhood of Mons: These are the most highly bituminous coals that are found in Belgium.
2. Hard-coal series (*charbon dur*), 21 seams: These are soft, coking coals, less rich in volatile matter than the *flenu*, but excellent for coking.
3. Forge-coal series, 29 seams: These are chiefly used for smithy purposes and ironworks, but the lower members approximate to dry steam coals.
4. Dry or lean coals, 20 to 25 seams, forming the bottom series: They are of small value, being used chiefly for brick and lime burning.

The Belgian coal available to the zinc smelters is only semi-fat in character and direct firing is still maintained to a large extent at the Belgian smelteries. Such Belgian works as have adopted gas firing are obliged to import German coal. The Société Anonyme de la Vieille Montagne has always sought to use at its works the coal most easily obtained and consequently in order to avoid recourse to foreign coal retains direct firing for the more part of its furnaces in Belgium, although it is admitted that gas firing is more advantageous than direct, when suitable coal is available.[1]

Cost of Coal in Belgium.—The cost of Liège coal delivered at the works near Liège in 1893 was 10@11 fr. (approximately \$2@\$2·20) per 1,000 kg. (2,204·6 lb.). Since then and especially during the last two or three years the cost has increased greatly, with the result that the profits of the Belgian zinc smelters have been seriously diminished, notwithstanding the increase in the market price of spelter in Europe during the same period. The Société Anonyme de la Vieille Montagne submitted to its stockholders the following data as to the cost of the coal used at its works in Belgium, France and Germany from 1896 to 1900, both years inclusive.

Year.	Metric Tons.	Cost, Francs.	Average per Ton, Francs.
1896	430,491	4,529,087·49	10·52
1897	443.746	4,954,975·68	11·17
1898	458.074	5,459,272·75	11·92
1899	471,285	6,414,165·81	13·61
1900	494,945	8,469,377·07	17·11

The largest part of the above consumption of coal occurred in the Belgian works of the company. The average cost of the coal used there was as

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[1] M. Gaston de Sinçay, director general of the Société Anonyme de la Vieille Montagne, in The Mineral Industry, VIII, 655.

follows: 1896, 9·96 fr.; 1897, 10·66; 1898, 11·71; 1899, 13·75; 1900, 18·19. At the end of 1900 contracts for a part of the requirements of the company were renewed at 22 fr. per metric ton. Not merely has the price of coal risen, but also its quality has deteriorated, necessitating a larger consumption to do the same work. The increase in the cost of coal in 1900 as compared with 1899 was equivalent to 18·27 fr. per share of the capital stock of the Société Anonyme de la Vieille Montagne, which was not offset by the increased value of the zinc produced, the average realized per metric ton of crude spelter in 1900 having been only 503·2 fr. against 616·5 in 1899.

Cost of Fire Clay.—The fire clay required for the manufacture of retorts, condensers and other refractory material is obtained chiefly from Andennes. In 1893 it cost 12@13 fr. ($2·40@$2·60) per 1,000 kg. delivered at or near Liège.

Cost of Labor.—The average wages for labor at the Belgian zinc smelteries are about 3·5 fr. ($0·70) per day, the maximum, paid to foremen, etc., being 6 fr. ($1·20) per day. The rate of wages has increased greatly during the development of the zinc industry in Belgium. In 1887 the average at Angleur was 3·15 fr. ($0·63); in 1837 is was only 1·35 fr. ($0·27).

Statistics of the Belgian Zinc Industry.—In 1895 there were 10 zinc smelters in Belgium, having 330 furnaces in operation and 58 idle, with an average of 25,609 retorts in operation. These works had 81 steam engines, an aggregate of 2,442 h. p., and employed 4,470 men, who received an average daily wage of 3·51 fr. There were smelted 12,842 metric tons of Belgian ore and 214,947 tons of imported ore, requiring 504,357 tons of coal and producing 96,944 tons of spelter, worth 356·86 fr. per ton. Of the spelter product, 34,081 tons were rolled into sheet at an expense of 13,616 tons of coal and the labor of 477 men, who received an average wage of 3·77 fr. per day. The average value of the sheet zinc produced was 394·41 fr. per ton. The number of zinc works in operation in Belgium in 1897 was nine, there being only one inactive plant. The active works had 359 furnaces with a total of 27,827 retorts. The inactive plant had 68 furnaces. The total number of men employed was 4,820, who received an average daily wage of 3·49 fr. The consumption of coal was 547,666 tons. Of Belgian ore 14,636 tons were smelted; of foreign ore 250,016 tons. The production of spelter was 103,885 tons, valued at 427·41 fr. per ton.[1] In 1898 the quantity of ore smelted in Belgium was 291,977 tons, of which 13,295 was from domestic sources and the remainder was imported.

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[1] British Foreign Office, Report No. 484, Miscellaneous Series, November, 1898. According to the Belgian official statistics the production of spelter in 1897 was 116,067 metric tons, and in 1895 it was 107,664 tons.

Character of Ore Smelted.—The diverse sources of the ore smelted at Belgian works is shown by the statistics for 1898, in which year 59,118 tons were obtained from Sardinia; 48,101 from France; 34,973 from Sweden; 20,076 from Algeria; and 17,552 from Germany. Besides the domestic production, additional supplies of ore were obtained from Greece, Australia, the Italian mainland, England, Tunis, Turkey, Austria and the United States. Analyses of ores received by Belgian smelters from various sources in 1898, which illustrate the nature of their ore supplies, are given in the following table:

ANALYSES OF ORES BOUGHT BY BELGIAN SMELTERS IN 1898.[a]

Kind of Ore.	Source.	% Zn	% Pb	% Fe Mn	% S	%CaO	%MgO	% Cd	% Ag	% SiO_2
Raw blende [b]	Belgium	35·00	tr.	.....	31·00	.....	.....	.....	.....	.....
" "	"	40·00	9·00	12·00	32·00	2·00	0·80	tr.	0·001	3·00
Calcined calamine [c]	Sardinia	54·00	3·00	.....	.....	.....	.....	.....	.....	.....
Roasted blende [d]	"	50·20	5·20	7·70	2·80	0·16	0·42	.....	0·020	15·50
Calcined calamine [e]	France	51·00	9·00	.....	.....	.....	.....	.....	.....	.....
" "	"	53·00	12·00	7·00	.....	1·50	0·80	0·15	0·018	3·00
" "	"	50·55	10·33	8·61	.....	3·30	2·95	.....	.....	2·75
Raw blende [e]	"	49·50	9·00	.....	21·50	.....	.....	.....	.....	.....
" "	"	50·00	12·00	4·00	18·00	2·00	1·00	0·14	0·001	2·50
Roasted blende	"	61·84	11·44	1·74	2·68	0·65	.....	.....	.....	1·00
" "	"	59·80	0·25	2·83	0·93	1·20	.....	.....	.....	15·46
" "	"	60·40	13·40	1·75	?	0·72	0·40	.....	.....	1·50
Raw blende	Sweden	37·00	9·00	10·00	23·00	.....	1·50	tr.	0·007	10·00
Calcined calamine	Spain	41·00	2·73	3·99	.....	5·90	.....	.....	.....	33·97
" "	"	40·00	2·00	7·00	.....	5·00	2·00	0·015	0·006	0·10
Raw blende [f]	"	33·50	4·50	.....	32·00	.....	.....	.....	.....	.....
Crude calamine	Algeria	39·12	8·13	7·00	1·22	0·60	tr.	.....	.....	12·15
Roasted blende	"	56·10	5·50	2·52	2·50	2·44	0·45	.....	0·005	11·00
" "	"	56·88	4·50	4·64	2·20	0·65	0·40	.....	.....	12·65
Calamine [g]	Greece	33·00	tr.	.....	.....	.....	.....	.....	.....	.....
"	"	30·40	1·30	24·29	.....	10·85	1·80	.....	.....	4·00
Roasted blende [h]	"	39·50	8·50	20·72	2·80	2·30	.....	.....	0·010	3·80
" "	Australia	35·10	12·16	13·80	2·43	0·65	0·20	.....	0·040	18·50
" "	"	35·00	8·30	12·60	?	2·42	1·10	.....	.....	20·60
Calamine	Italy	32·00	2·00	6·00	.....	4·00	1·50	.....	0·004	15·00
Raw blende	"	34·00	2·40	2·50	14·50	4·50	0·50	.....	0·004	28·00
Roasted blende	Great Britain	57·00	1·80	7·50	1·80	1·10	0·40	.....	.....	4·00
Calcined calamine	Austria	46·00	6·00	5·00	tr.	4·50	1·50	0·080	tr.	8·00
Raw blende	"	44·00	4·00	8·00	20·00	9·00	1·50	.....	0·001	8·00
Calcined calamine	Not stated	53·00	4·00	3·50	0·50	7·00		0·16	0·012	11·00
" "	" "	46·00	4·50	4·50	0·60	13·00		0·11	0·011	7·00
" " [i]	" "	49·00	6·00	7·00	0·50	9·00		0·20	0·011	7·00
" " [j]	" "	55·00	11·50	5·50	0·80	3·50		0·30	0·020	3·00
Raw blende [k]	" "	35·00	5·00	11·50	30·00	2·00		0·15	0·090	11·50
" " [l]	" "	47·00	4·00	8·50	28·00	2·00		0·35	0·070	6·50
" "	" "	37·00	14·00	6·50	22·00	11·50		0·08	0·011	3·00
" " [m]	" "	35·00	13·00	8·50	23·00	7·50		0·03	0·040	8·00
" " [n]	" "	34·00	5·50	10·00	18·00	1·25		0·08	0·025	17·50
" " [o]	" "	38·00	13·50	7·00	23·00	0·65		0·12	0·050	14·00
" " [p]	" "	24·00	6·00	5·00	19·00	6·00		0·10	0·060	34·00
" " [q]	" "	42·00	8·00	7·00	21·00	0·70		0·17	0·011	13·50
Roasted blende [r]	" "	48·50	7·50	9·20	3·40	4.17	1.10	.....	....	11·80

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[a] Compiled from data in a paper by Ad. Firket in Annales des Mines de Belgique, VI, i and ii.
[b] Gangue. calcareous, silicious and ferruginous.
[c] Gangue, silicious and ferruginous.
[d] This sample also contained 0·40% Sb.
[e] Gangue, silicious and ferruginous.
[f] This sample contained a little $CaCO_3$ and a good deal of iron.
[g] Also contained SiO_2, CaO and much Fe_2O_3.
[h] Also contained 0·4% Cu.
[i] Also contained 0·01% As.
[j] Also contained 0·009% As.
[k] Also contained 0·07% As and 0·2% Sb.
[l] Also contained 0·10% Sb.
[m] Also contained 0·042% As.
[n] Also contained 0·79% As.
[o] Also contained 0·011% As.
[p] Also contained 0·17% Sb.
[q] Also contained 0·02% Sb.
[r] Average of roasted blende bought by a certain works.

The assays of numerous Sardinian calamines received by Belgian smelters in 1898 will be found in Chapter X. M. Firket has computed that the entire quantity of ore reduced in Belgium in 1898 averaged about 45·48% Zn and 4·35% Pb, which figures it should be remembered refer to calcined calamine and roasted blende and not to raw ores.

France.—This Republic is a large producer of zinc ore and of spelter. The industry is rather scattered, however, instead of being concentrated at a few points, and the French zinc ore is largely exported to other countries for reduction, chiefly to Belgium, while the French smelters derive their supply chiefly from Greece, Sardinia and Spain. The most important zinc smelteries in France are those at Auby (Nord) belonging to the Compagnie Royale Asturienne des Mines, and at Viviez, in Aveyron, belonging to the Société Anonyme de la Vieille Montagne. The latter company has also a rolling mill at Panchot in Aveyron. There is a smeltery belonging to the Société de Mines de Malfidano, at Noyelles-Godault, where some of the ore from the Sardinian mines of that company is reduced. A zinc smeltery was also established at St. Amand, Nord, in 1896.

The conditions governing the zinc-smelting industry in the north of France are quite similar to those in Belgium, the Belgian coal fields extending through that portion of France. As in Belgium the coal is only semi-fat and direct firing is still generally maintained. At the works of the Société Anonyme de la Vieille Montagne in Aveyron, in the south of France, however, a coal containing 38% of volatile matter is available and gas firing is employed with considerable economy as compared with direct firing. In France as in Belgium the value of coal has risen greatly during the last five years, the average cost of the consumption at the French works of the Société Anonyme de la Vieille Montagne having been as follows: 1896, 9·62 fr. per metric ton; 1897, 9·99 fr.; 1898, 10·14 fr.; 1899, 10·94 fr.; 1900, 11·63 fr.

Germany.—The zinc smelters of Germany fall into two distinct divisions, namely, the western, or Rhenish, and the eastern, or Silesian. Outside of these there is only one small works at Freiberg, in Saxony, the production of which is insignificant.

Rhenish Prussia.—The eight zinc smelteries of Rhenish Prussia are divided into two groups, three in the Rhine Province, near the Belgian frontier on the west side of the Rhine, and five in Westphalia, east of the Rhine. The former group comprises the works at Eschweiler and Stolberg, near Aachen (Aix-la-Chapelle); the latter includes the works at Berge-Borbeck near Essen, Dortmund, Letmathe, Hamborn, and at Bergisch-Gladbach near Cologne. Practically the same conditions exist in both sections of the Rhenish district.

Coal Resources of Rhenish Prussia and Westphalia.—The coal fields of Rhenish Prussia and Westphalia, situated on the extension of the Franco-Belgian axis, are the two small basins of the Inde and Worm, east of Adelnau, near Stolberg and Eschweiler, and the great Westphalian basin east of the Rhine, in the valley of the Ruhr. The last, which is one of the most important in Europe, extends for about 30 miles east and west from Essen to Dortmund. Its breadth is unknown; the beds are exposed for about 15 miles at the widest part, but the actual boundaries to the north and north-east are hidden by Cretaceous rocks. The greatest depth from the surface to the bottom of the basin is probably about 5,000 ft. It is divided lengthways by transverse axes of elevation into four principal basins, besides several smaller ones. The total thickness of measures so far as proved is from 6,000 to 8,000 ft., with about 130 seams of coal, altogether about 300 ft. thick. These are divided into three series by two bands of barren measures. The thickness of the individual coal seams varies from 8 in. to 7 ft. Seventy-six are considered to be workable, having a combined thickness of 205 ft., and 54 are unworkable, containing 42 ft. of coal. The proportion of workable coal to the whole thickness of strata is as 1 : 33. The order of succession as regards quality is similar to that observed in Belgium, the most highly valued gas and coking coals being at the top of the series, and the dry semi-anthracite seams at the bottom. There is an abundant supply of the fat coal and gas firing, with or without heat recuperation, is now generally adopted by the zinc smelters of Rhenish Prussia and Westphalia.

At Stolberg the coal for heating the furnaces cost about 10 marks ($2·38) per 1,000 kg. in 1893, while reduction material (hard-coal fines from the washeries) was obtainable for about five marks. Since that time the cost of coal in Rheinland and Westphalia has risen materially, especially during the last two or three years, though not to so great an extent as in Belgium. The Société Anonyme de la Vieille Montagne has reported the average cost per metric ton of the coal consumed in its Westphalian works as follows: 1896, 10·13 fr.; 1897, 10·68; 1898, 10·82; 1899, 11·44; 1900, 12·41.

Labor, Ore and Refractory Materials.—The Rhenish works have to import from Belgium the fire clay needed for the manufacture of their retorts. In 1893 it cost them about 12 marks ($3) per metric ton. Wages are about the same as in Belgium, common labor being paid two marks (47·6c.) per day at Stolberg, and furnace men four marks (95·2c.). The Rhenish smelters import a part of their ore supply from foreign countries, via Antwerp and Rotterdam, from which ports they are not far distant, but they obtain the more part from domestic sources, the mines near the Belgian frontier and in Westphalia being still very productive, while considerable quantities are also derived from the Harz, and Friedrichssegen and elsewhere in Nassau. The Grillo works at Hamborn have been lately receiving a considerable quantity of New Jersey willemite, which is used for making a high-grade spelter. The Société Anonyme de la Vieille Montagne operates on a large scale the mines at Bensberg, near Cologne, on the east side of the Rhine, producing blende which is roasted at Oberhausen, where sulphuric acid is manufactured from the roast gases, the desulphurized ore being smelted subsequently at Berge-Borbeck, near Essen. Recently the productive capacity of the Bensberg mines has been largely increased, a new dressing works having been erected there in 1898.

Silesia.—The Silesian zinc district lies entirely within what was once Poland, but is now partitioned among Russia, Austria and Germany, the dividing line between Russia and Germany crossing the zinc-bearing formation. The larger part of the deposit lies, however, in Germany. The center of the district is Beuthen, a small city in the corner of the province of Oberschlesien (Upper Silesia), only a few kilometers from the Russian and Austrian frontiers. Next in importance is the town of Kattowitz. The 23 smelteries of Prussian Silesia are situated near those places. Just over the line in Poland there are works at Bendzin and Dombrowa, and in Austria (Galicia) at Siersza, Niedzieliska and Trzebinia.

Coal Resources of Silesia.—The coal fields of Silesia are the most important of eastern Europe. Those of Lower Silesia and Bohemia form a basin between Glatz, Waldenburg, Landshuet, and Schatzlar, about 38 miles long and 22 miles broad. The number of seams from 3·5 to 5 ft. thick is between 35 and 50, but it is difficult to trace any one continuously for a great distance. The lower seams usually lie at a higher angle than those above them. There does not appear to be any relation between the character of the coals and their geological position, and the same seam often varies in quality in neighboring mines.

The Upper Silesian coal field extends in several disconnected areas from Maehrisch-Ostrau in Moravia, in a northwesterly direction, by Rybnik and Gleiwitz in Prussia, and Myslowitz in Poland, lying partly in Austria, Prussia and Russia. The Prussian portion between Zrabze and Myslowitz is the most important, extending over 20 miles in length, by nearly 15 in breadth. The greatest thickness of coal in workable seams (which vary from 2·5 to 60 ft. in thickness) is estimated at a total of 333 ft., the thickness of the measures being about 10,000 ft.

The great coal measures of Upper Silesia exist in close proximity to the zinc mines, as may be seen from the map on page 215, and the smelters are favored consequently not only by comparatively cheap coal of very good quality, but also by low cost of carriage of the ore to it, which is of more importance in this district than elsewhere on account of the low grade of the ore. The best coal in 1893 cost about eight marks ($1·92) per 1,000 kg. at the smelteries, from which figure the price ranged downward to about 2@4 marks for the inferior grades which are used to a large extent. Since 1898 the Silesian coal has appreciated a good deal in value, but not in so great a proportion as has been experienced in Belgium. According to the statistics of the Oberschlesischen Berg- und Hüttenmännischen Vereins the average value per 1,000 kg. of the entire production of coal in Upper Silesia at the mines has been as follows:

Year	Marks.	Year	Marks	Year	Marks	Year	Marks
1886	3·688	1890	4·800	1894	5·228	1898	5·585
1887	3·550	1891	5·415	1895	5·197	1899	6·005
1888	3·550	1892	5·437	1896	5·216	1900	7·133
1889	3·730	1893	5·371	1897	5·319	1901	....

These figures represent the average of all kinds of coal produced—i.e., they may be assumed as indicating the value of the run of the mine. The Silesian zinc smelters use chiefly small sizes and slack for heating their furnaces. The relative value of the different sizes of coal in 1891 was as follows: No. 1, 7@8·5 marks; No. 2 (about nut size), 5@5·5 marks; Grieskohl, 3@4 marks; dust-coal (under 10 mm. size), 1·6@2 marks.

Labor, Ore and Refractory Material.—Refractory material costs considerably more in Silesia than in either Rhenish Prussia or Belgium; however, the consumption is less per ton of ore smelted. The best fire clay is imported from Briesen, in Moravia (Austria), and costs 30 marks ($7·20) per 1,000 kg. (2,204·6 lb.); other Moravian clay is obtained for 22 marks ($5·28); while ordinary Silesian clay is worth 15 marks ($3·60). The wages for labor are about the same as in the west of Germany, the best class receiving 3@4 marks ($0·72@$0·96) per day, but a larger proportion of cheaper labor is used, women who are paid only 0·90@1·20 marks ($0·22@$0·29) per 12 hours being employed for much of the common work, such as dressing the ore and sorting old refractory material. The employment of women in the smelting houses has been recently discontinued, however, in some of the largest plants of the Province, namely those of the Duke of Ujest. The working population of Upper Silesia is mostly (90%) Polish.

The ore supply of the Silesian smelters is obtained chiefly from the local mines and consists of both calamine and blende. The Polish works use only calamine, which is very low in grade. In Upper Silesia the proportion of blende to calamine is steadily increasing. The Silesian smelters are also receiving considerable quantities of blende (roasted) from Sweden and Styria (Austria). The Silesian ores generally contain lead, and the spelter derived from them contains so much of that impurity that it has to be subjected to a refining process. Numerous analyses of the ores smelted in Upper Silesia will be found in Chapter X.

GREAT BRITAIN.—The chief center of the British smelting industry is at Swansea, in Wales, where an abundant supply of extremely high-grade coal is available and a considerable quantity of ore is obtainable from mines in Wales, which are described in Chapter X. These mines formed the basis of the original Welsh smelting industry, but their production has for a long time been inadequate to meet the requirements of the Swansea smelters, who have therefore had to draw their supplies from other countries, chiefly Italy and Spain, and lately to some extent from the United States. Swansea being an important seaport, the zinc smelters at that place are situated favorably with respect to obtaining ore from all countries whence it would naturally be shipped by sea.

Coal and Fire Clay.—The coal used for heating the furnaces cost the Welsh smelters 7s. 6d. per ton of 2,240 lb. in the midsummer of 1899; anthracite fine coal for reduction material cost 4s. per ton. The best Stourbridge fire clay cost 60s. per ton. These prices for coal and fire clay represented the cost of those materials delivered at the works. It will be seen that the cost of coal to the Welsh smelters is less than to the Belgian and Rhenish smelters, but is rather high as compared with the cost to American smelters; it must be pointed out, however, that the Welsh coal is far superior in heating value to any bituminous coal which is used by American zinc smelters, and there is probably no great difference between Welsh coal at 7s. 6d. per 2,240 lb. and Kansas, Missouri, or Illinois coal at $1 per 2,000 lb.

Wages.—The wages of labor at Swansea are low as compared with American conditions and high as compared with the Continental. Common labor is paid from 3s.@3s. 6d. per day. Furnacemen are paid as follows: First hand 6s., second hand 5s., third hand 4s., all per shift of 12 hours.

Furnaces.—The Welsh smelters formerly employed chiefly the Silesian type of distillation furnace, which is still used extensively in the district. Some of the works use the Welsh-Belgian furnace with roasting furnaces attached thereto, but the employment of this type is not spreading.[1] In recent years there has been a tendency to replace the Silesian furnaces with the ordinary Belgian. Direct firing has almost entirely given way to gas firing,

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[1] William Blackmore, private communication, August 12, 1899; I am also indebted to Mr. Blackmore for the data as to the cost of coal and wages of labor in Wales.

in connection with which the tendency has been lately to abandon the more complicated forms of brick-filled regenerative chambers in favor of simpler heat-recuperative flues, the common checker-work type of regenerator becoming too rapidly choked by the escaping zinc fumes.[1] According to R. Forbes Carpenter, the British Alkali Inspector, in his report for 1896, "the English zinc works are substituting for the Belgian furnace, in the case of certain ores, a furnace heated by producer gas with two or three tiers of muffles. Arrangements have also been made in many English zinc works to remove the fumes of zinc oxide escaping from the condensers, this being done chiefly to improve the health of the workmen." Certain English zinc works have begun the manufacture of sulphuric acid from the waste fumes of the blende-roasting furnaces.

GREECE.—The Kingdom of Greece used to furnish an important supply of zinc ore to European smelters, it being derived entirely from the mines of Laurium, but the output from that source has lately diminished materially. None of the ore is smelted in Greece, but most of the calamine is calcined and the blende is roasted before exportation. M. Ed. Fuchs[2] states that the cost of mining zinc ore at Laurium is 4·70@7·20 fr. per metric ton, according to whether the exploitation is carried on open cast or underground and at depth, labor amounting to 2·50@5 fr. and powder and supplies to 2·20. The cost of production of sorted ore varies of course according to the tenor of mineral in the crude material mined. A laborer paid 3 fr. per day is able to sort three tons per day, making two classes—mineral and waste. A ton of mineral sorted from stuff containing respectively 75%, 50% and 25%[3] will cost:

	Francs	Francs	Francs
For mining	8·91	13·40	26·80
For sorting	1·33	2·00	4·00
Total	10·24	15·40	30·80

In the first case 1·33 tons of ore must be mined and sorted to get one ton of mineral; in the second, two tons; and in the third, four tons.

The cost of calcination is 3·80 fr. per ton of product. Assuming that one ton of raw mineral will yield 0·75 ton of calcined, whence one ton calcined =1·33 ton of raw, the total cost of delivery of one metric ton of calcined

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[1] Professor W. C. Roberts-Austen, in the official catalogue of the British section at the Chicago Exposition.

[2] Traité des Gîtes Minéraux, II, 385.

[3] These figures refer to the percentages of "mineral" in the crude ore.

calamine from Laurium ex ship at Antwerp, varying according to the percentage of mineral in the crude ore raised from the mines, is as follows:

	Francs	Francs	Francs
Mining and sorting...........	13·40	20·49	40·96
Calcination..................	3·80	3·80	3·80
Transportation to port........	7·00	7·00	7·00
Storage and loading..........	4·33	4·33	4·33
General expense.....	1·50	1·50	1 50
Transportation to Antwerp....	18·0C	18·00	18·00
Total....................	48·13	55·12	75·59

Italy.—The zinc ore produced on the Italian mainland is entirely exported, largely to Swansea, Wales, and to Belgium. Until recently the important production of Sardinia has also been entirely exported, going chiefly to Belgium, France and Wales, but within a year or two the Società di Monteponi has established a smelting department in connection with its other works at Monteponi and is reducing therein a small portion of its own output of ore. The fuel employed is lignite.

Netherlands.—Comparatively recently a somewhat important zinc-smelting industry has been established in the Netherlands, where there are three works, of which one has been in steady operation for several years. The Netherlands produce no zinc ore and the supply of the Dutch smelters is obtained from foreign countries, being imported through Rotterdam and Antwerp. The smelteries are situated in that portion of Holland which is in close proximity to Liège, Belgium, and the conditions of smelting are quite similar to those existing in the Liège district.

Russia.—The production of zinc in Russia is confined to the Kingdom of Poland, where the industry has existed since 1816. It is at present in a fair state of development, the annual production of spelter amounting to about 6,000 metric tons. There are two works, one at Bendzin, belonging to the Sosnowice Co., and the other at Dombrowa, belonging to the Derwiz-Szewcow-Pomeranoff Co. There are also two rolling mills in the Kingdom, one at Sosnowice and the other at Slawkow, besides a factory for the manufacture of zinc oxide. The ore is mined in the neighborhood of the town of Olkusz and is transported by carts or by the Ivangrod-Dombrowa Railway to the Bendzin and Dombrowa works. The Polish zinc ore is identical in character with what was mined in Upper Silesia in the early days of the zinc industry there; at present only calamine is treated. The furnaces employed and the method of smelting are the same as in Upper Silesia. Coal is obtained from the Dombrowa basin, a continuation of the Upper Silesian coal measures.

SPAIN.—The large output of zinc ore in Spain is chiefly exported, by far the most part going to France and Belgium. The calamine is generally calcined before shipment. A comparatively small quantity of the Spanish ore is smelted by the Compagnie Royale Asturienne des Mines at its works at Arnao, in Asturias, at which the Belgian method of smelting is employed. There is a rolling mill connected with those works, in which a part of their product is converted into sheet zinc.

The total production of zinc ore in Spain in 1899 amounted to 119,770 metric tons, of which the province of Murcia furnished 56,499 tons, Santander 43,825 tons, Cordoba 6,795 tons, Granada, Almeria and Guipuzcoa upward of 2,000 tons and Teruel 1,520 tons. The exportation of blende in 1899 amounted to 63,438 tons, of which upward of 50,000 tons were despatched to Belgium, 6,631 tons going to France and small quantities to Holland and Great Britain. The exportation of calamine amounted to 31,649 tons, of which upward of 15,000 tons went to France and Belgium and small quantities of a few hundred tons to Great Britain and Holland. The principal Spanish ports for the shipment of zinc ore are Cartagena, Santander, Seville, Almeria and Malaga.

UNITED STATES.—The zinc-smelting industry of the United States is characterized by great variations in practice depending upon the local conditions of the different districts. Fortunately all the zinc ore producing districts of the United States occur in close proximity to fuel, which favors greatly the metallurgical industry, but the character of the fuel as well as the character of the ore varies widely. The geological conditions under which the zinc ore of the United States is found are described in Chapter IX, to which reference should be made.

Eastern Districts.—The ore mined in New Jersey and Pennsylvania is reduced at works situated at Jersey City and Newark, N. J., and at South Bethlehem and Palmerton (near Hazard), Penn., all of which are controlled by the New Jersey Zinc Co. Their production of spelter is of much less importance than their production of zinc oxide. The New Jersey zinc ore as mined is not well adapted to the production of spelter, because of its high percentage of franklinite. A mechanical separation of the willemite and franklinite was not effected in an entirely successful manner until the invention of the Wetherill magnetic machines (vide Chapter XI). By means of those machines a clean willemite product is now obtained, which is an excellent ore for reduction to spelter, furnishing a metal of exceptional purity. So far it has not been the policy of the New Jersey Zinc Co. to make a large production of spelter, it being preferred evidently to make only a comparatively small quantity, which can be sold at a premium, and export the remainder of its willemite to European smelters. The difference between the market prices of the two kinds of spelter is variable, but in general it is in the neighborhood of 2c. per lb. In August, 1898, when prime Western spelter was obtainable in New York at 4·5c. per lb., the New Jersey high-grade spelter was fetching 6·75c. High-grade spelter is also produced by the Bertha Zinc and Mineral Co. at Pulaski, Va.

Both at Palmerton, Penn., which is the location of the chief works of the New Jersey Zinc Co., and at Pulaski, Va., the conditions are favorable to economical zinc smelting. At the former anthracite coal is obtainable cheaply from the near-by mines. The Bertha Zinc and Mineral Co. uses for heating its furnaces run of mine coal from the Pocahontas Flat Top region, which is 75 miles from Pulaski. This coal assays 74·27% fixed carbon, 18·79% volatile matter, and 6·94% ash. Reduction coal is obtained from Altoona, 10 miles from Pulaski, where there are seams of great thickness, having a minimum of 20 ft. and a maximum at one point of 117 ft. This coal, which is anthracite in character, has the composition: 62·72% fixed carbon, 10·52% volatile matter, 1·43% sulphur, and 25·33% ash. Labor is obtainable cheaply at all the works in New Jersey, Pennsylvania and Virginia. Refractory material is obtained from Woodbridge, N. J., and Cheltenham, St. Louis, Mo.

Western Districts.—The most important producers of spelter in the United States are situated in the West, at various points where the general conditions are somewhat similar. There are numerous smelteries at Pittsburg, Kan., and vicinity, which enjoy cheap coal and proximity to the ore supply, but are antiquated in design and uneconomical in operation.[1] There is a group of newer works at Iola and Cherryvale, Kan., which are a little less favorably situated as to ore and coal, but enjoy what is at present practically free fuel (natural gas) together with an important saving in labor.[2]

In the vicinity of St. Louis, namely at Carondelet, Mo., and Collinsville, Ill., are two smelters which have cheap coal (though of inferior quality), comparatively low freight on ore (taking into consideration the cost of getting the spelter to market), cheap fire clay and good metallurgical practice.

The three groups above mentioned derive their ore supply entirely from the Joplin district, with the exception of a little which is obtained from Arkansas and Colorado. The quantity of ore derived from Colorado has lately been increasing considerably, and this source of supply promises soon to become an important factor in the domestic zinc industry.

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[1] The smelters at Pittsburg, Kan., were unable to withstand the competition of the natural-gas smelters and their works are now idle (1901).

[2] Largely because of the less volume of material—fuel and its ash—which has to be handled. The ability to construct larger furnaces has also led to economies.

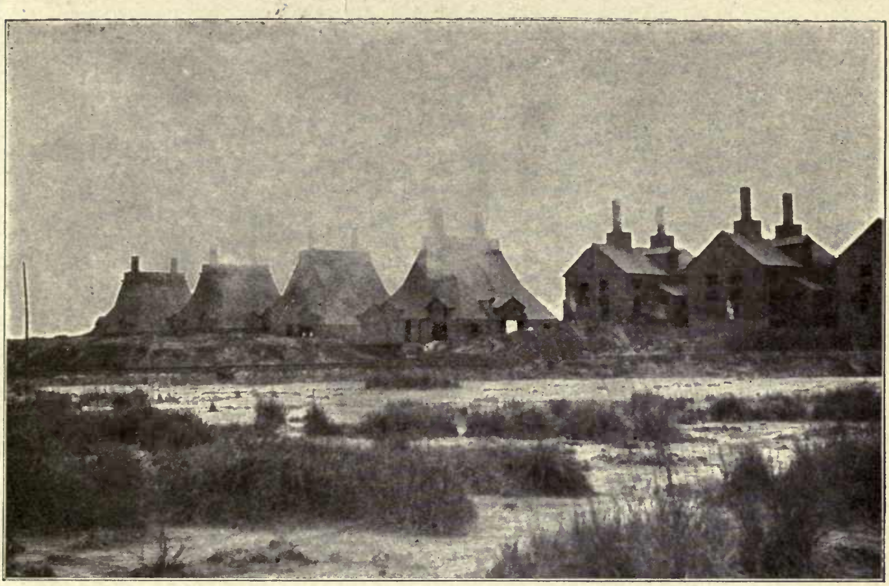

FIG. 2.—ZINC SMELTERY AT PITTSBURG, KAN.

The distillation furnaces are shown at the left and the roasting furnaces at the right of the plate.

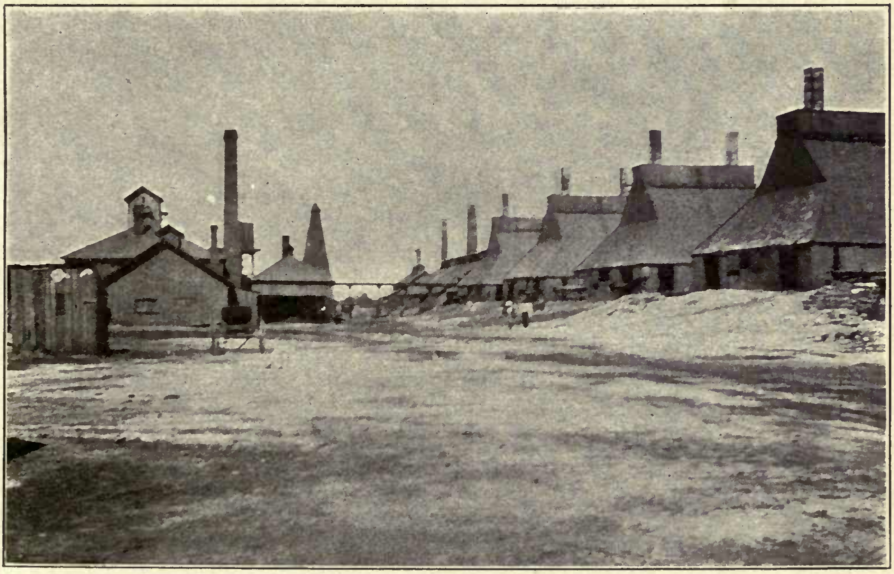

FIG. 3.—WORKS NO. 2 OF THE CHEROKEE-LANYON SPELTER CO., AT PITTSBURG, KAN.

Situated about 100 miles southwest of Chicago are the two large works at Peru and Lasalle, Ill., which though remote from the Joplin district have favorable railway rates from there and the additional advantage of a small supply of cheaper ore from Wisconsin and Iowa. Because of their more improved metallurgical practice, including the recovery of sulphuric acid, and their undertaking the manufacture of sheet zinc and acid phosphate of lime, in each of which industries they realize large profits, they are able to compete with the Iola smelters, notwithstanding the advantage of the latter as to fuel. The works of the Empire Zinc Co. at North Chicago, Ill., are also near the Wisconsin mines. The Mineral Point Zinc Co. fetches some ore from Colorado and New Mexico to its works at Mineral Point, Wis., where it makes zinc white and recently has made an installation to produce sulphuric acid by the catalytic process.

Near Marion, Ind., there are three small works which have higher freights to pay on Joplin ore than any of the other works, said rates being not offset by the lower freights on spelter to Pittsburg, Penn., which is the most important consuming district in the United States, wherefore those smelters are at a disadvantage; they burn natural gas but the supply is diminishing rapidly and is no longer to be had at little or no cost, but now involves an expense of perhaps 4@5c. per 1,000 cu. ft., which is equivalent to first-class run of mine coal, such as Pennsylvania bituminous, at $0·80@$1 per 2,000 lb. Besides the Joplin ore the Indiana smelters obtain small supplies from Wisconsin and Tennessee, the freight rates from the mines in the latter State being the same as from Joplin, Mo.

Tardy Development of Kansas-Missouri Zinc-Smelting Practice.—Until two or three years ago the smelters of Kansas and Western Missouri employed the same methods that were adopted when the zinc-smelting industry was first established there during the decade 1870-1880. A description of their conditions, written by F. L. Clerc in *The Mineral Resources of the United States* for 1882 is so accurate an account of the conditions at most of the works at Pittsburg, Kan., and vicinity in 1900 that it is here reproduced:

"The furnaces are built with the ash-pits above the ground, with a sloping bank of earth or cinder leading up to the furnace floor. The buildings are scarcely more than sheds, and are huddled together with little regard to their mutual relation. The first cost of the buildings is inconsiderable. In the smelting process the cheapness of the fuel renders economy in this direction unimportant, and the cheapness of living makes labor obtainable at wages as low as anywhere in the country. The works are usually owned by partners, who do the work of salaried employees, and consider as profit what would be only the interest on their money and their wages at some other occupation. The furnaces roughly constructed of inferior material will not long sustain the heat required to exhaust the zinc from the cinder, and it is the accepted opinion that there is no economy in "butchering" the furnace for the sake of a small additional percentage of metal; it is preferred to increase the production of the furnace and to reduce the cost of labor and fuel by increasing the charge of ore—in other words to butcher the ore and save the furnace. At the same time the personal supervision of the proprietors and their intimate knowledge of the business makes possible results that could not be expected by a company operating on a larger scale."

Since Mr. Clerc wrote as above there have been of course some improvements in the older Kansas and Missouri works and during the decade 1890-1900 most of them passed out of the hands of individual proprietors into those of joint stock companies, but nevertheless the smelting practice continued to be very backward in many respects. Recently, however, a radical change has taken place in zinc smelting in Kansas and because of the wonderful natural resources that have been taken advantage of spelter is now produced there under very different conditions than what formerly prevailed.

Rise of the Natural Gas Smelteries of Kansas.—Pittsburg, Kan., owed its development to the productive coal measures over which it is built, and being only 26 miles from Joplin, with excellent railway connections, it became naturally an advantageous place for smelting the ore mined at and near Joplin. About 1895 the natural-gas supply in the vicinity of Iola, which by railway is approximately 100 miles from Joplin, began to be used for zinc smelting and since then numerous works have been erected at that point. The experience has demonstrated that zinc ore can be smelted more cheaply at Iola, under the conditions which exist there, than at Pittsburg, Kan., and gradually the coal smelteries have been closed, though this result did not take place in a marked degree until 1900. In 1899 the conditions of the ore and spelter market were such that even the Iola smelters were unprofitable. With 1900 the unfavorable conditions were ameliorated, but the increasing competition of the Iola smelteries, of which the largest had previously been consolidated in strong hands, prevented the price of ore from falling to the former level and few of the coal smelters of Kansas and Missouri were able to meet the new scale of prices, especially under the further disadvantage of the increased price for coal which prevailed during 1900, and one by one their works were closed. Thus the metallurgical industry lost one of its most picturesque features, inasmuch as the irregular groups of buildings, with their high-peaked roofs and quaint gables, and chimneys

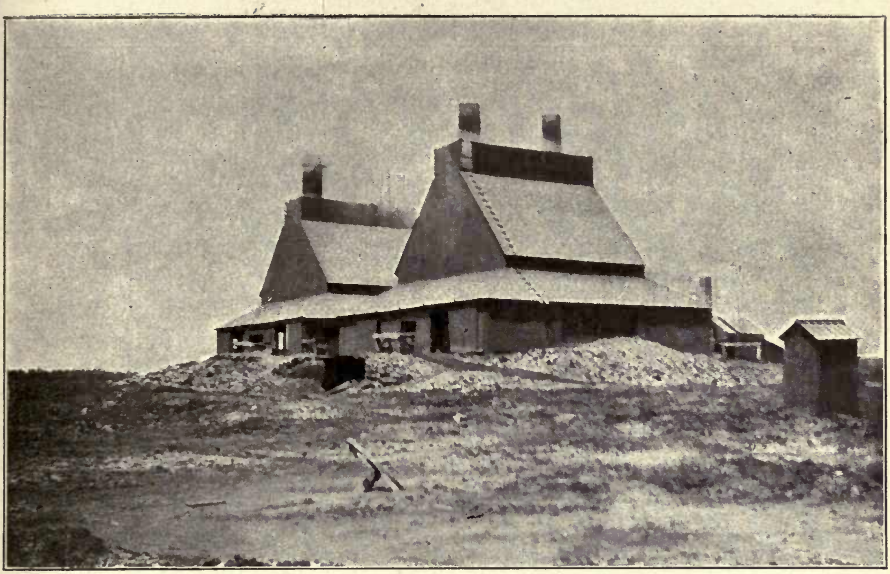

Fig. 4.—Zinc Smeltery at Cherokee, Kan.

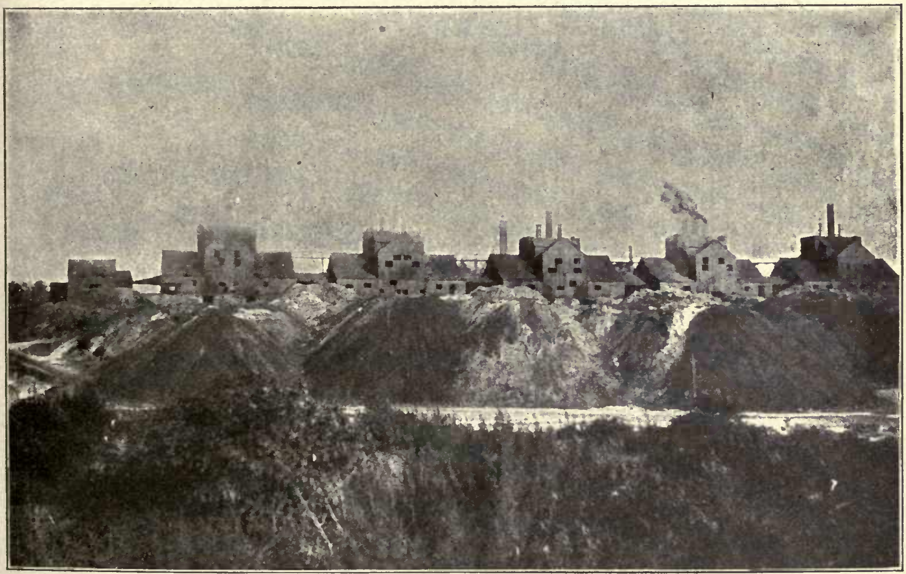

Fig. 5.—Zinc Smeltery at Nevada, Mo.

At this works the roasting furnaces are constructed on top of the distillation furnaces.

belching flames that made a far-seen landmark on the prairie at night, will soon become a thing of the past. With idleness a zinc smeltery rapidly goes to pieces and probably the Pittsburg works will never be started again for any but spasmodic campaigns. In the ordinary course of events the natural gas at Iola and elsewhere will be exhausted some day and even before it is wholly consumed it will begin to assume an expense increasing gradually up to the point where it will be more costly than coal; upon the arrival of that time Pittsburg may become a smelting center once more, but it will be, no doubt, with new furnaces and new appliances.

Coal Resources of the United States.[1]—Each of the zinc-smelting districts of the United States is situated in close proximity to a coal field or a natural-gas field, with the exception of the New Jersey works, to which the coal has to be carried from the anthracite mines of eastern Pennsylvania, and the works at North Chicago, Ill., and Mineral Point, Wis., which have to obtain their supply of coal from the northern field of Illinois. The works at South Bethlehem, Penn., and Palmerton, Penn., are situated in close proximity to the anthracite coal field of Pennsylvania. The Virginia smelters are located on the continuation of the great Eastern bituminous coal field which extends through Pennsylvania and West Virginia to Alabama. The smelters of Lasalle and Peru, Ill., are in the northern Illinois coal field, while those of Collinsville, Ill., and St. Louis, Mo., draw their supply from the southern Illinois field. Pittsburg, Kan., is located on the southern extension of the Iowa-Kansas coal field.

The anthracite coal mines of eastern Pennsylvania occur in three districts known as the Schuylkill, Lehigh and Wyoming. The area of the first is 138 square miles, of the second 38 square miles and of the third 196 square miles. There are 15 workable seams, having a total thickness of 107 ft. of coal, the thickness of the measures in which the seams are interstratified being about 3,000 ft.

At Lasalle and Peru in the northern Illinois coal field there are three workable seams of coal, one being from 4·5 to 5 ft. in thickness, another from 3 to 9 ft., averaging 6 ft., and a third 4 ft. The coal from the uppermost seam is light, dry and free-burning. The middle seam yields a poorer coal. The lowest seam is most highly bituminous, cakes in burning, is high in sulphur and throws off a heavy soot. In the Belleville district in southern Illinois, whence the Collinsville and St. Louis smelters obtain their coal, a seam from 5 to 7 ft. in thickness is principally worked. The Illinois coals are generally high in moisture and often are very high in sulphur and ash.

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[1] This section refers only to the coal resources of those parts of the United States in which zinc ores are smelted.

When burned in ordinary furnaces they produce great volumes of black smoke. The coal of northern Illinois is burned successfully in gas producers.

Pittsburg, Kan., and the adjacent towns of Weir, Scammon and Frontenac, overlie the thickest and best seam of coal in Kansas. This is known as the Cherokee seam. It extends from the Indian Territory, entering Kansas near Chetopa, and runs across the southeast part of Labette County, the west and northwest parts of Cherokee and the southeast part of Crawford, thence entering Missouri. The seam averages 40 in. in thickness. It is worked at Scammon, Weir, Cherokee, Fleming, Frontenac, Pittsburg, Arcadia and Minden. The Kansas coal is high in ash and moisture and throws off great volumes of black smoke in burning. It is high in pyrite, wherefore the ash clinkers badly, often forming a liquid slag which drips through the grate bars. Because of the high sulphur content iron grate bars are rapidly corroded unless water-cooled. This coal has not given successful results in gas firing; at least not in the early attempts in zinc smelting at Pittsburg, Kan., and Rich Hill, Mo., where, however, the unsatisfactory results may have been due largely to the imperfection of the system employed.

The anthracite coal of eastern Pennsylvania is in calorific power one of the highest grades of coal that is known, having a heating value of 14,600 to 14,800 B.T.U. per lb. of combustible. The bituminous coal of northern and southern Illinois and Kansas is very inferior as compared with the bituminous coal of western Pennsylvania, Wales and Belgium, having a heating value of only 13,500 to 14,800 B.T.U. per lb. of combustible; and being generally rather high in ash and moisture, its relative practical value is only about 66% of that of the semi-bituminous coal of Pennsylvania which is taken as the standard in the United States. In reckoning the advantages of different smelting districts it is highly important to bear in mind the relative efficiency of the coal available, since a good deal more work can be done with some kinds than with others.

The zinc smelters of northern Illinois employ gas firing and consequently run of mine coal, which the modern practice in gas firing has proved to be preferable to the use of slack and inferior grades of coal. The value of run of mine coal at Lasalle and Peru in 1899 was \$1@\$1·50 per 2,000 lb. The Matthiessen & Hegeler Zinc Co. mines its own coal. The smelters of southern Illinois and Kansas as a general thing use slack coal, which in 1899 cost \$0·35@\$0·60 delivered at their works. In 1900 there was a great rise in the value of coal, especially in Kansas, where the same class which had previously been available at about \$0·50 per 2,000 lb. delivered could not be contracted for under $0·75. Run of mine coal which in 1899 was worth $1@$1·10 per ton at Pittsburg, Kan., in 1900 rose to $1·50@$1·75. The common kinds of southern Illinois coal at the end of 1900 commanded $1·10@$1·20 for run of mine delivered at East St. Louis.

Natural-Gas Supply.—The competition of the natural-gas smelters of Iola and Cherryvale, Kan., has now rendered zinc smelting with coal-fired furnaces in Kansas and Missouri largely a thing of the past. The natural gas under existing circumstances is almost a costless fuel, the smelters being only at the expense of putting down the wells and piping the gas to their furnaces, the cost of maintenance being very small so long as the gas pressure continues high. Some of the smelters, however, have made large outlays for acquiring gas lands and consequently are burdened with interest charges and rentals, which are of course properly debited to the cost of the gas, but so far at least the smelter who has been in possession of a 10-acre lot with a producing gas well has been about as well off as one who has acquired 10,000 acres of land, inasmuch as wells which produce gas draw directly from the subterranean reservoirs without respect to land divisional lines. As the gas flow from a single well diminishes, however, the advantage of having a large area from which to draw by means of other wells becomes apparent, wherefore the more prudent smelters have acquired extensive gas rights. The rapidity with which such investments should be written off for redemption out of the profits of the smelting is necessarily an extremely uncertain element. The experience in Pennsylvania and Indiana has demonstrated that natural gas becomes more and more costly as the supply is exhausted, wherefore prudence would indicate that the resources of Kansas should be used economically.

The natural-gas field at Iola, Kan., is comparatively small in area. Its length in an east and west line has been demonstrated by the drill to be about seven miles, its breadth being about three miles. The gas rock which lies at a depth of 800 to 1,000 ft. below the surface is a sandstone of medium-sized grain, the stratum having an average thickness of 20 to 25 ft. According to Edward Orton, State Geologist of Ohio, who studied the geological structure of the Iola gas field in 1898, its gas-bearing formation is of the reservoir type, as distinguished from the shale-gas type. The characteristics of the reservoir type are: (1) Large flows of gas from single wells, volumes amounting to tens of millions of cubic feet per day being known; (2) approximately the same pressure in the wells tapping the rock, irrespective of their widely differing volumes; (3) accompaniment of the gas by petroleum with which is associated water, usually salt water. The gas of these porous rocks generally comes to a sudden end, says Professor Orton, oil comes in and fills the pipes or salt water shuts off the gas "like a light blown out by a gust of wind." Only by constant care and attention in removing those substances from the pipes can the life of the well be maintained, especially in its later stages.

At the time when Professor Orton made his examination upward of two dozen wells had been drilled in the field, their production ranging from 2,000,000 to more than 10,000,000 cu. ft. per 24 hours. There were half a dozen of the number producing about 7,000,000 cu. ft. per 24 hours apiece. The rock pressure of the field was 325 lb., with an outside range of 5 lb. in a single well. A little oil had been found, mainly on the western boundary of the field and at a lower depth than the gas. Salt water occurred below both gas and oil, but up to that time had not proved aggressive; the height to which it rose had not been determined, but was less than several hundred feet.[1]

So long as the pressure of a natural-gas supply is adequate to cause the delivery of the requisite volume of gas, the only expense for the latter is the rental of the land, and the first cost of putting down the wells and piping the gas to the point of consumption. As the pressure diminishes, however, it becomes necessary to put down more wells in order to obtain the same volume of gas at the reduced pressure; more and longer pipe lines must be laid; and eventually the wells must be pumped. Under those conditions the gas begins to assume a positive cost, which increases gradually as the supply falls off until the point is reached where it becomes more expensive than coal, after which it can be considered only as a luxury or as an industrial fuel for purposes in which the cost is a minor consideration. This stage in the history of natural gas has already been reached in western Pennsylvania and Indiana. At Pittsburgh, Penn., the cheapest gas furnished in large quantities for metallurgical purposes in the autumn of 1899 commanded 8c. per 1,000 cu. ft., at which price it was more expensive than gas made artificially from the local coal. With modern gas producers and run of mine coal costing $0·80@$0·85 per 2,000 lb., which was the average cost of coal at Pittsburgh at that time gas could be made artificially for 2·5c. per 1,000 cu. ft. of combustible, and reckoning 2 cu. ft. of combustible in that form as being equal in calorific power to 1 cu. ft. of natural gas[1] the latter would have had to be obtainable at 5c per 1,000 cu. ft. to be at a parity in so far as actual cost is concerned. A coal cost of $1 per ton would raise the cost of the combustible in producer gas to 2·8c. per 1,000 cu. ft., equivalent to natural gas at

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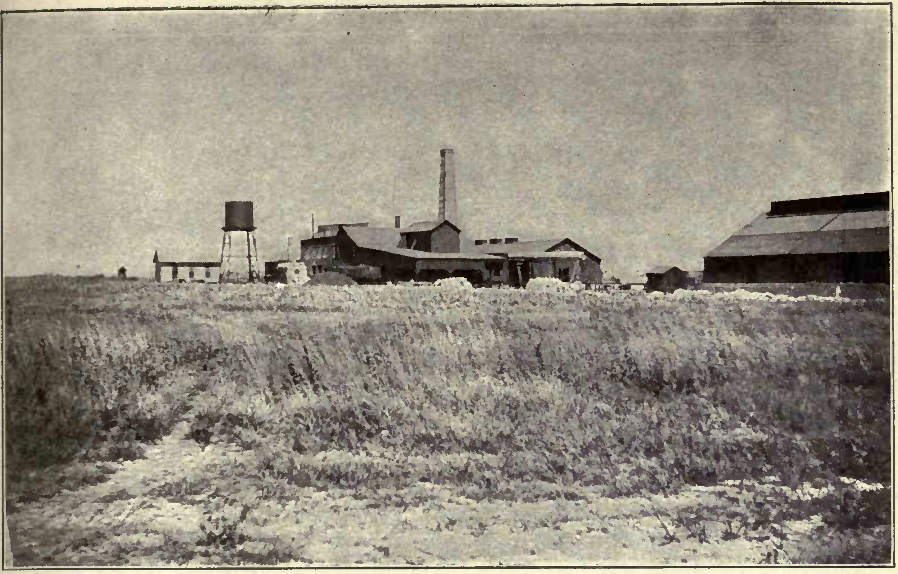
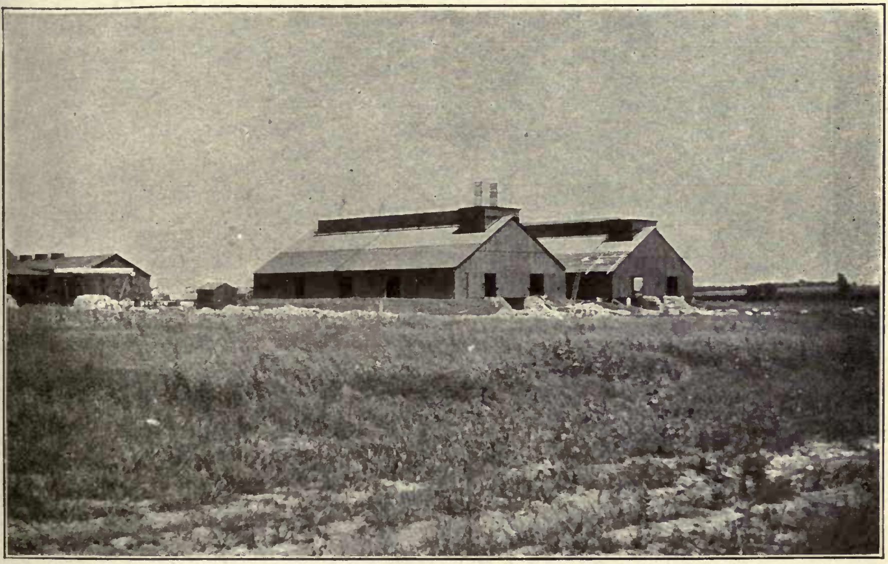
[1] Reports are conflicting as to how the Iola gas supply is holding out under the heavy drain upon it, especially during the last three years; in some parts of the field at least there appears to have been a serious diminution in pressure.

[1] This is the ratio adopted by experienced gas producer engineers at Pittsburg, Penn.





FIGS. 6 AND 7.—WORKS OF THE CHEROKEE-LANYON SPELTER CO., NEAR IOLA, KAN.

These engravings show the works in course of construction (in 1899). Fig. 7 is a continuation of Fig. 6 to the right.

5·6c. The producers of natural gas in Pennsylvania and West Virginia say that the actual cost of the gas to them at points near the wells is now about 5c. per 1,000 cu. ft. It has that cost because of the expense of their land leases, the cost of drilling the wells, the cost of piping the gas to points of consumption, the cost of pumping and the cost of maintaining the supply by means of new wells and additional pipe lines as it becomes exhausted by consumption. The drain on those fields is so great that the expense for maintenance is a large and constantly increasing charge. The experience in the Indiana gas field has been similar, and the zinc smelters who located there in 1892 have been for several years short of gas, to the great impediment of their operations. The gas which they now obtain is said to cost them 4@5c. per 1,000 cu. ft. Both at Iola, Kan., and in the Indiana gas field the original pressure was greatly inferior to that which was registered in western Pennsylvania and West Virginia. In Indiana the original rock pressure was 325 lb., or about the same as at Iola, Kan. In 1896, after 10 years' drain from the field, the rock pressure was 220 lb.; in 1897 it was only 195 lb. The average diminution in pressure in Indiana in recent years has been 20 lb. per annum. The wells in Indiana cease to be serviceable at 100 lb. pressure.

Besides the Iola field natural gas has been found in Kansas at numerous isolated points in a southerly direction from Iola; Independence and Cherryvale being the extreme southern extensions at the present time. There are zinc smelteries now in operation at Cherryvale and at Neodesha. The location of these various gas fields is shown by the map on page 184.

Character of the Zinc Ore Smelted in the United States.—The smelters of New Jersey and Pennsylvania use only the willemite produced in New Jersey; those of Virginia use only the calamine mined in that State, together with a little obtained from Tennessee. The Indiana smelters obtain their supply of ore from Wisconsin, Tennessee, and from the Joplin district; they smelt chiefly blende. The large smelters of northern Illinois derive the most of their ore supply from the Joplin district, but get a considerable quantity from Wisconsin and a small quantity from Iowa; they use chiefly blende. The smelters of the St. Louis district are also dependent chiefly upon the Joplin district. The Kansas smelters obtain by far the more part of their ore supply from the Joplin district, using both blende and calamine, but recently have been getting some ore from Colorado, which although of inferior quality to the Joplin ore has been found to give good results when properly mixed and offers the advantage of being obtainable at a much lower cost than the Joplin ore. The great zinc ore supply of the Western smelters is the Joplin district, where the chief part of the output is a concentrated blende, dressed to a standard of 60% Zn and containing as a general thing less than 2% Fe and 1% Pb. It is the highest grade of zinc sulphide ore produced anywhere in the world and because of its excellence commands naturally a high price.

Cost of Refractory Material.—The smelters of Kansas, the St. Louis district and northern Illinois all use fire clay dug at Cheltenham, a division of St. Louis, for the manufacture of their retorts, no other material having been found so well adapted for that purpose. This clay costs $1@$1·50 f.o.b. cars at the pits. The freight rate from St. Louis, Mo., to Pittsburg, Kan., is $2 per ton, making the cost delivered at works at the latter place $3@$3·50 per ton. This is for raw unmilled clay. Chamotte, or cement as it is commonly called in Missouri and Kansas, costs $4@$4·50 per ton delivered at the smelteries in Kansas. The Kansas smelters and most of those in Illinois employ St. Louis fire brick, of which the ordinary kind costs $13 per M. at St. Louis; a medium grade is obtainable at $16 per M., while the highest grade commands $25 per M. The freight rate on fire brick from St. Louis, Mo., to Pittsburg, Kan., is $2 per ton—i.e., $6·50 per M. Fire brick blocks and the special shapes required for relining furnaces cost $9@$10 per ton at St. Louis, the freight rate to Pittsburg, Kan., being also $2 per ton. The smelters of New Jersey and Pennsylvania use clay from Cheltenham, Mo., and from Woodbridge, N. J.

Wages of Labor.—At Pittsburg, Kan., the general rate of wages for common labor is $1·25 per 10 hours. Cokemen get $1·50 and brickmasons $4. The wages of smiths and mechanics are about as in other parts of the United States. Carpenters receive less, generally $1·50@$2. On the distillation furnaces the brigadiers are paid $4·50@$4·70 per double shift of 24 hours; "long shifts" $3·60@$3·80 per double shift; "short shifts" $1·30@1·40 per single shift.

At Iola, Kan., common labor is paid the same as at Pittsburg, namely $1·25 per 10 hours. For work on the distillation furnaces, brigadiers or firemen receive $2·50@$2·75; chargers, $2·30; metal drawers, $2·35; helpers $1·40@$1·60, these rates being per shift of 12 hours in each case, except in those where the daily work is finished in a shorter time.

In New Jersey, Pennsylvania and Indiana the furnacemen receive less than in the West. In Indiana chargers are paid $1·90@$2·00 per shift of 12 hours; "long shifts" get $1·75; "short shifts" $1·25@$1·35; metal drawers $1·50; ash wheelers and general laborers $1·25.

Centralization of the American Consumption of Spelter.—Although the zinc-producing industry of the United States has so far escaped the tendency toward consolidation of interests to the same degree which has been displayed in many other branches of American industry, both the productive capacity and the consumptive demand for spelter have been centralized in a striking manner. Upward of 50% of the consumption of spelter in the United States is for the purpose of galvanizing iron, which business is now chiefly in the hands of the constituent companies of the United States Steel Corporation.[1] The manufacture of sheet zinc is in the hands of four companies.[2] The manufacture of brass in Connecticut, which is the principal center of that industry, is controlled by one company. The consumption of spelter for use in the desilverization of lead is also chiefly in the hands of one corporation.[3] It is safe to say, therefore, that 75 or 80% of the demand for American spelter now comes from seven corporations. On the other hand the production of spelter has also been centralized, practically the whole of the active smelting capacity being now divided among seven strong concerns. What will be the effect upon the American zinc industry of this concentration of demand and supply it is too early to forecast.

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[1] The United States Steel Corporation through certain of its constituent companies controls the Edgar Zinc Co. and the Girard Smelting Co., which together produce about 25,000 tons of spelter per annum.

[2] New rolling mills have been built in 1901 by the New Jersey Zinc Co. and Lanyon Zinc Co.; they will be in operation in 1902.

[3] The United States Zinc Co., which is affiliated with the American Smelting and Refining Co., has planned to build a zinc smeltery at Pueblo, Colo., during 1902.

III

USES OF ZINC AND ZINC PRODUCTS.

Zinc, which in English-speaking countries is commonly known in trade under the name of spelter, is used in the arts for a great variety of puposes. Rolled into sheets it is employed in architecture for roofs of buildings, water tanks, conduits, etc. Alloyed with copper, in varying proportions, it forms the valuable compounds, brass and bronze, while its combinations with other metals find various uses in the arts. Iron dipped into the molten metal becomes coated with it and is thereby protected against oxidizing agents; large quantities of spelter are used in this operation, which is called galvanizing; it was first patented by Crawford in 1837. The oxide of zinc, produced either from the metal or directly from the ore, forms a white pigment which is second in value only to white lead, and is extensively employed.[1] A large quantity of zinc white is used by the rubber trade for admixture with the gum in the preparation of many articles. Most of the zinc produced in the world is used in these ways, but considerable quantities are consumed in galvanic batteries; in photo-engraving; in plates hung in boilers to prevent the formation of scale; for desilverizing lead bullion; for precipitating gold in the cyanide process; in the form of powder (known variously as zinc gray, zinc dust, blue powder, etc.) as a reducing agent in organic chemistry (especially the reduction of indigo blue) and as a paint for iron; and in the preparation of numerous salts, of which the most important are the chloride, employed as a preservative for wood, and the sulphate, employed in medicine, in dyeing, in the manufacture of glue and in the preparation of a zinc-barium white known as lithophone.

Itemization of the Consumption of Spelter.—There are no statistics compiled regularly as to the consumption of spelter itemized according to uses,

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[1] Opinions differ as to the relative value of white lead and zinc white. Probably a mixture of the two is better than either alone, and in many cases the addition of a certain proportion of barytes is doubtless advantageous. In point of consumption, white lead still holds the first place, but zinc white is gaining rapidly.

except in a few of the countries of Europe, where the production of sheet zinc is reported. A large portion of the production of spelter, both in France and Great Britain is rolled into sheet. The Vieille Montagne Company, which operates works in France, besides those in Belgium and the west of Germany, rolls nearly all of its spelter production, as will be seen by reference to Chapter IV. W. H. Seamon stated in the *Engineering and Mining Journal* of October 24, 1896, that at that time upward of 100,000 tons of zinc sheet were used annually for roofing purposes in Europe. Although there are no complete statistics to indicate the channels of consumption, there is no doubt that a larger proportion of the zinc production of Europe is employed in the form of sheet zinc than is the case in the United States.

In 1892 it was estimated by W. H. Seamon that out of a total consumption of spelter in the United States of 78,040 tons of 2,000 lb., the galvanizing trade used 35,000; the brass makers 20,500; the rollers of sheet zinc 15,500; the desilverizers of lead bullion 3,500; while the remaining 3,540 tons were employed for miscellaneous purposes. In 1898, according to statistics which I had occasion to collect, out of a total consumption of 105,000 tons the galvanizers used about 55,000 tons (52%); the brass makers 24,000 (23%); the rollers of sheet zinc 20,000 tons (19%); the lead desilverizers 1,500 tons (1·5%), and miscellaneous consumers 4,500 tons (4·5%). In *The Mineral Industry,* vol. VIII, it was estimated that of the consumption of spelter in the United States in 1899, 50% was used in galvanizing, 15% in brass making, 20% in the form of sheet, and 15% for other purposes. The agreement between those figures and my own for the previous year is very close with respect to the galvanizing and sheet zinc industries. With respect to brass and consumption for other purposes, I am disposed to consider that *The Mineral Industry* underestimated the former and overestimated the latter.[1] These statistics for the United States, it should be noted, do not take any account of the consumption of zinc in the form of zinc oxide for use as a pigment and other purposes, that product being made in the United States directly from ores, while in Europe an important part of the spelter production is consumed in its manufacture.

SHEET ZINC.—In Europe the chief use of sheet zinc is as roofing material, for which purpose it is in great favor. It was first employed for roofing

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[1] According to the statistics of the Eleventh Census, which have been published since this paragraph was put in type, the production of sheet zinc in the United States in 1899 was 17,723 tons, out of a total spelter output of 131,546 tons. It should be remarked, however, that during a part of 1899 the production of sheet zinc was considerably restricted by the abnormal market conditions.

in 1811, when the Abbé Dony, the founder of the Belgian zinc industry, in his efforts to create a market for the consumption of his small output of spelter, then practically a new and unknown metal, roofed a house with sheet zinc. During the same year portions of the roof of the church of St. Barthelemy at Liège were covered with sheet zinc. According to W. H. Seamon, both those roofs were in good condition in 1896.[1] In 1820 the Theatre de la Monnaie at Brussels was roofed with zinc and up to the time of its destruction by fire in 1855, it is authoritatively stated that no repairs had been required. Mosselman, the successor of Dony and the founder of the Société Anonyme de la Vieille Montagne, continued the experiments of his predecessor with marked success, and without any special effort on the part of the company many of the buildings in Belgium, France and Germany were roofed with the metal, to the extent that during the year 1836 it is said that upward of 12,000 tons of zinc were used for roofing purposes in France, while the consumption in England amounted to only between 2,000 and 3,000 tons.[2] In 1867 the zinc producers of Europe had careful examinations made of the then existing roofs, from the observations on which conclusions as to the best method of laying such roofs were deduced and workmen were trained to handle the metal properly. Since that time there has been a large increase in the consumption of sheet zinc for roofing material in Europe, where it is now recognized as highly desirable because of its great durability and its economy as compared with lead and copper. Its comparatively low cost has secured its application on structures of all kinds, including such buildings as the German Imperial Palace, the University of Bonn, the Berlin Academy of Fine Arts, the Cathedral of St. Mary at Duesseldorf, Germany, the Hôtel de Ville and the Cathedral de Sainte Clotilde in Paris, the Haymarket Theater, Canterbury Cathedral and the Government Dock Yards in England. In the United States previous to 1890 it is said that there were only three buildings roofed with sheet zinc. Probably the number of additions to the list between 1890 and 1900 was small, inasmuch as American architects and builders are still generally ignorant of the advantages of the material.

Dimensions and Weight of Sheet Zinc.—In Europe the business of zinc roofing is conducted partly by the manufacturers and partly by roofing companies. The sheets are commonly corrugated and stamped at the rolling mills, though some of the roofers cut and stamp their own sheets. The usual length of these sheets is 7 ft. or 8 ft., but sheets 10 ft. long may be obtained on special orders; in width the regular sheets vary from

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[1] Eng. & Min. Jour., Oct. 24, 1896.

[2] Journal of Gas and Sanitary Engineering, 1893, p. 419.

2 ft. 8 in. to 3 ft.[1] Sheets of the following weight and thickness are recommended for roofing purposes by the Société Anonyme de la Vieille Montagne, which company has had a very extensive experience in this business.

Gauge	THICKNESS		Weight in pounds per sq. ft.	WEIGHT OF SHEETS IN POUNDS		
	Inch	Mm.		33 in. × 72 in.	36 in. × 84 in.	36 in.×96 in
No. 13	0·029	0·740	1·088	17·95	22·85	26·11
No. 14	0·032	0·820	1·200	19·80	25·20	28·80
No. 15	0·038	0·950	1 425	23·51	29·93	34·20
No. 16	0·043	1·080	1·613	26·61	33·87	38·71
No. 17	0·048	1·210	1·800	29·70	37·80	43·20
No. 18	0·053	1·340	1·988	22·80	41·75	47·71

Sheet Zinc Gauges.—The thickness of metal in decimals of an inch and the weight in decimals of a pound per square foot by the conventional gauges for sheet zinc are shown in the following table:

WEIGHT AND THICKNESS OF SHEET ZINC. [a]

Gauge Number	AMERICAN		BELGIAN		VIEILLE MONTAGNE	
	Thickness Decimals of an inch	Weight per sq. ft. lb.	Thickness Decimals of an inch	Weight per sq. ft. lb.	Thickness Decimals of an inch	Weight per sq. ft. lb.
1	0·002	0·075	0·0018	0·068	0·004	0·150
2	0·004	0·150	0·0036	0·135	0·006	0·225
3	0·006	0·225	0·0055	0·206	0·007	0·263
4	0·008	0·300	0·0073	0·274	0·008	0·300
5	0·010	0·375	0·0091	0·341	0·010	0·375
6	0·012	0·450	0·0110	0·413	0·011	0·413
7	0·014	0·525	0·0128	0·480	0·013	0·488
8	0·016	0·600	0·0146	0·548	0·015	0·563
9	0·018	0·675	0·0165	0·619	0·018	0·675
10	0·020	0·750	0·0180	0·675	0·020	0·750
11	0·024	0·900	0·0217	0·814	0·023	0·863
12	0·028	1·050	0·0254	0·953	0·026	0·975
13	0·032	1·200	0·0290	1·088	0·029	1·088
14	0·036	1·350	0·0326	1·223	0·032	1·200
15	0·040	1·500	0·0364	1·365	0·038	1·425
16	0·045	1·688	0·0400	1·500	0·043	1·613
17	0·050	1·875	0·0437	1·639	0·048	1·800
18	0·055	2·063	0·0478	1·793	0·053	1·988
19	0·060	2·250	0·0509	1·909	0·058	2·175
20	0·070	2·625	0·0581	2·179	0·063	2·363
21	0·080	3·000	0·0728	2·730	0·070	2·625
22	0·090	3·375	0·0764	2·865	0·077	2·888
23	0·100	3·750	0·0800	3·000	0·084	3·150
24	0·125	4·688	0·0896	3·360	0·091	3·413
25	0·250	9·375	0·0992	3·720	0·098	3·675
26	0·375	14·063	0·1088	4·080	0·105	3·938
27	0·500	18·750	....	....	....	....
28	1·000	37·500	....	....	....	....

[a] The weight of a cubic foot of rolled zinc is 450 lb., whence a sheet 1 in. thick should weigh 37·5 lb., 0·1 inch 3·75 lb., and 0·01, 0·375 lb. per sq. ft. The weights in the above table have been computed on that basis.

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[1] Ordinary dimensions of zinc sheets are 7 ft.×2 ft. 8 in.; 7 ft.×3 ft.; and 8 ft.×3 ft. In Europe, sheets of those dimensions are made regularly in all gauges from No. 6 to No. 26, both inclusive; Nos. 1 to 5 are rolled only to order and of special dimensions.

The "American" gauge is also known as the "Matthiessen & Hegeler"; it is used by the Matthiessen & Hegeler Zinc Co. and the Illinois Zinc Co. What I have called the "Belgian" gauge is taken from *Notes on Building Construction,* by Col. V. Smith. The figures under the caption "Vieille Montagne" are taken from a pamphlet distributed by the Société Anonyme de la Vieille Montagne at the Chicago Exposition in 1893.

Directions for Laying Sheet Zinc Roofs.—As a general rule solder should not be used in laying roofs with sheet zinc, since the metal is always weakened somewhat at the soldered junction, although perhaps not more than other metals are. The expansion and contraction of zinc being greater than with other roofing metals, rigid joints should generally be avoided. There is not, however, any special difficulty in soldering sheet zinc, except that a little more care must be taken in wiping and smoothing the joints than with some other metals. The best solder for this purpose is composed of 33·33% Pb and 66·67% Sn, but any of the common varieties may be used. Soldering flux, known as "killed spirits," which is made by saturating commercial muriatic (chlorhydric) acid with strips of zinc, is applied in the usual way. The nails employed in laying a zinc roof should be made of zinc, but since such nails cannot be driven into hard wood, galvanized or plain iron nails are sometimes employed.

Advantages of Zinc Roofs.—Sheet zinc owes its value for roofing purposes to its durability, lightness and economy as compared with galvanized iron, tin plate, lead, copper, slate and tile. Galvanized iron being coated with zinc should possess theoretically as much durability as sheet zinc and less weight for the same strength, together with less first cost, but as a matter of fact the union of the zinc and iron effected in the process of galvanizing is not sufficiently strong to withstand long the unequal expansion of the two metals, wherefore the zinc coating gradually scales off, exposing the iron, and thus creates an electrical couple, which results in the more rapid corrosion of the iron and destruction of the roof, although that may be delayed somewhat by frequent and thorough painting. With the greatest care, however, 15 years is a long life for a roof of galvanized iron.[1] The superiority of sheet zinc over galvanized iron was shown in the case of the Northwestern Railway station at Birmingham, England, which was roofed in 1853 with the latter and was carefully painted on both sides every three years and repaired whenever necessary, but at the end of 13 years was found to be so rotten that it had to be removed; it was replaced by a zinc roof, which still exists in perfect condition and has given but little trouble or expense for repairs. The coating of basic zinc

[1] W. H. Seamon, loc. cit.

carbonate which forms on the surface of a zinc roof is practically insoluble in atmospheric water and thoroughly protects the underlying metal from further oxidation by atmospheric agents. A zinc roof of the proper gauge weighs from 125 to 180 lb. per 100 sq. ft., against 800 lb. for lead and copper, 700 to 900 lb. for slate, and 1,500 lb. for tiles.

Zinc should not be allowed to come in contact with iron, copper or lead, since thereby voltaic couples are established, which destroy the zinc, especially in the presence of moisture. Zinc should not be laid on wood, such as oak, which contains acid and should not be exposed to calcareous water. Zinc laid on flat roofs to which cats can gain access are also soon corroded.[1] Another objection to zinc as roofing material is that in case of fire it ignites and blazes furiously (Bloxam).

Methods of Roofing with Sheet Zinc.—Zinc roofs are commonly laid in Europe by the "roll-cap" system, or as corrugated sheets, or as shingles (or tiles). In the roll-cap system, which is recommended for slopes of not less than 20° and not more than 36°, the zinc in sheets of 6 to 8 ft. length, usually of No. 13 gauge, is laid upon a board sheathing, covering the rafters in the usual manner, with battens laid upon the sheathing from comb of the roof to eaves, parallel with the rafters and spaced at regular intervals apart. The edges of the sheets of zinc are turned up for about 1 in. against these battens, which are then covered with a roll cap of sheet zinc, making the union perfectly water- and weather-tight. Corrugated sheet zinc is used in the same manner as corrugated iron, but because of the greater flexibility of the metal can be worked more easily; on the other hand its inferior transverse strength necessitates that when the material is to be laid directly on the roof frame, the purlins must be closer together than with corrugated iron of the same weight. When corrugated sheet zinc is used on board sheathing, No. 13 gauge is sufficiently heavy, but when no sheathing is employed, a heavier gauge (up to No. 18) should be used. Zinc shingle roofs are adapted to all slopes greater than 10° and are considered to give the best results obtainable by the employment of zinc for roofing material. The shingles or tiles are made in square, hexagonal and diamond shapes. When used on dwelling houses they are made of No. 13 gauge zinc and are usually 10·5, 13·5 or 17·5 in. square. For use on large roofs they are made 23·5 and 29 in. square. These shingles are folded over on their edges so as to engage with the succeeding overlying ones of the next row, the method of laying being the same as with ordinary board shingles—i.e., in rows beginning at the eaves and proceeding upward to the comb of the roof.

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[1] Col. V. Smith, Notes on Building Construction, vol. III (Rivingtons, London).

Cost of Roofing with Zinc.—The Société Anonyme de la Vieille Montagne gives the following data as to the roof surface which can be covered on the average by an experienced zinc worker, with a helper, in a day of 10 hours: ordinary roll-cap system, 170@190 sq. ft.; ordinary corrugated sheet zinc, 210 sq. ft.; patent corrugated sheet zinc, 190 sq. ft.; diamond-shaped zinc tiles, 110 sq. ft.; 11×11 in. square tiles, 130 sq. ft.; 13⅜×13⅜ in., 150 sq. ft.; 17×17 in., 180 sq. ft.; 23¼×23¼ in., 190 sq. ft.; fish scales, 100 sq. ft.; flat roof, sheets divided from each other by sunken gutters, 170 sq. ft.; wall facing, imitation of brick, 140 sq. ft.; wall facing, imitation of stone, 170 sq. ft. These figures represent work on medium-sized surfaces, with the average number of chimneys and other interruptions of the roof's continuity. On large roofs the figures can be increased considerably over those specified above, while on roofs with more than the average number of interruptions the figures will be decreased.

COST OF ZINC ROOFING.

Style of roofing.	Size of sheets in inches.	Gauge of zinc.	Weight per 100 sq. ft., including clips—lb.	Cost of material in Europe.	Contract price of material and laying in Europe.	Cost of material in New York.	Estimated cost of material and laying in New York.
Ordinary roll cap.	36 × 96	No. 13	133	$5·36	$8·37	$7·65	$12·15
		No. 14	145·5	6·26	9·26	8·37	12·87
Patent roll cap.	36 × 96	No. 13	138	5·56	8·57	7·93	12·43
		No. 14	151	6·08	9·08	8·68	13·18
Patent double ribbed ....*a*	39⅜ × 51½	No. 13	140·5	5·66	9·30	8·11	13·57
		No. 14	155·5	6·27	9·91	8·86	14·32
Tiles, square.	13⅜ × 13⅜	No. 13	185·5	8·28	10·45	10·66	14·06
	23¼ × 23¼	No. 13	150	6·65	9·50	8·62	12·89
Tiles, hexagonal.	10⅛ × 16	No. 13	188	8·33	10·65	10·81	14·29
Tiles, fish-scale.	24⅜	No. 13	151	8·14	12·20	12·22	18·39
Ordinary corrugated......*b*	29½ × 83½	No. 15	171	7·23	9·84	10·34	14·25
" " ......*c*	29½ × 83½	No. 15	175	7·40	10·10	10·58	16·13

a Distance between pairs of ribs, 13 in.; number of sheets per square, 7·44.
b Iron frame. *c* Timber frame.

COMPARATIVE COST OF VARIOUS ROOFS UNDER AMERICAN CONDITIONS (1896).

	Zinc.	Tin.	Slate.	Galv. iron.	Tiles.	Lead.	Copper.
Original cost per square, including laying	$11·75	$8·85	$10·50	$5·75	$12·50	$21·00	$30·00
Repairs in 30 years.	1·15	11·25	3·50	11·50	2·50	....	....
Interest, 6% on cost, 30 years.	21·15	16·75	19·90	11·35	22·50	37·80	54·00
Interest on repairs @6%, 15 years.	1·00	10·12	3·15	10·35	2·25	....	....
Totals.	$35·05	$46·87	$37·05	$38·95	$39·75	$58·80	$84·00

The above tables are due to W. H. Seamon, who computed the cost of zinc roofing on the bases of value of $4·03 per 100 lb. for sheet zinc in Europe and $5·75 per 100 lb. at New York. In estimating the cost of laying zinc roofs in New York, the European prices for the same work were assumed as bases and were increased by 50%.

In the item for repairs on the tin roof, it was assumed that it would last 30 years, provided it were painted every two years, at a cost of $0·75 per sq. ft., but no allowance was made for the re-soldering of the tin roof which is occasionally necessary, a similar re-soldering being required in the case of zinc roofs. At the expiration of 30 years the tin roof as well as the galvanized iron roof must be renewed; the slate roof may possibly be good for another 30 years, but the zinc roof will be good for 50 more at least. It is proper to take into consideration also that sheet zinc has a direct value as old metal when it is necessary to replace it, or in case of demolition of the building, which is not possessed by tin plate, galvanized iron, slates or tiling. Another important advantage over slate and tile roofs is that in many buildings the roof framing can be made lighter.

Employment of Zinc Plates to Prevent Boiler Corrosion.—Weakening of the plates by corrosion is one of the greatest dangers to which boilers are liable. Various methods are used to prevent it. One of the best has been found to be the suspension of zinc plates in the water in the boiler by wires or rods soldered to the upper part of the shell, so as to make an electrical connection. The zinc plates thus suspended in the corrosive water form with the steel plates of the boiler a galvanic battery, and the zinc being gradually consumed, the steel is protected thereby. In Europe especially this is a favorite method of preserving steam boilers and a considerable quantity of zinc is consumed in connection with it. The manner in which the zinc is employed was described in a report of the Committee on Boilers of the Institution of Mechanical Engineers in 1884 as follows:

"Of all the preservative methods adopted in the British service, the use of zinc properly distributed and fixed has been found the most effectual in saving the iron and steel surfaces from corrosion, and also in neutralizing by its own deterioration the hurtful influences met with in water as ordinarily supplied to boilers. The zinc slabs now used in the navy boilers are 12 in. long, 6 in. wide, and 0·5 in. thick; this size being found convenient for general application. The amount of zinc used in new boilers at present is one slab of the above size for every 20 i. h. p., or about 1 sq. ft. of zinc-surface to 2 sq. ft. of grate-surface. Rolled zinc is found the most suitable for the purpose. To make the zinc properly efficient as a protector especial care must be taken to insure perfect metallic contact between the slabs and the stays or plates to which they are attached. The slabs should be placed in such positions that all the surfaces in the boiler shall be protected. Each slab should be periodically examined to see that its connection remains perfect, and to renew any that may have decayed; this examination is usually made at intervals not exceeding three months. Under ordinary circumstances of working these zinc slabs may be expected to last in fit condition from 60 to 90 days immersed in hot sea-water; but in new boilers they at first decay more rapidly. The slabs are generally secured by means of iron straps 2 in. wide and 3/8 in. thick, and long enough to reach the nearest stay, to which the strap is firmly attached by screw-bolts."

Steam engineers are not unanimous, however, with regard to the efficiency of zinc as a preventive of boiler corrosion. Some consider that zinc is not only of no use, but may even be harmful. In an article concerning this subject the *Locomotive* stated that in one case a tubular boiler had been troubled with a deposit of scale consisting chiefly of organic matter and lime. Zinc was tried as a preventive. Its beneficial action was so obvious that its continued use was recommended, with frequent opening of the boiler and cleaning out of detached scale until all the old scale should be removed and the boiler become clean. Eight or ten months later the water supply was changed, the new supply being supposed to be free from lime and to contain only organic matter. After two or three months the tubes and shell of the boiler were found to be coated with an obstinately adhesive scale, composed of zinc oxide and the organic matter or sediment of the water used. The deposit had become so heavy in places as to cause overheating and bulging of the plates over the fire.

The experience cited by the *Locomotive* should not be considered as evidence why zinc should not be used to prevent boiler corrosion, but merely as indicative that the use of zinc may not be advisable with all kinds of water. The experience in Europe and elsewhere has demonstrated the general efficiency of zinc for this purpose, but there are exceptions to all generalizations and the failure of zinc to act in boilers as desired should not be ascribed to inefficiency of the principle, but on the contrary to the lack of preliminary investigation as to the conditions under which it was to be applied.

Consumption of Sheet Zinc in the Cyanide Process of Gold Extraction.—The metallurgical works of the West which employ the cyanide process for extraction of gold consume a considerable quantity of sheet zinc, which is prepared by the rolling mills in the form of disks, 12 in. in diameter, with a 1-in. hole in the center, these disks being made usually of zinc of No. 9

gauge. A 12-in. disk of that thickness weighs about half a pound. They sell at the rolling mill for about 2c. per lb. above the price of prime Western spelter.

The cyanide works use such disks for the preparation of zinc shavings, turning off by means of a suitable lathe threads or minute ribbons of metal approximately 0·1 mm. in thickness and 0·5 mm. in width. The method of preparing such shavings is described in detail in a paper on the "Precipitation of Gold from Cyanide Solutions" by myself in *The Mineral Industry,* IV, 331. Theoretically 1 lb. of zinc should precipitate about 6 lb. of gold; practically from 5 oz. to 1 lb. of zinc are required for every ounce of gold recovered. Zinc is sometimes employed in the form of zinc dust instead of shavings. At Deloro, Canada, where zinc dust was used the consumption averaged 0·54 lb. per ounce of fine gold recovered on a month's test. The consumption of zinc in connection with the cyanide process is frequently overestimated. The total production of gold by the cyanide process in 1897 was estimated by G. T. Beilby as follows:[1] Africa, 825,000 oz. of bullion; Australia, 308,000; New Zealand, 263,000; United States, 190,000; India, 18,800; Mexico, 10,200; other countries, 5,000; total, 1,620,000, equivalent to 1,215,000 oz. of fine gold. Estimating the zinc consumption at 1 lb. per fine ounce, a high figure, the total requirements in 1897 would have been only 1,215,000 lb.=607·5 tons.

Miscellaneous Uses of Sheet Zinc.—The production of sheet zinc in the United States is consumed chiefly in the manufacture of miscellaneous articles, such as washboards, linings for refrigerators, floor sheets for stoves to stand upon, covers for preserve jars, bath tubs, etc. Special sheets are prepared for the use of paper and card makers, who require them for glazing purposes. The American rolling mills also have a large trade in the manufacture of plates for the use of etchers and lithographers, and in the art of zincography; there is a considerable consumption of rods and plates of zinc in connection with galvanic batteries. More or less sheet zinc is used in the building trade for chimney flashings, roofing piazzas, etc.

Rolled zinc can be chased, punched and stamped in many useful and ornamental forms required for ceilings, moldings, friezes and other architectural purposes. Some sheet zinc is consumed in lining packing cases for valuable and perishable goods; also for lining coffins. The metallic foundations for cloth-covered buttons are now made almost entirely of stamped zinc. Of the many small articles made of sheet zinc mention of water cans and buckets, sprinkling pots, oil cans, stair treads, coal scuttles and toys illustrates the manifold uses of the metal. It is important to note, however,

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[1] Journ. Soc. Chem. Ind., Feb. 28, 1898.

that in many of these lines of consumption, where durability against weather or some other especial requirement is absent, zinc is not indispensable and merely takes the place of some other metal, wherefore the demand for it is subject to a sharp restriction when the price of zinc rises above a certain figure. This was experienced by the rolling mills in a striking manner during the period of abnormally high prices in 1899.

ZINC CASTINGS.—Zinc is well adapted for ornamental castings, being capable of taking very sharp impressions of the most delicate lines and molds, and small quantities of the metal are employed for that purpose. Many of the statuettes and other ornaments to be seen in house furnishings are composed entirely of zinc, being plated subsequently with silver or copper under the designation of "white bronze." Monuments cast from spelter are made to some extent in the United States, a sand blast being employed to give their surface a pleasing dull finish; such monuments are claimed to be highly durable and capable of very artistic treatment, and are cheaper than stone monuments.

CONSUMPTION OF ZINC IN BRASS MAKING.—After the galvanizing and sheet zinc industries the manufacture of brass is the most important channel of consumption for zinc. Brass is employed chiefly in the form of castings, sheets, wire and tubes. In preparing brass castings the metal previously melted in a crucible is poured directly into the mold, which is formed in a free, fine-grained sand of uniform character and contained in a sectional box of wood or cast iron, of which the parts are held together by clamps. The uses of cast brass are so manifold that it would be futile to undertake to enumerate all of them. The manufacture of electrical apparatus is a highly important channel of consumption. Large quantities of cast brass are employed in naval construction. Bells, gas fittings, cocks and plumbers' supplies, and many small articles required in architecture and building call in the aggregate for a large supply of brass. Sheet brass is employed for the manufacture of plates intended for sheathing purposes, for wire drawing, and for stamped work and jointed tubing. Many useful and ornamental articles which were formerly produced by casting are now more cheaply and expeditiously made by stamping out of sheets of rolled brass. Tubes are made from sheet brass by bending strips of the latter to the proper gauge and shape and soldering the junction. Tubes are also made by drawing down short, thick, cast cylinders of brass to the desired gauge and thinness. The consumption of brass in the form of Muntz's metal, a variety which contains a high percentage of zinc, fell off largely when ships ceased to be sheathed, but a new demand developed in the electrical industry. To some extent the demand for brass, and consequently for the zinc which is consumed in its manufacture, is governed by the same conditions which affect the demand for copper.

In rolling brass the metal is melted and cast into broad flat molds of iron, which are rubbed with oil and powdered with charcoal before being used. The ingots for rolling, termed "strips," are in the cold state passed successively between steel rollers of large size, which squeeze them out and extend them lengthwise. As often as necessary the sheets are annealed in a muffle or reverberatory furnace, being allowed to cool after each annealing. After pickling in acid the sheets are finished by passing through a set of highly polished rolls. Muntz's metal can be rolled hot, wherefore it is more cheaply and expeditiously prepared than ordinary sheet brass.

Use of Zinc for Desilverizing Lead.—Zinc plays a highly important part in the modern metallurgy of lead by virtue of its ability to rob the latter of gold and silver, forming an alloy therewith which can be readily removed. So perfect is this property that the last traces of both gold and silver can be recovered if desirable, although in practice it is not economical to desilverize the lead below a tenor of 0·1 oz. per 2,000 lb. Besides removing the gold and silver, zinc also combines with whatever copper and tellurium may be contained in impure lead, which elements are not completely separated by any process of "softening" or liquation, and thereby enables the production of purer metal than is otherwise obtainable. This property of zinc finds application in the Parkes process of desilverizing lead, which has now almost entirely displaced the older methods of cupellation and Pattinsonizing.

In the Parkes process a certain quantity of zinc in the form of the ordinary slabs is stirred into the gold- and silver-bearing lead in a large iron pot. The zinc combines with the gold and silver, forming a mushy alloy, which is removed by skimming, the lead being left practically free from gold and silver after repeated "zinkings," three to four repetitions being usually necessary. The quantity of zinc required varies according to the purity of the lead and increases with the percentage of silver present. The quantity is computed by Roswag[1] by the formula:

$$Z=10{\cdot}39+0{\cdot}035T=Z^{1}23{\cdot}32+0{\cdot}223T^{1}$$

in which

Z=quantity of zinc required in kilograms per metric ton and Z^{1}=quantity in pounds per 2,000 lb.

T=grams of silver in 100 kg. lead and T^{1}=ounces troy of silver per 2,000 lb.

In practice, lead containing 0·1% Ag requires about 1·34% Zn, and lead

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[1] La Désargentation de Plomb, Paris, 1884, p. 241.

containing 1% Ag requires about 2·5% Zn. By distillation the zinc is driven out of the gold-silver alloy, leaving the gold and silver behind in the retort and restoring the zinc, minus a certain loss, for use again. The actual consumption of zinc in desilvering lead containing 1% Ag is about 0·65% to 1% of the lead desilverized. Although the production of lead has increased largely during the last 10 years the consumption of zinc for its desilverization has not increased because of the great economy in the use of the latter which has been effected by means of the Howard stirrer and press and similar devices.

The zinc employed for the desilverization of lead should be low in iron. Hofman found[1] in experimenting with cheap zinc containing iron, obtained from galvanizing works, that the desilverization process was so retarded and the quantity of impure zinc required was so great, that no saving was effected by the use of the inferior material. Jernegan recorded a similar experience;[2] and Foehr stated that in using a spelter which contained 2·75% Pb, 0·61% Fe, 0·077% Cu, and traces of tin, arsenic, antimony, cadmium, sulphur and carbon, four times the quantity of zinc usually required had to be added in order to effect a proper desilverization.[3]

USE OF ZINC IN GALVANIZING.—The uses of galvanized iron are so manifold and well known that it is unnecessary to refer to them specifically in a treatise of this kind. The subject falls anyway rather into the domain of structural material than into that of the metallurgy of zinc. The zinc smelter is interested in it only so far as it affects the marketing of his product. In general the zinc consumed in galvanizing is employed for the production of a protective coating on iron and steel, which may be applied to any article of such size and shape that it can be dipped into the galvanizing bath. Another limitation is that the article to be galvanized is not to be exposed to the action of liquids or vapors which would corrode the coating of zinc. Galvanizing is employed probably to the largest extent in coating pipes or tubes and sheet iron or steel, plain or corrugated. Galvanized corrugated iron constitutes one of our most valuable building materials.

ZINC DUST.—A considerable quantity of zinc is consumed in the arts in the form of zinc dust, zinc gray, or indigo auxiliary, being known in the trade by all of those terms, which is obtained as a by-product in smelting. Zinc dust is employed chiefly as a reducing material in dyeing, whence its name of "indigo auxiliary." It is employed to a less extent for the preparation of a paint for covering iron, and for the precipitation of gold in the cyanide process for the extraction of that metal.

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[1] Metallurgy of Lead, 5th edition, p. 430.

[2] Trans. Am. Inst. Min. Eng., II, 288.

[3] Berg- u. Hüttenm. Ztg., 1888, p. 28.

Zinc White.—In Europe a considerable quantity of spelter is employed for the production of zinc oxide or zinc white; in the United States that substance is made directly from ore. Zinc white is consumed chiefly as a pigment, either alone or in admixture with white lead or barytes. It has excellent covering capacity and possesses the advantage that it is not discolored by vapors containing sulphureted hydrogen, wherefore it is especially valuable for use inside of such buildings as are lighted by ordinary illuminating gas, which is always likely to contain traces of sulphureted hydrogen. Besides being used as a pigment, zinc white is employed extensively in the manufacture of rubber goods.

Other Uses of Zinc.—Most of the chemical compounds of zinc which are used in the arts are derived from metallic zinc. Of these zinc sulphate and zinc chloride are employed to a considerable extent, the consumption of each amounting to several thousand tons per annum. Zinc sulphate is made in considerable quantity in Germany directly from ore, and in the United States to some extent from a zinky by-product recovered from spelter that has been used for the desilverization of lead. The zinc chloride of commerce is derived entirely from the treatment of metallic zinc with chlorhydric acid.

Limitation of the Use of Zinc.—Notwithstanding the important applications which zinc finds in the arts it cannot be considered an indispensable metal, like copper for example, wherefore its price is limited; that is to say if the price rises above a certain figure the consumption is immediately restricted. This was demonstrated in a striking manner in the United States in 1899 when the price of spelter rose to 7c., New York, and was for a long time higher than 5c. At those figures the consumption of spelter was much restricted, especially in the sheet zinc and galvanizing industries. With respect to the latter there is a certain difference between the price of black, painted corrugated sheets and galvanized sheets at which the consumer gives the preference to the galvanized, but when that difference is exceeded he will take the painted sheets instead. The difference in 1899 exceeded the parity of choice, wherefore the demand for galvanized sheets fell off. In the sheet-zinc trade a high price for zinc leads to the use of other metals as substitutes. In the zinc-white trade it increases the difficulty of competition with white lead, if the price of the latter be not high in proportion, and in any case gives headway to the use of barytes and other inferior substitutes. Except for the manufacture of brass there are few important uses for which zinc is an absolutely indispensable metal, and its price is therefore limited to the point at which consumers will give it the choice in preference to substitutes. With a price for spelter of 4@5c. per lb.

at New York there ought to be a large consumption at an increasing rate corresponding to the industrial development of the country. A price of 5c. per lb. or over cannot be expected except for short periods when there may be a brisk demand and a shortage in the supply; at least not under the existing conditions of abundant ore supply and cheap fuel. The average price for the 10 years, 1891-1900, was 4·36c. at New York, according to the quotations of *The Mineral Industry*, and 4·16c. at London, according to the statistics of Henry R. Merton & Co., reduced to U. S. currency at par of exchange.

IV

STATISTICS OF PRODUCTION AND PRICES.

In a study of the economic conditions of the zinc industry of the world an investigation of the statistics of production and consumption and price is highly important. In the present chapter I have collected all the available statistics of a general character, most of which are derived from the official sources, partly by direct reference to the original authorities, and partly from the compilations of *The Mineral Industry,* which are, however, taken from the various official publications with the exception of the statistics of production in the United States, which are based upon direct reports from the producers. In making this acknowledgment to *The Mineral Industry* I do not, however, seek to throw off responsibility for the accuracy of the work here presented, inasmuch as I have in many cases introduced figures which do not appear in the tables of that publication. Free use has also been made of the excellent and highly valuable statistical compilations of the Metallgesellschaft of Frankfurt am Main, which is represented in the United States by the American Metal Co., Ltd., of New York. It is to be understood, however, that all of the statistics presented in this chapter are taken from the official publications of the various governments unless some other authority is specifically referred to. In a comparatively few instances, where statistics have been lacking or unavailable, estimates have been inserted in order to permit totals to be arrived at. Such esimates are conventionally indicated by an asterisk.

In view of the existence of so many different statistical publications, it would perhaps have been advisable to adopt a particular one as the most authoritative and adhere uniformly to it. That would have involved, however, the repetition of a vast amount of work, besides involving the introduction of various complications, which would have been useless in view of the really close agreement of all of the authorities. Inasmuch as each statistician presents some valuable information which the others do not, it has appeared best to preserve their own statements in spite of some discrepancies (really unimportant) and at the sacrifice of uniformity, considering that the use of any of their figures will not lead to any material error. The zinc industry is to be congratulated upon the possession of statistics of production, consumption and price, which are so complete and authoritative as to be of the highest commercial importance.

PRODUCTION OF ZINC ORE IN EUROPE AND AUSTRALIA.

(In metric tons.)

Year	Algeria	Austria	Belgium	France	Germany	Great Britain	Greece	Italy	N. S. Wales	Russia	Spain	Sweden
1840	....	....	20,482	....	....	....	....	....	....	....	....	....
1841	....	....	18,380	....	....	....	....	....	....	....	....	....
1842	....	....	18,466	....	....	....	....	....	....	....	....	....
1843	....	....	25,668	....	....	....	....	....	....	....	....	....
1844	....	....	22,689	....	....	....	....	....	....	....	....	....
1845	....	....	30,027	....	....	....	....	....	....	....	....	....
1846	....	....	42,365	....	....	....	....	....	....	....	....	....
1847	....	....	47,663	....	....	....	....	....	....	....	....	....
1848	....	....	48,582	....	127,396	....	....	....	....	....	....	....
1849	....	....	49,712	....	128,743	....	....	....	....	....	....	....
1850	....	....	69,501	....	147,840	....	....	....	....	....	....	....
1851	....	....	80,266	....	150,950	....	....	....	....	....	....	....
1852	....	....	78.345	....	181,048	....	....	....	....	....	....	....
1853	....	....	80,092	....	162,383	....	....	....	....	....	....	....
1854	....	....	79,472	....	178,929	....	....	....	....	....	....	....
1855	....	5,465	81,273	....	214,365	....	....	....	....	....	....	....
1856	....	5,236	83,274	....	226,625	....	....	....	....	....	18,293	....
1857	....	5,916	76,236	....	218,890	....	....	....	....	....	....	....
1858	....	5,304	75,392	....	244,363	....	....	....	....	....	....	....
1859	....	5,382	70,390	....	278,277	....	....	....	....	....	....	....
1860	....	....	66,141	....	303,596	15,807	....	162	....	....	....	....
1861	....	....	73,155	....	328,682	16,029	....	168	....	....	24,744	7,209
1862	....	....	74,008	....	333,598	7,620	....	157	....	....	41,104	8,518
1863	....	....	61,767	....	291,693	13,153	....	265	....	....	48,124	9,033
1864	....	....	58,066	....	313,299	15,310	....	202	....	....	80,222	14,248
1865	....	....	56,185	....	335,348	18,135	....	732	....	....	70,158	25,949
1866	....	....	54,516	....	353,149	12,979	....	4,492	....	....	73,423	20,841
1867	....	....	58,046	....	368,929	13,710	....	6,443	....	....	86,822	18,790
1868	....	....	68,699	....	369,874	12,990	....	51,012	....	....	131,407	23,831
1869	....	16.564	66,918	....	405,025	15.788	....	80,524	....	....	113,485	31,741
1870	....	13,694	57,099	....	366,780	13,809	....	92,833	....	....	113.583	28,146
1871	....	....	61,129	....	335,173	18,027	....	56,426	....	....	107,380	32,172
1872	....	....	55,537	....	419,543	18,847	....	80,861	....	....	89,371	33,236
1873	....	14,642	42,582	1,279	444,950	16.231	....	79,036	....	....	101,010	27,444
1874	....	21,147	43,299	3.848	451,222	17,106	....	64,716	....	....	106,477	28,198
1875	....	25,728	42,504	4,088	467 953	24,371	....	61,968	....	....	100,174	31,642
1876	....	26.458	37,713	....	533,559	24,000	....	66,034	....	....	107,063	35,523
1877	....	24,002	44,987	....	577,312	24,806	....	88,844	....	....	70,951	39,966
1878	....	33,387	45,293	....	597,193	25,855	....	62,703	....	....	71,558	40,795
1879	....	19.389	42,689	....	589,546	22,564	....	73,411	....	....	60,980	43,817
1880	....	21,564	38,805	....	632,895	28,006	....	85,289	....	....	50,521	43,460
1881	....	27,340	23,553	....	659,530	36,109	....	72,176	....	....	42,911	43,811
1882	....	25,300	20,443	8,372	694,711	33,069	....	91,366	....	97,000	57,353	46.255
1883	....	28,749	20,738	7,156	677,794	30,215	43,731	100.574	....	....	54,193	45,347
1884	....	29,454	27,606	3,120	632,040	25,982	....	104,974	....	....	49,838	44,893
1885	....	23.598	18,185	5,078	680,654	25,072	....	107,887	....	....	49,509	48,589
1886	....	21,320	19,042	11,103	705,177	23,535	....	107,548	....	38,182	39,810	49,571
1887	....	21,099	20,879	13,321	900,712	25,862	42,258	93,143	....	....	69,012	46,241
1888	8,521	26,312	24,537	20,702	677,761	26,841	43,405	87,310	....	....	74,353	49,972
1889	12,556	30,096	21,184	34,290	708,829	23,582	33,025	97,059	....	....	71,774	59,381
1890	13,091	32.632	15,410	47,540	759,437	22,402	33,054	110,926	....	44,125	81,398	61,843
1891	13,636	28,828	14,280	56,300	793,544	22,580	28,344	120,685	....	47,390	....	61,591
1892	21,907	33,944	12,260	69,236	800.237	27,311	27,695	129,731	....	....	74,265	54,981
1893	24.400	30,531	11,310	74,400	787,911	23,880	22,589	132,767	....	....	62,616	46,623
1894	29,703	28,491	11,585	80,065	728,616	22,170	20,830	132,777	....	62,420	58,964	47,029
1895	14,300	25,862	12,230	72,989	706,423	17,758	24,031	121,197	....	57,213	54,109	31,349
1896	17,587	26.887	11,630	81,346	729,942	19,629	22,700	118,171	....	59,680	64,828	44,041
1897	32,269	27,463	10,954	83,044	663,850	19,587	30,906	122,214	29,303	54,524	73,848	66,636
1898	29,800	27,395	11,475	85,550	641,706	23,929	32,045	132,099	39,561	....	99.836	61,627
1899	42,970	36,100	13,190	84,813	664,536	23,505	22,907	150,629	50,680	....	119.710	65,159
1900	30,281	38,243	11,715	67,059	639,215	25,071	18,505	139,679	20.593	....	86.158	61,034

Besides the productions reported in the above table there was an output in Bosnia of 69 metric tons in 1882, 697 in 1883, 20 in 1889, 61 in 1890, 47 in 1891, and 16 in 1892. Norway produced 300 tons of zinc-lead ore in 1882, 200 in 1883, 571 in 1884, 300 in 1885, 1,540 in 1888, 3,278 in 1889, 3,941 in 1890, 498 in 1891, 576 in 1892, 200 in 1894, 750 in 1896, 908 in 1897 and 320 in 1898.

In comparing the zinc ore statistics of Europe allowances have to be made for differences in the method of computation employed by the statisticians of the several countries. All of them include both blende and calamine, but some report the production of raw ore and some the production after roasting and calcination, which processes are frequently performed at the mines before shipment of the ore.

The statistics previous to 1862 under the caption "Germany" represent only the production of Prussia; the output of the mines outside of that Kingdom was insignificant at that period, however, and the statistics for Prussia from 1852 to 1862 are practically representative of the production of entire Germany.

There was a small production of zinc ore in Hungary between 1863 and 1884, of which there are no statistics; this ore was smelted at works in the Kingdom, producing about 500 tons of spelter per annum (vide statistics of spelter production).

The above table is deficient in failing to take into account the small production of zinc ore in Turkey and the considerable production in Tunis[1] and the neutral territory of Moresnet. The famous mines of the Vieille Montagne are situated in the last; their output is not included in the statistics of any official publication.

Many of the European countries which collect statistics of their zinc ore production report them by provinces and districts, a valuable system, but one that is rather too minute for a general summary of this kind. Other countries report the production classified as blende and calamine. Such a one is Belgium, the statistics of which are presented in the subjoined table. It will be observed therefrom how the production of calamine attained a good deal of importance before blende was mined at all, how subsequently the production of blende increased while that of calamine diminished, and how finally with the exhaustion of the mines, the production of both kinds dwindled down to insignificant proportions. These statistics go so far back that they present a very interesting record of the zinc mining industry of Belgium.

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[1] Tunis produced 4,400 tons (1,000 kg.) of zinc ore in 1893; 31,000 in 1894; 14,800 in 1895; 12,100 in 1896.

PRODUCTION OF ZINC ORE IN BELGIUM.

(In metric tons.)

Year	Blende	Calamine	Total	Year	Blende	Calamine	Total
1840	nil	20,482	20,482	1870	15,783	41,316	57,099
1841	"	18,380	18,380	1871	19,970	41,159	61,129
1842	"	18,466	18,466	1872	20,623	34,914	55,537
1843	"	25,668	25,668	1873	13,952	28,630	42,582
1844	"	22,689	22,689	1874	17,087	26,212	43,299
1845	264	29,763	30,027	1875	18,750	23,754	42,504
1846	2,461	39,904	42,365	1876	21,739	15,974	37,713
1847	1,786	45,877	47,663	1877	26,310	18,677	44,987
1848	4,378	33,204	48,582	1878	27,134	18,159	45,293
1849	7,442	42,270	49,712	1879	23,229	19,460	42,689
1850	7,308	62,193	69,501	1880	23,080	15,735	38,815
1851	14,183	66,083	80,266	1881	8,169	15,384	23,553
1852	10,442	67,903	78,345	1882	2,171	18,272	20,443
1853	13,640	66,452	80,092	1883	3,814	16,924	20,738
1854	11,333	68,139	79,472	1884	12,057	15,549	27,606
1855	10,905	70 368	81,273	1885	11,597	6,588	18,185
1856	11,418	71,856	83,274	1886	12,718	6,324	19,042
1857	10,487	65,749	76,236	1887	12,405	8,474	20,879
1858	19,467	55 931	75,398	1888	12,370	12,167	24,537
1859	13,751	56,639	70,390	1889	10 248	10,936	21,184
1860	17,284	48,857	66,141	1890	10,370	5,040	15,410
1861	17,267	55,888	73,155	1891	10,200	4,080	14,280
1862	18,884	55,124	74,008	1892	8,250	4,010	12,260
1863	14,899	46,868	61,767	1893	7,300	4,010	11,310
1864	16,309	41,757	58,066	1894	7,570	4,015	11,585
1865	14,657	41,528	56,185	1895	8,080	4,150	12,230
1866	15,734	38,782	54,516	1896	7,070	4,560	11,630
1867	16,594	41,452	58,046	1897	6,804	4,150	10,954
1868	16,485	52,214	68,699	1898	7,350	4,125	11,475
1869	17,334	49,584	66,918	1899	9,460	3,730	13,190

There are few countries in Europe which smelt the whole of their own output of ore, the latter being shipped naturally to the most convenient market, often in some other country, while the domestic smelters draw their supplies from the most convenient and cheapest sources at home or abroad. This is shown most strikingly in the subjoined table of the exports and imports from and into France, from which it appears that a large proportion of the French production of zinc ore is exported while the supply is made good by approximately equivalent importations.

FRENCH IMPORTS AND EXPORTS OF ZINC ORE.

(In metric tons.)

Year	Imports	Exports	Year	Imports	Exports	Year	Imports	Exports
1892	....	....	1895	41,622	61,291	1898	60,481	60,664
1893	....	....	1896	50.899	62,415	1899	78,192	76,104
1894	34,955	58,281	1897	58,074	79,909	1900	66,178	54,665

The relative importance of the production of blende and calamine is also shown in the subjoined statement of the output of the mines of Upper Silesia, which is the most important single zinc producing district of Europe.

PRODUCTION OF THE ZINC MINES AND SMELTERIES OF UPPER SILESIA. *a*

(In metric tons.)

Year	Blende	Calamine	Total zinc ore	Iron pyrites *b*	Iron ore *b*	Lead ore	Zinc	Sheet zinc	Zinc white, etc. *f*	Average value of zinc per ton
1861	....	....	283,487	....	....	3,149	42,033	8,406	968	312
1862	....	....	279,722	....	....	4,855	41,700	9,165	969	315
1863	....	....	234,744	....	....	8,580	40,600	8,975	1,180	314
1864	....	....	237,540	....	....	10,973	38,573	7,430	833	396
1865	....	....	268,384	....	....	6,164	35,430	9,164	834	382
1866	....	....	286,166	....	....	8,767	34,864	6,016	756	392
1867	....	....	299,424	....	....	9,912	36,832	5,084	753	389
1868	....	....	290,362	....	....	11,860	37,631	8,084	719	378
1869	....	....	324,669	....	....	13,123	37,917	11,762	280	382
1870	....	....	310,909	....	....	16,010	36,516	10,047	346	349
1871	....	....	269,626	....	....	14,339	32,091	13,452	488	357
1872	....	....	332,066	128	15,507	14,610	33,065	13,854	386	408
1873	....	....	367,582	355	8,686	14,589	36,382	13,092	692	478
1874	....	....	361,747	1,101	6,746	16,866	41,181	16,121	842	423
1875	....	....	377,567	1,713	8,598	17,871	42,855	15,746	937	454
1876	....	....	442,837	2,253	6,055	19,105	49,376	18,612	795	431
1877	....	....	472,422	2,074	10,546	19,370	57,478	18,699	925	368
1878	57,782	432,678	490,460	2,891	15,556	20,273	59,789	19,031	931	322
1879	62,291	430,041	492,332	3,213	15,908	19,064	63,564	19,805	893	300
1880	81,547	445,407	526,954	4,028	19,608	17,760	66,044	16,732	916	340
1881	99,809	444,281	544,090	2,578	28,795	21,078	67,771	24,517	1,008	304
1882	120,291	459,056	579,347	2,840	35,867	24,230	69,992	20,682	3,716	316
1883	122,799	505,185	627,984	2,131	36,178	24,810	71,468	24,846	3,818	283
1884	143,344	445,985	589,329	1,457	46,858	25,861	76,897	25,474	3,778	267
1885	159,276	447,330	606,606	1,585	54,780	26,313	78,477	25,347	3,707	253
1886	172,780	398,490	571,270	2,083	53,112	29,286	82,712	25,066	3,746	256
1887*c*	193,826	611,535	805,361	2,930	57,559	28,580	82,640	29,141	3,128	275
1888	212,264	319,316	531,580	1,583	33,344	29,601	84,777	25,821	2,811	324
1889	246,955	225,705	572,660	1,971	20,268	32,146	86,947	32,562	922	359
1890*c*	261,921	368,495	630,416	1,949	11,287	32,498	88,699	32,547	896	440
1891*c*	271,277	391,891	663,168	2,076	8,088	28,716	88,420	37,669	1,151	441
1892*d*	291,617	368,230	659,847	2,520	9,371	29,049	88,175	33,266	895	386
1893*d*	287,375	348,654	636,029	2,104	7,083	30,825	91,659	35,187	207	324
1894*d*	251,040	323,295	574,335	2,874	5,808	33,898	92,546	34,518	1,250	284
1895	267,673	273,151	540,824	2,316	7,920	31,927	95,430	35,676	1,435	270
1896*e*	275,514	263,338	538,852	3,543	7,556	31,096	98,323	39,488	1,105	299
1897	270,426	240,260	510,686	4,825	12,814	35,847	95,547	36,618	1,336	328
1898	289,684	219,538	509,222	7,306	11,478	42,494	99,011	39,863	1,452	370
1899*e*	343,677	184,637	528,314	5,716	15,918	40,828	100,113	35,646	1,431	466
1900	312,486	190,304	502,790	6,965	24,143	42,098	102,093	38,469	1,375	389

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a For the years 1861 to 1885, both inclusive, from Die Bergwerks- und Hüttenverwaltungen des Oberschlesischen Industriebezirks, Kattowitz, 1892; for 1886 et seq. from Statistik des Oberschlesischen Berg- und Hüttenmännischen Vereins.

b The iron pyrites and iron ore here reported are the production of the zinc mines and do not include the output of any of the Silesian iron mines proper.

c The statistics for 1887, 1890 and 1891 include 252,747. 25,000 and 67,500 tons, respectively, of calamine recovered from the old dumps of the Scharley mine.

d The statistics for 1892, 1893 and 1894 include, respectively, 30,935, 53,586 and 39,810 tons of calamine recovered from the dumps of the Scharley and Paul-Richard mines.

e The statistics for 1896 and 1899, respectively, include 7,500 and 189 tons of calamine from the Scharley and Paul-Richard dumps.

f Includes zinc white, zinc gray and residues.

Blende was first produced in Upper Silesia in 1870, and the increased supply of ore which was thereby afforded had a powerful effect upon the zinc industry at that time. The sulphide production increased rapidly, and during the last decade it became the more important source of zinc in Upper Silesia, its tenor in metal being much higher than that of the calamine;

since 1895 its tonnage also has been the greater. The increasing ratio of blende to calamine in the ore production of Upper Silesia now promises to lead to some more important changes in the metallurgical practice of that district, to which reference has been made in a previous chapter.

The average value in marks per 1,000 kg. of the zinc ore product of Upper Silesia since 1882 is summarized in the following table:

Year	Calamine	Blende	Year	Calamine	Blende	Year	Calamine	Blende
1883	5·41	12·67	1889	10·43	31·27	1895	5·35	17·08
1884	4·43	13·25	1890	11·70	45·50	1896	7·80	28·35
1885	3·67	11·35	1891	12·41	47·56	1897	8·87	29·98
1886	3·44	11·49	1892	7·90	38·68	1898	10·97	41·44
1887	5·76	14·72	1893	5·57	21·57	1899	11·50	58·47
1888	8·74	24·86	1894	3·05	17·52	1900	7·96	40·24

The above table is computed on the basis of the ore mined, leaving out of account the quantity and value of the calamine recovered from old dumps, which if included would reduce the average somewhat.

GERMANY: IMPORTS AND EXPORTS OF ZINC ORE.

(In metric tons.)

Year	Imports	Exports	Year	Imports	Exports	Year	Imports	Exports
1880	19,132	12,798	1887	11,232	20.971	1894	14,712	35,682
1881	15,461	15,610	1888	8,901	23,683	1895	25,818	31,031
1882	24,567	8,847	1889	26,812	20,957	1896	21,493	37,959
1883	19.651	11,923	1890	38,098	16,542	1897	24,735	30,047
1884	17.078	12,217	1891	37,762	22,123	1898	48,050	30,408
1885	27.180	13,429	1892	41.558	24,475	1899	57,880	25,192
1886	19,717	14,414	1893	23,883	25,059	1900	68,982	34,941

EXPORTS OF ZINC ORE FROM SPAIN.

(In metric tons.)

Year	Quantity	Year	Quantity	Year	Quantity
1871	33,807	1881	39,774	1891	39,581
1872	43,503	1882	38,701	1892	39,574
1873	46,562	1883	45,556	1893	30,814
1874	66,169	1884	35.580	1894	34,119
1875	42,778	1885	36,045	1895	29,360
1876	60,081	1886	29,873	1896	36,656
1877	65,968	1887	27,253	1897	41,040
1878	37.332	1888	32,004	1898	65,573
1879	29,625	1889	36,108	1899	95,088
1880	36,414	1890	47,025	1900	*a* 60,970

a The exportation in 1900 is given as reported by the Revista Minera, of Madrid. The figures for the previous years are from the official publication.

Outside of Europe itself and the immediately adjacent colonies in Africa, the smelters of Great Britain and the Continent have derived but little ore until within the last five years, when some important supplies have been obtained from New Jersey and from Leadville, Colo., and Broken Hill, New South Wales, the ore from the last two sources being a product dressed from the mixed sulphide ore which exists in vast quantity at each of those places. The exports of zinc ore from New South Wales, which practically represent the production of the Broken Hill mines, are stated in the following table:

EXPORTS OF ZINC ORE FROM NEW SOUTH WALES.

(In metric tons.)

Year	Quantity	Value	Year	Quantity	Value
1897	29,303	£23,688	1899	50,680	£49,207
1898	39,561	28,941	1900	20,593	44,187

The production of spelter by distillation from this ore has recently been begun in Australia.

PRODUCTION OF ZINC ORE IN THE UNITED STATES.

There are no complete statistics of the production of zinc ore in the United States, but the output of the most important districts, namely, New Jersey and Kansas-Missouri, is reported satisfactorily, the former by the New Jersey Geological Survey and the latter by the local journals, the reports of which are summarized regularly by *The Mineral Industry*. The statistics of the production of those two districts are presented in the following table:

PRODUCTION OF ZINC ORE IN THE MOST IMPORTANT DISTRICTS OF THE UNITED STATES.

(In tons of 2,000 lb.)

Year	*a* New Jersey	*b* Kansas Missouri	Year	*a* New Jersey	*b* Kansas Missouri	Year	*a* New Jersey	*b* Kansas Missouri	Year	*a* New Jersey	*b* Kansas Missouri
1881	55,079	58,200	1886	49,142	85,400	1891	85,157	145,550	1896	78,490	153,082
1882	44,955	60,300	1887	56,246	98,300	1892	86,574	154,800	1897	86,210	177,975
1883	62,814	63,700	1888	51,942	102,350	1893	62,554	139,770	1898	111,349	235,123
1884	44,905	74,300	1889	62,892	106,750	1894	66,508	139,779	1899	194,881	256,456
1885	43,149	74,000	1890	55,572	122,850	1895	*c*.....	144,487	1900	221,053	242,500

a From reports of the New Jersey Geological Survey. *b* As reported by The Mineral Industry. *c* No statistics were collected for 1895.

The only statistics of zinc ore production in other parts of the United States, which are available, are very incomplete. According to *The Mineral Industry*, vol. II, the production of zinc ore in Virginia and Tennessee was 8,420 tons in 1887, 11,500 in 1888, 12,906 in 1889, 14,969 in 1890, 20,287 in 1891, 20,295 in 1892, and 21,000 in 1893. According to recent volumes of *The Mineral Industry* there was a production of 3,799 tons of blende in Tennessee in 1899 and 3,968 in 1900. There are no recent statistics of the production in Virginia, but the output of that State is probably not more than 15,000 tons per annum, all of which is calamine assaying about 40% Zn. The production of zinc ore in Wisconsin, chiefly blende, was reported to have been 15,000 tons in 1900. About 2,000 tons per annum of low grade calamine has lately been produced in Iowa. Small lots of zinc ore are received occasionally from Arkansas and New Mexico. Since 1898 there has been a large and increasing production of zinc ore in Colorado. The output of that State in 1900 was reported by H. A. Lee, State Commissioner of Mines, as having amounted to 77,984 tons of ore, averaging 42% Zn. That total undoubtedly includes, however, the ore that was consumed for the manufacture of zinc-lead pigment at Cañon City, Colo., and for the manufacture of zinc white at Mineral Point, Wis.[1] During 1901 the Leadville mills produced 23,261 tons of zinc ore, of which about 80% was exported. The Leadville product averages approximately 45% Zn, 6% Pb and 12% Fe. Some which is concentrated at Denver from the same kind of crude ore, by means of the Wetherill magnetic machines, averages 50% Zn, 1% Pb and 10% Fe. Besides from Leadville there are also shipments of zinc ore from Rico, Kokomo, Silver Plume, Montezuma, Creede and elsewhere in Colorado. The ore shipped from Creede averages about 56% Zn, 5% Pb and 1% Fe.

EXPORTS OF ZINC ORE FROM THE UNITED STATES.

Year	Short Tons	Total Value	Value per Ton	Metric Tons
1897	9,251	211,350	$22·85	8,391
1898	11,782	299,870	25·54	10,686
1899	28,221	725,944	25·72	25,602
1900	40,561	1,133,663	27·95	36,796

There were occasional exports of zinc ore from the United States previous to 1897, but they were not reported separately in the official statistics. The quantity was not large in any year.

Of the exportation of ore reported in the above table, 7,040 tons ($213,205) in 1899 and 12,794 tons ($373,528) in 1900 were despatched through the ports of Galveston, Tex., and New Orleans, La., the remainder going out chiefly via New York and Philadelphia. The exports through New York and Philadelphia are of New Jersey willemite; those from Galveston and New Orleans are chiefly of Colorado ore, but include occasional lots from the Joplin district.

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[1] It appears as if there might be a mistake in the figures reported for Colorado in 1900.

WORLD'S PRODUCTION OF ZINC.

(In metric tons.)

Year	Austria	Belgium	France	Germany	Great Britain	Netherlands	Russia	Spain	United States	Total
1845	388	7,221	....	17,100	....	....	3,569	....	....	....
1846	413	8,963	....	22,260	....	....	....	....	....	....
1847	352	10,241	....	22.400	....	....	....	....	....	....
1848	1,395	10,850	....	20,190	1,000	....	4,000	....	....	....
1849	....	13,579	....	26,270	....	....	....	....	....	....
1850	....	14,808	....	28,670	....	....	2,606	....	....	....
1851	....	15.250	....	30,620	....	....	....	....	....	....
1852	....	16,672	....	34,721	....	....	....	....	....	....
1853	....	18,817	....	34,673	....	....	....	....	....	....
1854	*1,500	19,553	....	36,873	*1,000	....	....	....	....	....
1855	....	20,633	....	38,254	....	....	1,107	....	....	....
1856	866	22,900	....	38,326	....	....	....	....	....	*a* 71,000
1857	1,055	24,526	....	43,611	....	....	....	....	....	*a* 79.000
1858	1,580	34,191	....	52,777	3,522	....	....	....	....	....
1859	1,246	28,631	....	49,282	3,757	....	....	....	....	....
1860	1,301	22,027	....	55,346	4,428	....	1,838	....	....	....
1861	....	28,150	....	58,573	4,487	....	....	....	....	....
1862	....	25,861	....	59,767	2,186	....	....	....	....	*a* 100.000
1863	....	28,978	....	60,315	3,897	....	....	....	....	....
1864	....	30,718	....	59,248	4,106	....	....	....	....	....
1865	....	34,244	....	56,490	4,533	....	3.089	....	....	....
1866	....	34,659	....	60,221	3,244	....	....	....	....	....
1867	....	38,684	....	63,874	3,811	....	....	....	....	*a* 110,000
1868	....	44,347	....	66,132	3,774	....	....	....	....	....
1869	1,866	47,407	....	69,851	4,573	....	....	....	....	....
1870	1,908	45,754	....	63,980	4,000	....	3,780	....	....	*a* 130,000
1871	....	45,623	....	58,297	5,047	....	....	3,166	....	....
1872	....	41,838	....	58,386	5,276	....	....	2,940	....	....
1873	2.285	42,314	12,627	62,755	4,544	....	3,378	2,993	6,664	137,560
1874	2,818	46,088	12,783	70,426	4,543	....	4,128	3,295	9,074	153,155
1875	2,940	49,960	13,739	74,337	6,823	....	3,988	3,831	13,914	169,532
1876	3,979	47,981	*14,000	83,227	6,750	....	4,626	4,349	14,520	179,432
1877	4,519	55,923	*15,000	94,996	6,384	....	4,635	3,780	15,281	200,518
1878	3,623	61,227	*16,000	94,953	6,412	....	3,646	3,775	17,242	206,878
1879	3,280	57,157	*17,000	96,756	5,645	....	4,321	3,800	19,057	207,016
1880	3,756	59,880	*18,000	99,646	7,279	....	4,390	4,221	21,080	218,252
1881	4,119	69,800	18,509	105.478	23,660	....	4,542	7,032	27,225	260,365
1882	4,791	72,947	18,525	113.418	26,501	....	4,462	7,310	30.642	278,596
1883	4,539	75,366	15,915	116,854	29,630	....	3,809	6.843	33,375	286,331
1884	4,536	77,487	16,884	125,276	30,238	....	4,313	4,295	35,585	298,614
1885	3,948	80,298	15,108	129,098	24,690	....	4,579	4,247	36,921	298,889
1886	3,843	79,246	16,132	130,854	21,572	....	4,190	4,327	38,696	298,860
1887	3,609	80,468	16,712	130,444	20,228	....	3,621	5,349	45,682	306,113
1888	4,001	80,675	16,960	133,224	27,214	....	3,869	5,117	50,731	321,791
1889	4,840	82,526	17,982	135,974	31,302	....	3,681	5,640	53,414	335,359
1890	5,449	82,701	19,372	139,266	29,614	....	3,768	5,919	61,111	347,200
1891	5,006	85,999	20,596	139,353	29,883	....	3,675	5,656	72,836	363,004
1892	5,237	91.546	20,680	139,938	30,798	....	4,369	5,925	76,279	374,772
1893	5,870	95,665	22,419	142,956	28,829	*2,743	4,522	5,752	69,159	377,915
1894	6,810	97,041	23,387	143,577	32,578	*3,565	5,014	5,100	67,135	384,207
1895	6,456	107,664	24,200	150,286	29,967	4,266	5,029	5,845	79,462	413,175
1896	6,888	113,361	35,585	153,082	25,278	4,847	6,257	6,133	74,280	425,711
1897	6,236	116,067	38,067	150,739	23,805	6,706	5,868	6,244	91,070	444,802
1898	7,302	119,067	37,155	154,867	28,387	6,808	5,664	6,031	103,514	468,795
1899	7,192	122,843	39,274	153,155	32,222	6,335	6,325	6,184	117,644	491,174
1900	6,742	119,317	36,325	155,790	30,309	*6,500	5,970	5,785	111,794	478,532

* Estimated. *a* According to F. Laur, Bull. de la Soc. de l'Ind. Minerale, 1874, p. 395 et seq.

The consumption of spelter in Europe did not attain much consequence previous to 1840. The statistics of production previous to 1845 are very incomplete. Germany produced 10,000 tons of spelter in 1838, 10,980 in 1840 and 18,000 in 1843. Austria produced 105 tons in 1840, and Russia made 2,739 tons in the same year. In 1846 the production of zinc in

Europe amounted to 40,000 tons, rising to 50,000 tons in 1850 and 62,000 tons in 1853. In 1901 the world's production of spelter in tons of 2,240 lb., according to Henry R. Merton & Co., was as follows, the figures in brackets being the corresponding ones for 1900: Belgium, Holland, Rhenish Prussia and Westphalia, 199,285 (186,320); Silesia, 106,385 (100,705); Great Britain, 29,190 (29,830); France and Spain, 27,265 (30,620); Austria and Italy, 7,700 (6,975); Poland, 5,935 (5,875); United States 122,830 (110,465); total, 498,590 (470,790).

The above table of spelter production of the world does not include the zinc contents of white vitriol made directly from ore in Germany; nor of zinc white made directly from ore in the United States. It does not include, moreover, the zinc gray (zinc dust) recovered as a by-product and marketed in that form by the smelters of Upper Silesia, and presumably does not include such of the same product as is marketed by the smelters of Rhenish Prussia, Westphalia and Belgium. The statistics of spelter production are transcribed from the official reports of the respective countries with the exceptions noted below, in connection with which some additional information is presented.

United States.—The statistics used are those of *The Mineral Industry* except for the years 1895 and 1896, for which the statistics of the American Metal Co. are adopted, the figures of *The Mineral Industry* for those years being probably incorrect. There is also ground for suspicion as to *The Mineral Industry* figures for 1892 and 1893, but they have been permitted to stand, the errors (if they exist) being less important than those for 1895 and 1896.

Great Britain.—Statistics for Great Britain subsequent to 1880 have been computed from the reports of Henry R. Merton & Co. of London, the British bluebooks giving only the quantity of zinc estimated as obtainable in smelting the ore produced in the United Kingdom. The inclusion of the metal derived from foreign ores explains the apparently great increase in the British production from 1880 to 1881. The spelter product of ore mined in the United Kingdom is reported by the British Government for the years subsequent to 1880 as follows:

Year	Tons	Year	Tons	Year	Tons	Year	Tons
1881	15,192	1886	9.036	1891	9,037	1896	7,224
1882	16,344	1887	9,920	1892	9,496	1897	7,162
1883	13,826	1888	10,166	1893	9.585	1898	8,711
1884	10,081	1889	9,546	1894	8,260	1899	8,837
1885	9,938	1890	8,692	1895	6,760	1900	9,211

Germany.—Previous to 1862 the production of Germany is assumed as identical with the production of Prussia as reported in the statistics for that Kingdom. By far the larger part of the spelter production of Germany is due to the works of Upper Silesia, the output of which has been stated in a previous table. Practically the entire remainder of the German production is made in Rhenish Prussia and Westphalia. The German output of spelter in 1901 was 166,283 tons.

Hungary.—Hungary produced spelter between 1863 and 1884 as follows:

Year	Tons	Year	Tons	Year	Tons	Year	Tons	Year	Tons
1864	116	1868	323	1872	557	1876	567	1880	554
1865	204	1869	458	1873	616	1877	66	1881	612
1866	375	1870	415	1874	622	1878	418	1882	605
1867	348	1871	463	1875	513	1879	13	1883	205

Netherlands.—The production of spelter in Holland has been computed from Merton's reports, there being no official statistics for that Kingdom.

Italy.—A small production of spelter in 1900 has not been included.

Russia.—The production in 1899 and 1900 is entered as reported by Henry R. Merton & Co., the official statistics for those years not having been published yet. The entire production of zinc in Russia is derived from the Kingdom of Poland, where there are only two producers, the reports for which by the Messrs. Merton are always practically the same as the official figures.

Comparison of Statistics of Production.

The statistics of total zinc production of the world compiled in the above table compare with those of Henry R. Merton & Co., which are adopted by the Metallgesellschaft, as follows:

Year	Official	Merton	Year	Official	Merton
1881	260,365	....	1891	363.004	362.204
1882	278 596	....	1892	374,772	372.900
1883	286 331	285,036	1893	377,915	378,093
1884	298,614	299.420	1894	384.207	380,877
1885	298 889	300 200	1895	413,175	416.621
1886	298.860	298 604	1896	425,711	424.141
1887	306.113	307,020	1897	444,802	443,302
1888	321 791	323.397	1898	468,795	469.031
1889	335.359	335.450	1899	491,331	490,205
1890	347,200	348,585	1900	478,532	478,323

The agreement of the above compilations is sufficiently close to pass without comment. It is very complimentary to the statisticians of Henry R. Merton & Co. and the American Metal Co., who make a practice of bringing out their figures within a few months after the termination of each year and apparently sacrifice nothing in accuracy for the sake of commercial timeliness. Such differences as appear in the general table are due partially to discrepancies in the statistics of the production in the United States, for which there are three authorities, whose reports are compared in the following table:

PRODUCTION OF SPELTER IN THE UNITED STATES.

(In tons of 2 000 lb.)

Statisticians	1888	1889	1890	1891	1892	1893
American Metal Co.	56 000	59.269	66.496	79 632	86 122	78,343
The Mineral Industry.	55 903	58 860	67.342	80.262	84.082	76.255
U. S. Geological Survey. . . .	55 903	58 860	63 683	80.873	87 260	78.832

Statisticians	1894	1895	1896	1897	1898	1899
American Metal Co.	72 138	87 591	81,878	98.792	114 682	129.758
The Mineral Industry.	74,004	81.858	77.637	100 387	114,104	129.675
U. S. Geological Survey. . . .	75.328	89 686	81.499	99 980	115,399	129 051

The statistics of the American Metal Co. for 1900 were 123,721 tons; of The Mineral Industry, 123,231; and of the U. S. Geological Survey, 123.886. The production of the United States in 1901, according to the U. S. Geological Survey and The Mineral Industry, was 140,822 tons.

It appears from the above comparison that the statistics of *The Mineral Industry* were probably about 2,000 tons too low in 1892 and 1893 and were a good deal too low in 1895 and 1896, this conclusion being indicated by the close agreement of the reports for those years by the American Metal Co. and the U. S. Geological Survey.

PRODUCTION OF SPELTER BY STATES.

(In tons of 2.000 lb.)

Year	N. Jersey Penn.	Virginia Tennessee	Total East	Indiana	Illinois	Kansas	Missouri	Total West	Grand Total
1882	5.698		5.698	. . . .	18.201	7.366	2.500	28.067	33,765
1883	5.340		5.340	. . . .	16,792	9.010	5,730	31.532	36.872
1884	7.861		7.861	. . . .	17,594	7.859	5,230	30,683	38,544
1885	8.082		8.082	. . . .	19,427	8,502	4.677	32 606	40.688
1886	6,762		6,762	. . . .	21.077	8,932	5,870	35.879	42,641
1887	7,446		7,446	. . . .	22.279	11,955	8,660	42,894	50,340
1888	9,561		9.561	. . . .	22,445	10,432	13,465	46,342	55.903
1889	10,265		10.265	. . . .	23,860	13.658	11,077	48,595	58,860
1890	9,114		9,114	. . . .	26.243	15,199	13,127	54,569	63,683
1891	8,945	4.217	13,162	. . . .	28,711	22,747	16,253	67,711	80.873
1892	9,582	4,913	14,495	. . . .	*a* 31,383	24,715	16,667	72.765	75,328
1893	8,802	3.882	12.684	. . . .	*a* 29,596	22,815	13,737	66.148	89.686
1894	7,400	1,376	8,776	2,173	26,799	25,588	11,992	66.552	81 499
1895	9,484	3,697	13,181	3,736	31,996	25,775	14,998	76.505	99 980
1896	8,139	2.427	10,566	1,595	34,578	20,759	14.001	70,933	74,399
1897	7,218	3,365	10,583	1,506	36,370	33,396	18,125	89.397	1387.22
1898	8,631		8,631	2,654	44,449	40.132	19,533	106,768	14,115
1899	8,805		8.805	4,020	46,098	52,021	18,107	120,246	129.051
1900	8,259		8,259	. . . .	*a* 38,750	62,136	78.806	115,627	123,886

a Including Indiana

The statistics of the U. S. Geological Survey make a division of the production of spelter in the United States according to districts, which is useful, although no separation is made between Illinois and Indiana. It may be arrived at with sufficient accuracy, however, by taking for Illinois the difference between the total for the two States as reported by the Geological Survey and the production of Indiana as reported by the American Metal Co. This has been done in the above table.

PRODUCTION OF ZINC OXIDE IN THE UNITED STATES.

Year	QUANTITY		VALUE F.O.B. WORKS		TENOR IN ZINC AT 80%	
	Short Tons	Metric Tons	Total	Per Short Ton	Short Tons	Metric Tons
1880	10.107	9.172	763,738	$75·38	8 086	7 338
1881	10,000	9,083	700,000	70·00	8,000	7.266
1882	10 000	9,083	700,000	70·00	8,000	7 266
1883	12 000	10.899	840,000	70·00	9,600	8.719
1884	13.000	11,797	910 000	70·00	10 400	9.438
1885	15.000	13,625	1.050 000	70·00	12.000	10,900
1886	18,000	16,344	1 440.000	80·00	14,400	13,075
1887	18,000	16,344	1.440 000	80 00	14 400	13 075
1888	20,000	18.149	1,600 000	80·00	16 000	14 519
1889	16,970	15,390	1,357.600	80 00	13.576	12.312
1890	*20.000	*18 140	1 600,000	*80·00	*16 000	* 14 512
1891	23 700	21 496	1 600 000	67·51	18,960	17 197
1892	27.500	24,946	2 200,000	80·00	22 000	19 957
1893	25,000	22 678	1 875 000	75·00	20.000	18 142
1894	22,814	20 697	1,711 275	75·00	18 251	16 554
1895	22.690	20.498	1 588.300	70·00	18 152	16 398
1896	15 863	14,391	1,189.725	75·00	12 690	11 513
1897	26 262	23 825	1 686 020	64·20	21 010	19 060
1898	32,747	29.708	2 226,796	68·00	26.198	23 766
1899	39.663	35 982	3.331 692	84·00	31 730	28.786
1900	47 151	42,775	3,772.080	80·00	39,721	34,220

The statistics of production and price in the above table from 1880 to 1892 both years inclusive, are from the reports of the U. S. Geological Survey. Apparently the figures for 1881–1888 are approximate estimates rather than totals based on reports from the producers The statistics from 1893 to 1900, both years inclusive, are as reported by The Mineral Industry The figures marked with an asterisk are estimated.

The production of zinc oxide in the United States, with the exception of an insignificant quantity, is made directly from ore, whereas the oxide produced in Europe is obtained by the combustion of spelter, its manufacture in that way constituting one of the important channels of European spelter consumption. In considering the relative importance of the United States as a zinc-producing country, therefore, its production of zinc oxide, reduced to the basis of its tenor in metallic zinc, should be added to the production of spelter.

STATISTICS OF CONSUMPTION.

The only feasible method open to statisticians in determining the consumption of a commodity is the combination of the statistics of production of each country with the movement into and out of it as shown by the statistics of imports and exports. The production plus the imports indicates the total supply; deducting the exports from the supply should show the consumption. This method is admittedly inexact, because it takes no account of the stocks on hand, which may be added to or drawn from. The extension of such statistics over a series of years will, however, indicate truly the amount and trend of consumption, inasmuch as additions to, or draughts upon, stocks are not cumulative but will balance in the long run, although the consumption for any single year is likely to be inaccurately measured.

It is not merely the difficulty mentioned above that perplexes the statistician who undertakes to study the consumption of such a metal as spelter: he must also decide how the imports and exports are to be classified and tabulated in order to lead to results which will be commercially valuable. Should the statistics be based simply upon the crude metal? or upon the crude metal and all the manufactures thereof? or upon something intermediate between the two extremes?

The statisticians of the Metallgesellschaft, in their valuable annual publication, compute the consumption first upon the basis of the crude metal alone, and compare the total thus obtained with the statistics of production. This is probably the best that can be done, because although the statistics of crude spelter consumption do not correctly indicate the ultimate consumption of zinc in the respective countries, any attempt to follow the consumption far into the domain of manufacture would necessarily lead to complications which could not be easily avoided. The statisticians of the Metallgesellschaft, however, present supplementary tables showing the result of such an investigation so far as possible, which are probably sufficient for all commercial purposes. Those supplementary tables have been transcribed herein only in so far as sheet zinc is concerned, inasmuch as the advisability of attempting to trace the consumption of zinc any further is doubtful. If for example spelter is consumed in Germany for the manufacture of brass wares which are subsequently exported, it seems that it may properly be considered that Germany was the place of consumption of that spelter, irrespective of whither the brass wares may have gone. The case of sheet zinc may be regarded in a different light; although it is truly a manufactured product, it is only in the early stage of manufacture, in fact only one degree beyond the spelter itself, and is to a large extent produced in direct connection with the smelting process; it would obviously be incorrect to reckon the spelter consumed for the manufacture of sheet zinc in Germany as being entirely consumed in that country.

The following tables of the consumption of zinc in the various countries of the world, with the exception of that for the United States, are taken from the eighth annual report of the Metallgesellschaft, which bases its computations upon the statistics of production compiled by Henry R. Merton & Co., but separated according to political divisions. The table of the consumption in the United States has been compiled from the statistics of *The Mineral Industry,* with corrections in the statistics of production in 1895 and 1896 as previously noted. In the general table showing the world's consumption, however, the figures for the United States are those which are used by the Metallgesellschaft.

CONSUMPTION OF ZINC IN AUSTRIA-HUNGARY.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production ..	6.542	5.100	7.681	8.715	8,488	9.403	8,314	7.229	7.305	7,026
Imports.....*a*	11.244	14.010	15.083	15.314	17.156	17.539	16.599	17.471	15 225	17,844
Supply..	17.786	19,110	22.764	24.029	25.644	26 942	24.913	24,700	22 530	24 872
Exports.....*a*	546	591	718	447	504	1.256	770	1 184	1.614	1 080
Consumption.	17.240	18,519	22.046	23,582	25,140	25,686	24,143	23 516	20,916	23.788

a Inclusive of spelter and old zinc.

CONSUMPTION OF ZINC IN BELGIUM.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production. ...	83,751	87,050	90,676	92,813	105,853	110,920	114,650	118,085	121,500	117,300
Imports .....*a*	3,139	3,033	4,956	8,354	4,455	10,000	5,000	6,700	4,200	*3,500
Supply.	86,890	90,083	95,632	101,167	110,308	120,920	119,650	124,785	125,700	120,800
Exports .....*a*	*54,600	*55,300	*60,900	*60,600	*72,000	*82,000	*81,000	*85,000	*79,000	*75,000
Consumption *a*	32,290	34,783	34,732	40,567	38,308	38,920	38,650	39,785	46,700	45,800
Exports of sheet zinc ..	*18,000	*17,000	*18,000	*18,000	*19,000	*20,000	*21,000	*20,000	*20,000	*21,000
Net consumption........	14,300	17,800	16,700	22,600	19,300	18,900	17,700	19,800	26,700	24,800

a Crude spelter only. *Estimated.

CONSUMPTION OF ZINC IN FRANCE.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production. ...	20,298	20,809	22,439	23,090	26,860	33,980	37,239	37,932	39,305	36,329
Imports .....*a*	30,243	29,741	33,090	33,256	25,051	33,021	30,201	32,140	25,094	32,894
Supply.	50,541	50,550	55,529	56,346	51,911	67,001	67,440	70,072	64,399	69,225
Exports .....*a*	1,823	1,993	3,283	3,419	2,644	4,438	3,875	8,816	7,403	6,933
Consumption..	48,718	48,557	52,246	52,927	49,267	62,563	63,565	61,256	56,996	62,291

a Crude spelter only.

MOVEMENT OF SHEET ZINC FROM AND INTO FRANCE.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Exports......	2,966	3,665	3,965	3,176	3,205	6,048	7,102	8,379	7,556	5,779
Imports......	205	143	811	396	601	438	1,011	202	422	245
Difference.....	2,761	3,522	3,154	2,780	2,304	5,610	6,091	8,177	7,134	5,534

CONSUMPTION OF ZINC IN GERMANY.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production....	139,353	139,938	142,956	143,577	150,286	153.100	150,739	154,867	153.155	153,350
Imports.....*a*	7,969	13,021	13,211	17,988	17,542	15,668	18,925	22,771	22,171	22.758
Supply.......	147,322	152,959	156,167	161,565	167,828	168,768	169,664	177,638	175,326	176,108
Exports.....*a*	57,853	53,287	62,592	61,799	56.933	55,937	49,600	49,471	45.031	50,302
Consumption..	89,469	99,672	93,575	99,766	110,895	112,831	120,064	128,167	130.295	125,806

a Crude spelter only.

MOVEMENT OF SHEET ZINC FROM AND INTO GERMANY.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Exports......	15,488	16,304	17,459	16,038	15,921	16,242	17,453	14,477	18,281	16,709
Imports......	40	48	74	274	128	180	130	53	95	145
Difference.....	15,448	16,256	17,385	15,764	15,793	16,062	17,323	14,424	18.186	16,564

CONSUMPTION OF ZINC IN GREAT BRITAIN.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production....	29,880	30,794	28,829	32.577	29,965	25,277	23,927	28.386	32.223	30,307
Imports.....*a*	59,448	53,638	57,836	53,743	63,525	77,861	71.002	78,711	71,067	70.649
Supply.......	89,328	84,432	86,665	86,320	93,490	103,138	94,929	107,097	103.290	100.956
Exports.....*a*	8,573	10,885	11,623	10,816	9,662	10.487	8.342	8.450	6,520	8.230
Consumption..	80,755	73,547	75,042	75,504	83,828	92,651	86,587	98.647	96,770	92,726

a Crude spelter only.

MOVEMENT OF SHEET ZINC FROM AND INTO GREAT BRITAIN.

(In metric tons.)

Year.	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Imports.....*a*	20,480	19,260	18,716	18,841	19,670	21,339	21.395	21,612	21,526	22,105
Exports.....*a*	1,879	1,387	1,517	1,388	1,489	1,518	1,650	1,889	1,833	1,675
Difference.....	18,601	17,873	17,199	17,453	18,181	19,821	19 745	19,723	19,693	20,430

a Classified in the British statistics as manufactures of zinc, but chiefly sheet.

CONSUMPTION OF ZINC IN ITALY.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Imports . . . . . *a*	1,682	1,583	1,682	2,230	2,378	2,596	3,278	2,813	3.498	3,627
Exports . . . . . *a*	54	8	nil	nil	4	33	309	156	227	359
Consumption *a*	1,628	1,575	1,682	2,230	2,374	2,563	2,969	2,657	3,271	3,268
Imports of sheet zinc. . *b*	3,200	3.136	3.029	3.248	3,137	3.482	3,556	3,200	3,221	3,543
Total. . . . . *c*	4,828	4,711	4,711	5,478	5,511	6,045	6,525	5,857	6,492	6,871

a Crude spelter only.
b Including other manufactures of zinc.
c This is equivalent to the total consumption of zinc in Italy, inasmuch as the exports of sheet zinc and other manufactures are insignificant.

CONSUMPTION OF ZINC IN RUSSIA.

(In metric tons.)

Year.	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production. . .	3,821	4,338	4.600	5,100	5,040	6.263	5.853	5 665	6.324	5,968
Imports. . . . . *a*	4.700	5,100	5,000	5,700	6.630	5,650	7.800	9,000	9 300	8 600
Consumption *a*	8,521	9,438	9,600	10,800	11,670	11,913	13,653	14,665	15,624	14,568
Imports of sheet zinc. . .	200	100	400	600	400	300	350	250	540	630
Total.	8,721	9,538	10,000	11,400	12,070	12.213	14,003	14.915	16 164	15.198

a Crude spelter only.

CONSUMPTION OF ZINC IN SPAIN.

(In metric tons.)

Year	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Production. . . .	5,656	5 925	6,000	6.000	6.000	6.133	6.244	6.031	6.184	5 785
Exports . . . . . *a*	2,045	2,762	2.500	2.680	1,367	3,561	2 171	4.553	2.701	2.081
Consumption. .	3,611	3,163	3,500	3,320	4,633	2,572	4.073	1,478	3,483	3.704

a Includes sheet zinc.

It will be observed from the above tables that Austria-Hungary, Italy and Russia have to import a good deal of spelter and export almost none at all. France and Great Britain, although large producers themselves and exporters of considerable quantities, are obliged to make large importations to meet the home consumptive demand. The great exporting countries are Germany and Belgium, especially the latter. The small production in Spain is more than that country produces, while Denmark, Sweden, Norway, Switzerland, Turkey and the Balkan countries, which produce no spelter, have to import their entire supplies.

CONSUMPTION OF ZINC IN THE UNITED STATES.

(In tons of 2,000 lb.)

Year	Production	Imports			Supply	Exports	Consump'n.
		Spelter	Sheet	Total			
1873	7,343	3 420	5,561	8,981	16,324	37	16.287
1874	10 000	1,797	3,008	4,805	14,805	22	14,783
1875	15,833	1,017	3,660	4,677	20,510	19	20,491
1876	16 000	474	2,306	2,780	18,780	67	18,713
1877	17,500	633	671	1 304	18,804	720	18,084
1878	19 000	635	628	1,263	20,263	1,273	18,890
1879	21,000	710	556	1,266	22,266	1,066	21.200
1880	23.239	4,046	2,035	6,081	29,320	684	28,636
1881	30,000	1,430	1.364	2,794	32,794	746	32.048
1882	33,765	9,204	2,207	11,411	45,176	745	44,431
1883	36,872	8,534	1,655	10,189	47,061	426	46,635
1884	38.544	2,935	476	3,411	41,955	63	41,892
1885	40.688	1,758	920	2,678	43,366	51	43.315
1886	42.641	2.150	546	2,696	45,337	459	44,878
1887	50,340	4,194	463	4,657	54,997	68	54,929
1888	55,903	1,913	148	2,061	57,964	31	57,933
1889	58,860	1,026	507	1,533	60,393	440	59,953
1890	67.342	1,000	391	1,391	68,733	1,648	67,085
1891	80,262	404	11	415	80,677	2,147	78,530
1892	84,082	149	14	163	84,245	6,247	77,998
1893	76,255	213	14	227	76,482	3,723	72,759
1894	74,004	194	20	214	74,218	1,804	72,414
1895	87,591	372	21	393	....	1,530	86,454
1896	81,878	520	14	534	....	10,130	72,282
1897	100.387	1,453	8	1,461	101,848	14,245	87,603
1898	114,104	1,371	....	1,371	115,475	10,500	104,975
1899	129,675	1,493	....	1.493	131,168	6.755	124,413
1900	123,231	1.007	....	1 007	124,238	22,410	101,828

The world's consumption of zinc from 1891 to 1900 is summarized in the following table, compiled by the Metallgesellschaft, in which its figures for the United States have been retained:

THE WORLD'S CONSUMPTION OF ZINC.

(In metric tons.)

Country	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Austria.......	17.240	18.519	22,046	23,582	25,140	25,686	24,143	23,516	20,916	23,782
Belgium......	32,290	34,783	34,732	40,567	38,308	38,920	38,650	39,785	46,700	45,800
France	48,718	48,557	52,246	52,927	49,267	62.563	63.565	61,256	56,996	62,291
Germany	89,469	99,672	93,575	99,766	110,895	112,831	120,064	128,167	130,295	125,806
Great Britain .	80,755	73,547	75,042	75,504	83,828	92,651	86,587	98,647	96,770	92,726
Italy.........	1,628	1,575	1,682	2.230	2.374	2,563	2,969	2,657	3,271	3.328
Netherlands. .	*3,600	*3,600	*3,600	*3,600	*3,600	*3,600	*3,600	*3,600	*3,600	*3,600
Russia........	8,521	9,438	9,600	10,800	11,670	11,913	13,653	14,665	15,624	14,568
Spain.........	3,611	3,613	3,500	3,320	4,633	2,572	4,073	1,478	3,483	3,704
United States. .	71,327	73,465	69,058	64.028	78.424	65,427	77,778	95,711	112,905	90,360
Other countries	*8,000	*9.500	*13,000	*8.000	*7,000	*7,600	*7,000	*6,500	*7,000	*7,000
Total.........	365,159	376,269	378,081	384.324	415,139	426,326	442,082	475,982	497,560	472.965
Production....	362,204	372,900	378,093	380,877	416,621	424,141	443,302	469,031	490,205	478,323
Av. pr. London	£23¼	£20¾	£17⅜	£15½	£14⅝	£16⅝	£17½	£20½	£24⅞	£20¼
" " " *a*	5.054	4.511	3.778	3.368	3.179	3.615	3.805	4.457	5.408	4.402
" " N. Y. *b*	5.02	4.63	4.075	3.52	3.63	3.94	4.12	4.57	5.75	4.39

a The average price at London in pounds sterling per 2,240 lb., as reported by the Metallgesellschaft, has been converted into dollars and cents per 100 lb. at the uniform rate of £1=$4·87.

b Averages reported by The Mineral Industry.

The above table shows the total consumption and production of zinc during the last ten years to have been as follows:

Period.	1891–1895	1896–1900	1891–1900
Production.....	1,910,695	2,305,002	4,215,697
Consumption...	1,918,972	2,314,915	4,233,887

Statistics of consumption compiled in the manner of the above introduce as an important factor the statistics of production. In comparing consumption and production, therefore, the chief reliance is in the accuracy of the statements as to imports and exports, inasmuch as errors in the statistics of production are reproduced equally in the computations as to total production and consumption.

According to the above statistics the consumption of zinc during the 10 years 1891-1900 has been greater than the production, whence the natural inference is that stocks on hand must have been drawn upon. In view however, of the really small difference in the grand totals, that would probably be an unsafe conclusion, taking into consideration the numerous chances of small errors in the statistics, and the movement of old metal which may appear in the reports of imports and exports and be counted twice.

CONSUMPTION OF ZINC WHITE IN THE UNITED STATES.

(In tons of 2,000 lb.)

Year	Production	Imports	Supply	Exports	Consumpt'n	TENOR IN ZINC AT 80%	
						Short tons	Metric tons
1894	22,814	1,686	24,500	nil	24,500	19,600	17,777
1895	22,690	2,273	24,963	24	24,939	19,951	18,096
1896	15,863	2,286	18,149	2,324	15,825	12,660	11,483
1897	26,262	2,782	29,044	1,859	27,185	21,748	19,725
1898	32,747	1,671	34,418	3,925	30,493	24,394	22,125
1899	39,663	1,506	41,169	5,343	35,826	28,661	25,995
1900	47,151	1,309	48,460	5,695	42,765	34,212	31,030

Previous to 1886 the only available statistics of American imports and exports of zinc white are for fiscal years ending June 30, which are of course useless for comparison with the production reported for calendar years.

The imports entered in the above table represent only dry zinc oxide, besides which a small quantity of zinc oxide ground in oil is brought into the United States, the quantity of the latter product being 65 tons in 1895, 155 tons in 1896, 251 tons in 1897, 13 tons in 1898 and 21 tons in 1899.

Previous to 1897 the exports reported above include zinc ore, corresponding to "oxide and ore" of the U. S. Bureau of Statistics enumeration. There was, however, but little ore exported before 1897, when large shipments from New Jersey first began to be made.

The imports of zinc white into the United States and the supply from 1885 to 1894 were as follows (in tons of 2,000 lb.):

Year	1886	1887	1888	1889	1890	1891	1892	1893
Production	18,000	18,000	20,000	16,970	20,000	23,700	27,500	25,000
Imports	1.763	2,481	701	1,343	1,316	1,420	1,221	1.950
Supply	19,763	20,481	20,701	18,313	21,316	25,120	28,721	26,950

STATISTICS OF PRICE.

The chief markets of the world, wherein the prices which govern the zinc industry are established, are New York, London and Breslau. The price made in London practically governs the industry in Europe, although the business in Upper Silesia is transacted on the basis of the price at Breslau; the latter generally preserves, however, a certain relation to the London price. Similarly in the United States the business of the Kansas-Missouri district is transacted on the basis of the price of spelter at St. Louis. The St. Louis price, however, corresponds with the New York price, minus the difference in the cost of transportation to the two points. That difference is not merely the freight rate from St. Louis to New York, but is the variation between the rates from Kansas smelting points to New York and St. Louis respectively. For example, at the present time (1901) the rate on spelter from Iola to New York is $0·34 per 100 lb.; and from Iola to St. Louis, $0·18 per 100 lb. The difference between the New York and St. Louis prices should, therefore, be normally $0·16 per 100 lb.; practically the difference during 1901 has been $0·15 most of the time. The difference between the New York and St. Louis price varies consequently from time to time according to the freight rates, taking into account rebates, etc., which may be allowed by the railways. There are no available statistics of the average price at St. Louis for a long period of years, but considering the last ten years it would probably be nearly correct to reckon it as having been $0·20 per 100 lb. below the New York price.

Although the importation of spelter into the United States from Europe is practically prohibited by the tariff of 1·5c. per lb., and the American price of spelter is consequently to a considerable extent independent of the European price, there is nevertheless at many times an intimate relation between the prices of the two Continents, because of the ability of American producers to export zinc at a profit under certain conditions. The freight rate on spelter from Iola to Liverpool is now $0·35 per 100 lb. (within five years or so it has been as low as $0·22 per 100 lb.). Consequently if at any time the price of spelter at London rises materially above the price at New York and there be an excess of productive capacity in the United States, which generally there is, the exportation of spelter from Kansas to Europe tends to reduce the European price to the American level.

The average monthly price of spelter at New York and the average annual price at London and in the principal markets of Germany, for a long period of years, are given in the subjoined tables. The New York prices are as quoted by *The Mineral Industry;* the authorities for the English and German prices are stated in each case.

AVERAGE MONTHLY PRICE OF PRIME WESTERN SPELTER AT NEW YORK, IN CENTS PER POUND.

	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year
1875	6·56	6·46	6·35	6·75	7·20	7·20	7·30	7·175	7·175	7·275	7·275	7·275	7·00
1876	7·50	7·625	7·685	7·80	7·875	7·625	7·185	7·125	6·96	6·685	6·495	6·435	7·25
1877	6·375	6·56	6·435	6·31	6·125	5·995	5·745	5·85	5·81	5·80	5·745	5·625	6·03
1878	5·625	5·435	5·435	5·125	4·81	4·435	4·625	4·685	4·81	4·66	4·625	4·31	4·88
1879	4·375	4·51	4·495	4·50	4·375	4·245	4·56	5·21	5·81	6·185	6·06	6·125	5·036
1880	6 185	6·56	6·625	6·31	5·81	5·31	4·935	5·06	4·935	4·935	4·775	4·70	5·51
1881	5·06	5·185	4·935	4·935	5·935	4·875	4·875	5·06	5·125	5·31	5·685	5·935	5·243
1882	5·875	5·685	5·495	5·375	5·435	5·31	5·245	5·31	5·245	5·245	4·995	4·685	5·325
1883	4·56	4·56	4·685	4·675	4·625	4·495	4·40	4·35	4·45	4·40	4·385	4·36	4·495
1884	4·285	4·325	4·50	4 575	4·525	4·455	4·50	4·57	4·56	4·475	4·35	4·125	4·443
1885	4·31	4·275	4·21	4·21	4·175	4·05	4·25	4·50	4·56	4·56	4·525	4·525	4·345
1886	4·40	4·425	4·55	4·55	4·50	4·375	4·35	4·35	4·325	4·275	4·275	4·425	4·40
1887	4·55	4·55	4·475	4·45	4·55	4·55	4·575	4·55	4·50	4·525	4·775	5·40	4·625
1888	5·425	5·35	5·10	4·85	4·65	4·55	4·55	4·75	4·975	5·05	4·90	4·875	4·91
1889	5·00	4·95	4·75	4·675	4·75	4·975	5·10	5·20	5·175	5·10	5·20	5·40	5·023
1890	5·41	5·28	5 187	5·085	5·35	5·575	5·55	5·275	5·06	6·012	6·122	6·106	5·55
1891	5·55	5 025	5·125	5·00	4·85	5·083	5·063	5·01	4·958	5·02	4·83	4·75	5·02
1892	4·69	1·62	4·89	4·68	4·79	4·71	4·78	4·69	4·53	4·41	4·47	4·40	4·63
1893	4·39	4·33	4·28	4·38	4·41	4·27	4·13	3·89	3·69	3·68	3·65	3·80	4·075
1894	3·56	3·85	3·89	3·62	3·47	3·40	3·43	3·38	3·44	3 45	3·36	3·43	3·52
1895	3·28	3·20	3·23	3 30	3·50	3·65	3·75	4·15	4·30	4·10	3·55	3·49	3·63
1896	3·75	4·03	4·20	4·09	3·98	4·10	3·97	3·76	3·60	3·72	3·99	4·14	3·94
1897	3·91	4·02	4·12	4·13	4·21	4·21	4·32	4·26	4·18	4·17	4·03	3·89	4·12
1898	3·96	4·04	4·25	4·26	4·27	4·77	4·66	4·58	4·67	4·98	5·29	5·10	4·57
1899	5·34	6·28	6·31	6·67	6·88	5·98	5·82	5·65	5·50	5·32	4·64	4·66	5·75
1900	4·65	4·64	4·60	4·71	4·53	4·29	4·28	4·17	4·11	4·15	4·29	4·25	4·39
1901	4·13	4·01	3·91	3·98	4·04	3·99	3·95	3·99	4·08	4·23	4·29	4·31	4·08

AVERAGE ANNUAL PRICE OF ORDINARY SILESIAN SPELTER AT LONDON, PER TON OF 2,240 LB.

(From statistics of the Metallgesellschaft.)

	£	s.	d.		£	s.	d.		£	s.	d.		£	s.	d.
1869	20	7	4	1877	19	18	8	1885	13	19	11	1893	17	8	0
1870	18	10	4	1878	17	17	10	1886	14	5	1	1894	15	9	2
1871	18	8	9	1879	16	12	0	1887	15	4	0	1895	14	12	2
1872	22	9	1	1880	18	7	1	1888	18	1	9	1896	16	11	10
1873	26	3	6	1881	16	5	6	1889	19	15	7	1897	17	9	10
1874	22	17	7	1882	16	19	9	1890	23	4	6	1898	20	8	9
1875	24	1	4	1883	15	6	6	1891	23	4	8	1899	24	17	2
1876	23	6	3	1884	14	8	11	1892	20	16	6	1900	20	5	6

The average price of Silesian spelter at London in 1901 was £17 0s. 7d.

AVERAGE ANNUAL PRICE OF ENGLISH SPELTER AT LONDON, PER TON OF 2,240 LB.

(From British blue books on mines and minerals in the United Kingdom.)

	£	s.	d.		£	s.	d.		£	s.	d.		£	s.	d.
1872	22	14	9	1880	19	3	9	1888	19	2	10	1896	17	6	8
1873	26	16	6	1881	16	17	8	1889	20	9	2	1897	17	19	10
1874	23	6	9	1882	17	15	6	1890	23	13	11	1898	20	18	8
1875	24	5	0	1883	16	1	0	1891	23	18	5	1899	25	6	2
1876	23	7	11	1884	15	8	2	1892	21	15	9	....	..	..	..
1877	21	15	0	1885	14	18	10	1893	17	8	0	....	..	..	..
1878	19	9	9	1886	15	14	0	1894	16	2	4	....	..	..	..
1879	17	5	0	1887	16	1	5	1895	15	5	8	....	..	..	..

AVERAGE YEARLY PRICE OF SPELTER IN THE PRINCIPAL MARKETS IN GERMANY. *a*

(Price in marks per 100 kg.; 1 mark=23·8 cents.)

Year	Breslau	Cologne	Frankfurt	Halberstadt	Hamburg
	Good Silesian from works	Rhenish brands W.H. and S.S. 3 mos.	Refined calamine and blende from works	Rhenish and Westphalian, crude, from works, 1 to 3 mos	Silesian in slabs
1879	.....	34·27	36·38	32·44	35·14
1880	.....	37·88	38·47	36·23	38·61
1881	.....	32·66	33·53	31·09	32·57
1882	.....	34·32	34·88	33·84	35·04
1883	28·60	30·58	31·35	30·19	31·68
1884	27·20	29·00	29·06	28·66	30·16
1885	26·08	28·14	27·91	27·68	28·60
1886	26·71	28·64	28·03	28·11	29·58
1887	28·38	30·47	29·96	29·88	31·23
1888	35·41	37·21	36·02	36·10	39·26
1889	38·25	40·56	39·68	39·48	41·55
1890	45·11	47·92	47·02	46·58	49·27
1891	44·98	47·34	46·44	46·54	48·95
1892	40·54	43·08	42·17	42·23	45·30
1893	33·60	35·78	35·13	34·42	38·07
1894	29·90	32·40	30·90	30·90	34·30
1895	28·30	30·20	29·20	29·30	31·90
1896	31·60	34·00	33·20	32·20	34·60
1897	33·90	36·00	35·30	34·70	37·00
1898	39·50	41·80	41·70	41·40	43·20
1899	48·10	51·40	50·50	50·00	51·90
1900	39·46	42·97	41·31	41·12	42·50

a From Vierteljahrshefte zur Statistik des Deutschen Reichs.

Equivalent Prices of Spelter in Pounds Sterling per 2,240 lb., Marks per 100 kg. and U. S. Currency per 100 lb.

The price of spelter (or any other commodity) quoted in pounds sterling per 2,240 lb. can be converted into the equivalent in U. S. currency per 100 lb. by the following formula:

$$P = (P^1 \times E) \div 22{\cdot}4$$

in which

P=price per 100 lb. in dollars and cents.

P^1=price in pounds sterling, shillings and pence being expressed in decimal parts of a pound.

E=the value of £1 in U. S. currency—i. e., at the current rate of exchange.

For example, the equivalent of £22 10s. 6d. per 2,240 lb. in dollars and cents per 100 lb., when exchange is at $4·866, is computed as follows:

£22 10s. 6d.=£22·525
£22·525×4·866÷22·4=$4·983

In the same manner the equivalents of £14 to £29 at rates of exchange of $4·83 to $4·89, with intervals of £1 and 1c. respectively, have been calculated in the following table:

EQUIVALENT PRICES IN POUNDS STERLING PER 2,240 LB. AND DOLLARS AND CENTS PER 100 LB. AT DIFFERENT RATES OF EXCHANGE.

£ 1 =	$4·83	$4·84	$4·85	$4·86	$4·87	$4·88	$4·89	Increase
£14	†$3·019	†$3·025	$3·031	$3·038	$3·043	†$3·050	$3·056	0·6250
15	3·234	3·241	3·248	3·254	3·259	3·268	3·275	0·6696
16	†3·450	3·457	3·464	3·471	3·479	3·486	3·493	0·7143
17	3·666	3·673	3·681	3·688	3·696	3·704	3·711	0·7589
18	3·881	3·889	3·897	3·905	3·913	3·922	3·929	0·8035
19	4·097	4·105	4·114	4·122	4·131	4·139	4·148	0·8482
20	4·313	4·321	4·330	4·339	4·348	4·357	4·366	0·8927
21	4·528	4·538	4·547	4·556	4·566	†4·575	4·584	0·9375
22	4·744	4·754	4·763	4·773	4·783	4·793	4·803	0·9821
23	4·959	4·970	4·980	4·990	5·000	5·011	5·021	1·0268
24	†5·175	5·186	5·196	5 207	5·218	5·229	5·239	1·0714
25	5·391	5·402	5·413	5·424	5·435	5·446	5·458	1·1161
26	5·606	5·618	5·629	5·641	5·653	5·664	5·676	1·1607
27	5·822	5·834	5·846	5·857	5·870	5·882	5·894	1·2054
28	6·038	†6·050	6·063	†6·075	6·088	†6·100	6·113	1·2500
29	6·253	6·266	6·279	6·292	6·305	6·318	6·331	1·2946
Increase	21·5625c.	21·6071c.	21·6518c.	21 6964c.	21·7411c.	21·7857c.	21·8304c.	0·0446

The figures marked with daggers in the above table are absolutely correct, the dividend giving with the divisor, 22·4, the quotient as entered, without remainder. The other figures in the table involve errors to the maximum extent of plus or minus $0·0005=1c. per 2,000 lb. The extreme right hand column gives the differences in cents per 100 lb. caused by fluctuations of 1c. in the rate of exchange, from which the equivalents of prices at intermediate rates of exchange can be found by interpolation. At the foot of each column headed by the rate of exchange is given the difference in cents per 100 lb. caused by a difference of £1 per 2,240 lb., from which the equivalents of fractional values can be quickly arrived at by interpolation. It may be assumed roughly that a difference of 1s. per 2,240 lb. corresponds to 1c. per 100 lb., 6d.=0·5c. and 3d.=0·25c. for all rates of exchange between $4·83 and $4·89. In determining the equivalent of a fractional price with an intermediate rate of exchange with absolute accuracy a double interpolation must be made.

The German mark being equivalent to 23·8c. U. S. currency, at par of exchange, and one kilogram being equal to 2·2046 lb., a difference of one mark per 100 kg. corresponds to 10·8c. per 100 lb. The quotation of 40 marks per 100 kg. is equivalent, therefore, to $4·32 per 100 lb. In order to convert a quotation in marks per 100 kg. into the equivalent dollars and cents per 100 lb., multiply the number of marks by 0·108; thus:

$$48{\cdot}1 \times 0{\cdot}108 = 5{\cdot}19.$$

For convenient comparison of the prices of spelter at New York and London, and at the works in Upper Silesia, the averages reported in pounds sterling per long ton and marks per 100 kg. have been reduced in the following table to the common basis of cents per pound. The statistics for New York have been taken from *The Mineral Industry;* those for London represent the price of ordinary Silesian spelter as reported by the Metallgesellschaft; and those for Silesia are from the *Statistik des Oberschlesischen Berg- und Hüttenmännischen Vereins.* The last have been used rather than the official statistics of the German Government because they agree closely with the latter and go further back.

AVERAGE ANNUAL PRICE OF SPELTER IN VARIOUS MARKETS, REDUCED TO CENTS PER POUND.

Year	New York	London	Silesia	Year	New York	London	Silesia
1871	.....	4·008	3·856	1886	4·400	3·097	2·765
1872	.....	4·881	4·406	1887	4·625	3·302	2·970
1873	.....	5·691	5·162	1888	4·910	3·932	3·499
1874	.....	4·973	4·568	1889	5·023	4·299	3·877
1875	7·000	5·232	4·903	1890	5·550	5·048	4·752
1876	7·250	5·068	4·655	1891	5·020	5·049	4·763
1877	6·030	4·333	3·974	1892	4·630	4·527	4·169
1878	4·880	3·888	3·478	1893	4·075	3·783	3·499
1879	5·036	3·609	3·240	1894	3·520	3·359	3·067
1880	5·510	3·989	3·672	1895	3·630	3·175	2·916
1881	5·243	3·538	3·283	1896	3·940	3·608	3·229
1882	5·325	3·693	3·413	1897	4·120	3·803	3·542
1883	4·495	3·329	3·056	1898	4·570	4·443	3·996
1884	4·443	3·140	2·884	1899	5·750	5·405	5·033
1885	4·345	3·043	2·732	1900	4·390	4·407	4·201

It will be observed from the above table that the price of spelter has been subject to wide fluctuations, especially in the United States, where the range has been from 7·875c. (the average for May, 1876) to 3·20c. (the average for February, 1895).[1] The maximum price was attained at a time when the market was controlled by a combination of producers which was organized for the purpose of enhancing the value of spelter and was temporarily successful in doing so. The minimum price was quoted during the period of depression which followed the panic of 1893.

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[1] This statement covers the period from Jan. 1, 1875, to Dec. 31, 1901. In February, 1895, spelter touched 2·90c. at St. Louis.

It is a common practice in considering statistics of price to compare the average of a single year with the average of the previous 10 years. Such decennial averages have been computed for each year in the following table:

AVERAGE PRICE OF SPELTER FOR DECENNIAL PERIODS, REDUCED TO CENTS PER POUND.

Ten years ending	New York	London	Silesia	Ten years ending	New York	London	Silesia
1871	....	....	3·947	1886	4·972	3·566	3·250
1872	....	....	4·047	1887	4·832	3·463	3·149
1873	....	....	4·224	1888	4·835	3·467	3·151
1874	....	....	4·253	1889	4·833	3·536	3·215
1875	....	....	4·331	1890	4·837	3·642	3·323
1876	....	....	4·373	1891	4·815	3·793	3·471
1877	....	4·703	4·350	1892	4·745	3·877	3·547
1878	....	4·653	4·290	1893	4·703	3·922	3·591
1879	....	4·571	4·201	1894	4·611	3·944	3·609
1880	....	4·567	4·191	1895	4·539	3·957	3·628
1881	....	4·520	4·134	1896	4·493	4·008	3·674
1882	....	4·401	4·035	1897	4·442	4·058	3·731
1883	....	4·165	3·824	1898	4·408	4·109	3·781
1884	5·522	3·982	3·656	1899	4·481	4·220	3·897
1885	5·257	3·763	3·439	1900	4·365	4·156	3·842

The production of zinc in Europe, which was inaugurated during the first decade of the nineteenth century, did not attain much importance until about 1840, in which year it was probably not to exceed 20,000 metric tons. The next decade witnessed a great expansion and in 1850 the production was 50,000 tons, or close to it. In 1860 it was somewhat more than 85,000 tons. From 1831 to 1840 the average price of spelter in Upper Silesia was 26·08 marks per 100 kg.; from 1841 to 1850 it was 36·34; from 1851 to 1860 it was 36·84. The annual price since 1860 may be found from the tables immediately preceding this paragraph and from the table of ore production in Upper Silesia in the early part of this chapter. An account of the fluctuations in price previous to 1841 may be found in Chapter I.

Between 1841 and 1860 the price of spelter varied a good deal. By 1840 the Silesian zinc industry had recovered from the crisis of 1829, when spelter sold at 18 marks per 100 kg. at Breslau (the lowest recorded price), and having been established on a sounder basis the increasing demand began to outstrip the production, leading to a range of prices from 32 to 52 marks. The political disturbances, which arose in most parts of Europe in 1848, upset the market and from 1848 to 1852 the price ruled at 22 to 27 marks. From 1852 to 1870, and later, there was steady prosperity in the business and a general upward tendency in price. Probably about the same conditions existed in the other zinc-producing districts of Europe as in Upper Silesia, but anyway Upper Silesia was the most important market, inasmuch as up to about 1870 its works made 50% or more of the total production of Europe, or it might be said of the world since the output of the United States was of no consequence prior to that time.

It is to be observed then that from the minimum price of 18 marks per 100 kg. (1·944c. per lb.) in 1829, there was in spite of temporary ups and downs a gradual rise to the maximum of 5·162c. in 1873. In the early part of that period of 44 years the consumptive demand for the still comparatively new metal was small, and the facilities for producing it were great. The supply of ore was obtained from near the surface, and it was of high grade and docile in smelting. In the latter part of that period the demand for the metal had greatly increased, but the cost of production had also increased, and the demand could be supplied only at a higher range of prices. We have it on the authority of the Bergwerksgesellschaft Georg von Giesche's Erben that in the 50 years from 1834 to 1883 the cost of production per centner of zinc more than doubled itself, the greater poverty of the ore and the increase in wages and the price of coal more than offsetting the economy in labor and coal and the other improvements in the metallurgical practice.

The decline in the price of spelter which began in 1874 and continued, with only two checks, until 1885 is atttributable chiefly to the new supply of rich ore that in 1870 began to be offered in the form of blende. At about the same time the United States began to be an important producer of zinc, its importations from abroad dwindled down, and this outlet for European spelter gradually became closed. Simultaneously there was a heavy increase in the world's production. Up to 1870 the spelter product of Europe was derived almost exclusively from calamine. Blende had been mined and smelted in Belgium as early as 1845, but the output of that Kingdom never attained much magnitude. The Belgian production of calamine had been on the wane since 1856. In 1870 the Scharley and Marie mines, which had previously been the most important producers in Upper Silesia, came to the end of their resources, but in the same year the blende of the district began to be utilized, although its production did not assume large proportions until nearly 10 years later.

The abnormally high prices for spelter in the United States in 1875 and 1876 were to a large extent artificial, being due to the manipulations of a combination of the western producers, which was formed in the spring of 1875. In April, 1876, it succeeded in raising the nominal price of spelter to 8c., New York, but production had been stimulated, consumption restricted and stocks accumulated, so that in June, 1876, the combination was practically disrupted, this being followed by a rapid decline in the price. In 1879 and 1882 syndicates to control production and price were organized in Europe, but their efforts were of only temporary effect on the market, which continued to sag under the weight of the heavy production. During the decade 1881-1890 the exports of spelter from Europe to the United States again became of considerable importance, attaining a maximum of 11,411 short tons in 1882 (in which year the American production was 33,765 tons), but since 1887 foreign spelter has ceased to be of any consequence in the American market. In 1888 there was formed in Europe a combination of the French, German, and British producers to restrict production, which went into effect in 1889 and continued to the end of 1894. This was probably the best-sustained effort to regulate the price of spelter, but although it had a temporary influence on the market it could not prevent production by new concerns, who were led into the business by the attraction of high prices and large profits, and its ends were thus defeated.

Since 1890 the predominant features in the zinc market have been the sagging of prices under increasing production in the early part of the decade; the enormously increasing production in the United States and the beginning (in 1896) of large exports to Europe; the decrease in the cost of smelting in the United States because of the utilization of the natural-gas resources of Kansas and improvements in the metallurgical practice; and the increase in the cost of smelting in Europe because of the rise in the value of coal, especially toward the end of the decade. There was a period of industrial depression in both Europe and America in 1893 to 1895; and a recovery, which culminated in a boom in 1899 and the early part of 1900; followed by a depression in Europe, which caused a great decline in the price of spelter there, and sympathetically a corresponding decline in the price in the United States, although the period of general industrial prosperity continued here. It will be observed, from the accompanying tables how since 1890 the American price for spelter has preserved a rather constant relation to the European price, although spelter in Europe can no longer, under normal conditions, enter the American market.

AVERAGE MONTHLY PRICE OF ZINC BLENDE ORE AT JOPLIN, MO.

(Price per 2,000 lb. of ore, in producers' bins.)

Yr.	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	12mos.
1896	$24.00	$23.50	$23.00	$23.00	$21.50	$21.00	$21.50	$21.00	$20.00	$20.50	$23.50	$25.50	$22.33
1897	22.125	21.50	21.00	21.125	21.60	21.875	22.50	22.50	22.625	22.75	23.50	24.25	22.28
1898	23.00	22.50	23.00	24.62	26.50	28.50	28.00	28.37	31.00	33.70	36.25	37.00	28.44
1899	32.25	43.37	43.40	51.50	50.50	45.50	44.20	45.00	43.75	43.50	35.00	36.00	38.54
1900	30.23	29.36	28.45	28.42	26.92	25.00	24.23	25.67	24.25	24.25	24.45	25.40	26.50
1901	23.73	23.96	23.70	24.58	24.38	24.22	24.68	23.88	22.82	21.63	26.15	28.24	24.21

The statistics of the above table are taken from various volumes of *The*

Mineral Industry with the exception of those for 1900 and 1901, for which years the values reported by the U. S. Geological Survey have been adopted. *The Mineral Industry* and the U. S. Geological Survey agree as to the values for the years 1896-1899, both inclusive. There are no complete statistics available for the years previous to 1896. *The Mineral Industry* reports the following averages: 1889, $25; 1890, $23·90; 1891, $25·90; 1892, $22·50; 1893, $19·25; 1894, $17·10. *The Mineral Industry* made no quotation for 1895; Prof. Erasmus Haworth, State Geologist of Kansas, gives the average for that year as $19·68. All of the above figures refer to the average grade of ore. In this connection it is important to remark that the average grade of the ore produced in the Joplin district has been increasing, especially of late when the tendency has been strongly toward making the cleanest ore possible. In the early part of the decade, 1890-1900, the average grade of the ore produced in the district was probably between 56% and 58% Zn; certainly not more than 58%. At present the average is probably very close to 60%, which is rated as the "standard" ore of the district. A good deal of ore assaying 62% to 63% Zn is produced, and occasionally lots assaying as high as 64·5%. The increase in the zinc tenor of the ore should be borne in mind in making comparisons between present prices and those of previous years. Unfortunately ore was not sold on an assay basis at all previous to 1899; consequently there is uncertainty as to what the average values reported for earlier years really indicate.

MISCELLANEOUS STATISTICS.

There are no complete statistics of the production of sheet zinc in Europe, the output being reported officially only from Belgium, Spain and Upper Silesia. There is scarcely a zinc-producing country in Europe, however, which does not have rolling mills. The statistics for Belgium, Spain and Upper Silesia are presented in the following table:

PRODUCTION OF SHEET ZINC IN CERTAIN COUNTRIES.
(In metric tons.)

Country	1892	1893	1894	1895	1896	1897	1898	1899	1900
Belgium	....	32,388	31,724	34,081	36,238	37,011	35,587	34 289	38 825
Silesia	32,266	35,187	34,518	35,676	39,488	36,618	39,863	35,646	38,469
Spain	....	2,462	2,421	2,487	2,648	2,337	1,731	2,084	2.756

Reference has previously been made to the fact that the official statistics of spelter production do not include the make of zinc dust, which is obtained as a by-product in smelting and is marketed in that form. The output of zinc dust in Belgium and Germany amounts to several thousands of tons per annum. Germany also produces a considerable quantity of white vitriol (zinc sulphate) which is obtained directly from ore and contains about 22·5% Zn. The production of zinc sulphate in Germany is shown in the following table:

PRODUCTION OF ZINC SULPHATE IN GERMANY.

(In metric tons.)

Year	Quantity	Year	Quantity	Year	Quantity	Year	Quantity	Year	Quantity	Year	Quantity
1871	746	1876	685	1881	617	1886	1,147	1891	4,117	1896	4,811
1872	733	1877	443	1882	788	1887	1,268	1892	4,479	1897	5,488
1873	707	1878	327	1883	992	1888	1,494	1893	4,527	1898	6,104
1874	418	1879	511	1884	1,155	1889	2,685	1894	4,249	1899	7,117
1875	550	1880	812	1885	1,130	1890	3,769	1895	4,018	1900	6,026

Previous to 1893 a small quantity of nickel sulphate was included in the statistics of production of zinc sulphate.

PRODUCTION OF SPELTER IN UPPER SILESIA BY WORKS.

(In metric tons.)

Works	1891	1892	1893	1894	1895	1896	1897	1898	1899	1900
Bernhardi	......	......	......	......	......	......	212	3,669	4,289	4,108
Beuthener	852	962	*a*2,105	*a*2,014	1,065	*a*2,595	*a*1,955	*a*1,860	*a* 1,809	*a* 1,915
Carls	1,532	1,429	1,459	1,585	1,613	1,624	1,679	1,482	1,406	1,229
Clara	783	842	805	770	1,144	1,108	1,179	1,148	1,342	1,291
Fanny Franz	3,528	3,440	3,416	3,175	3,120	3,312	3,139	2,986	2,743	2,734
Flora	......	......	......	......	......	......	......	......	......	71
Franz	865	890	971	950	935	940	978	1,102	1,051	1,172
Kgl. Friedrichs	217	145	163	144	146	142	188	195	176	164
Godulla	5,390	6,341	6,991	7,045	7,582	7,924	7,282	7,198	7,323	7,108
Guidotto	3,181	4,072	4,276	4,250	4,463	4,383	4,092	3,945	4,085	5,043
Hohenlohe	11,993	11,739	11,396	11,289	11,774	12,112	11,348	10,695	10,178	10,296
Hugo	2,921	2,864	3,429	3,776	3,807	3,828	3,858	5,161	6,319	8,016
Kunigunde	1,010	968	979	1,065	1,291	1,252	1,453	1,556	1,521	1,826
Lazy	2,602	2,686	2,640	2,580	2,524	2,402	2,619	2,562	2,500	2,440
Liebehoffnungs	5,878	5,736	5,803	5,829	6,269	6,368	5,902	5,948	5,896	5,585
Lydognia	1,198	1,249	1,183	1,301	1,407	1,373	1,278	1,173	143	......
Norma	1,264	1,248	1,235	1,241	1,061	1,426	1,435	1,314	1,468	1,475
Pauls	6,684	6,710	7,011	6,793	7,130	7,596	7,186	6,901	7,576	7,496
Rosamunde	1,112	1,209	*a*	*a*	1,100	*a*	*a*	*a*	*a*	*a*
Silesia II	12,384	12,281	12,370	12,180	12,660	12,501	12,520	12,960	12,887	12,855
Silesia III	11,883	11,628	11,784	12,014	12,002	2,619	12,545	12,755	12,732	12,717
Theresia	746	736	1,251	1,478	1,438	1,440	1,399	1,260	1,167	1,177
Thurzo	1,361	1,384	1,396	1,421	1,659	1,735	1,627	1,663	1,634	1,752
Wilhelmine	11,036	10,616	10,963	11,646	10,973	11,733	11,673	11,478	11,868	11,743
Totals. *b*	88,420	89,175	91,716	92,546	93,509	98,323	95,547	99,011	100,113	102 213

a The production of the Rosamundehütte in these years is lumped with that of the Beuthenerhütte.

b In addition to the spelter product there was an output of 1,093 tons of zinc dust in 1896; 993 in 1897; 886 in 1898; 915 in 1899, and 1,472 in 1900. This substance generally contains about 90% Zn. It is produced chiefly by the Godulla, Guidotto and Hohenlohe works.

The statistics of the Société Anonyme de la Vieille Montagne are of special interest inasmuch as it is the largest zinc-producing company of the world. The following table is compiled from the annual official reports of that company:

STATISTICS OF THE SOCIĒTÈ ANONYME DE LA VIEILLE MONTAGNE.

(Capital stock 9,000,000 francs.)

Year	Metric tons spelter produced	Metric tons sheet zinc produced	Metric tons zinc white produced	(*a*) Average price of spelter—francs	Gross profit—francs	Dividends paid—francs
1894	54,839	54,515	7,669	....	....	....
1895	64,497	57,000	8,988	353.26	....	2,250,000
1896	68,581	66,893	8,777	402.75	6,328,726	2,925,000
1897	68,604	68,024	8,357	434.85	6,647,641	3,375,000
1898	69,351	68,745	8,894	507.80	6,953,074	4,050,000
1899	69,672	58,369	7,937	616.50	8,788,639	4,612,500
1900	69,846	66,122	9,111	503.20	5,314,983	2,925,000

a Average price per metric ton.

It will be observed that the combined production of sheet and zinc white is greater than the spelter product of the company. This is explained by the fact that the Vieille Montagne purchases spelter from other producers.

V.

ANALYSIS OF ZINC ORES AND PRODUCTS.

In the control of a zinc smeltery the most important chemical determinations that have to be made are for zinc, iron, lead and sulphur. Less frequently it is necessary to make analyses for cadmium, manganese, lime, magnesia and silica. Occasionally the chemist may be called upon for an analysis of the coal and refractory material which are employed in the works, but the composition of those substances having once been satisfactorily determined, they are usually assumed to be uniform. At works where the coal supply has to be purchased, it would be advantageous to make frequent determinations of at least the tenor of the coal in moisture and ash, to furnish a guide in the purchase of what is the most important material consumed in the works, excepting ore. Such an investigation would, however, serve only for the information of the smelter, since there are practically no coal companies in the United States at least which will make any guarantee as to the quality of their product, and the consumer is frequently compelled to content himself with what is available, especially where the coal supply is necessarily derived from a particular and limited district.

Determination of Zinc.

There are many methods for the determination of zinc, both gravimetric and volumetric, but in technical practice only the volumetric methods are employed, they being quite as accurate and much more rapid than the standard gravimetric methods. Consequently only the volumetric methods will be described in this chapter, and the chemist who is interested in the gravimetric should refer to the standard textbooks of Fresenius, Cairns, Rose and others. There are two methods of volumetric analysis which are commonly employed: (1) titration with a standardized solution of potassium ferrocyanide, and (2) titration with a standardized solution of sodium sulphide; in Europe the sodium sulphide method is still generally in vogue, although the potassium ferrocyanide titration is meeting with some favor. The sodium sulphide method is cumbersome and troublesome and decidedly inferior to the ferrocyanide method in points of accuracy, simplicity and speed. This, however, is simply a matter of opinion, as to which chemists may disagree. Even in the United States there are some who prefer the sodium sulphide method to the ferrocyanide.

Potassium Ferrocyanide Method.—This method is based on the fact that zinc salts in an acid solution decompose potassium ferrocyanide, forming a white insoluble zinc compound; and that an excess of the ferrocyanide can be detected by the brown coloration produced with uranium acetate or nitrate. Unfortunately this method does not, like many volumetric processes, lessen the necessity for a complete separation of other metals which appear in solution with the zinc, since most metals give precipitates with potassium ferrocyanide in zinc solutions, but it will give, with proper precautions, results which will agree as closely with those obtained by any of the standard gravimetric methods as two gravimetric determinations of the same sample will agree with each other, assuming that the proportion of zinc in the sample be not less than 4%.[1] H. van F. Furman expressed the opinion, based on numerous experiments, that once the zinc is obtained in solution in the proper form, its percentage may be more safely determined by this volumetric method than it can be by precipitation and subsequent ignition and weighing.[2]

To prepare the standard solution of potassium ferrocyanide, 88 g. of the chemically pure salt[3] are dissolved in 2,000 c.c. of water. It is best to prepare the solution at least one day before standardization. In standardizing, two portions of 0·200 g. each of pure zinc oxide, which has previously been ignited to convert any zinc carbonate into oxide, are dissolved in 10 c.c. of strong, pure chlorhydric acid. Then add 7 g. of ammonium chloride (which must be chemically pure) and 250 c.c. of hot water and titrate with the ferrocyanide. As the end-point is neared, from time to time a drop of the solution is removed on the glass stirring-rod and added to a drop of a solution of pure uranium acetate (or nitrate) on a porcelain plate. So long as there is not an excess of ferrocyanide in the solution, the uranium acetate will retain its yellow color; but so soon as the ferrocyanide is in slight excess, the uranium acetate will turn a light brown, the shade being darker according to the quantity of ferrocyanide in excess. The titration should be carried to the point where a slight brownish tint is produced with the drop of uranium acetate. About 16 c.c. will be required and nearly that amount may be run in before making a test, but the titration should be finished carefully by testing after each additional

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[1] See Proceedings of the Colorado Scientific Society, June, 1892; and the Columbia School of Mines Quarterly, XIV, i, 40; also ibid. XXI, iii, 267 to 272.

[2] Manual of Practical Assaying, third edition, p. 205.

[3] This should be tested especially to prove the absence of potassium ferricyanide.

drop of ferrocyanide. As soon as a brown tinge is obtained note the reading of the burette and then wait a minute or two to observe if one or more of the previous tests have not also developed a brown tinge. Usually the end-point will be found to have been passed by a test or two, and the proper correction must then be applied to the burette reading. A further correction must be made for the quantity of ferrocyanide required to produce a brown tinge under the same conditions when no zinc is present. In calculating the result, it will be reckoned that 200 mg. of zinc oxide contain 160·688 mg. of zinc; 1 c.c. of ferrocyanide solution made up with 44 g. per liter will equal about 0·01 g. of zinc, or about 1% when 1 g. of ore is taken for assay.

The uranium acetate solution is prepared by dissolving a sufficient quantity of the salt in water to give a rather strong solution, which is clarified by adding a few drops of acetic acid. This solution should be kept in a small well-stoppered bottle in a dark place, since it is decomposed by the action of sunlight. The solution of potassium ferrocyanide should be kept in a tightly stoppered green glass bottle. This solution will keep for a long time without alteration, providing the bottle be well stoppered and be not exposed to strong sunlight, but it is advisable to restandardize it every two or three weeks.

Von Schulz and Low Process of 1892.—The best method for the technical determination of zinc in ores was the subject of a report by a committee of the Colorado Scientific Society, which was read at a meeting of that society June 11, 1892. That committee decided in favor of a method of determination by titration with potassium ferrocyanide as described by Messrs. von Schulz and Low, of Denver, Colo., and their method, or a modification of it, is now generally employed in the United States. In the method as originally described by von Schulz and Low, the ore is decomposed only by means of nitric acid saturated with potassium chlorate; this single treatment is sufficient for most ores. In the modification described by Furman, 1 g. of finely pulverized ore is digested in a 3·5 or 4 in. casserole with 15 c.c. of aqua regia and evaporated nearly to dryness. If the ore be incompletely decomposed by that treatment, which will seldom happen, however, the solution must be evaporated to dryness, the silica dehydrated, and the insoluble residue fused with carbonate and nitrate of soda, the fused mass being then digested with water and acid and the silica separated in the usual manner. In this case the first and second filtrates are combined, nitric acid is added, and the solution is evaporated nearly to dryness.

In either case the subsequent procedure is the same. To the nearly dry mass 25 c.c. of a saturated solution of potassium chlorate in nitric acid, prepared by shaking an excess of the crystals with strong, pure nitric acid in a flask, is added gradually. The dish should not be covered at first, but should be warmed gently until violent action is ended and greenish fumes cease coming off. The dish is then to be covered with a watch glass and its contents boiled down to dryness, but overheating or baking is to be avoided; a drop of nitric acid adhering to the convex side of the watch glass does no harm. When the preliminary treatment with aqua regia is omitted, the ore is digested directly with 25 c.c. of the chlorate solution in nitric acid, which is evaporated as above described.

The dish is then permitted to cool sufficiently to be handled with comfort and 7 g. of ammonium chloride, 15 c.c. of strong ammonia water and 25 c.c. of hot water are added to its contents. The latter are boiled for one minute, the dish or casserole being covered during that operation, and are then stirred with a rubber-tipped glass rod in order to insure that all solid matter on the cover, sides and bottom be either dissolved or disintegrated. The contents of the dish are next filtered into a flask of about 250 c.c. capacity and the insoluble matter remaining on the filter is washed several times with a hot solution of ammonium chloride, containing 10 g. of the salt in 1,000 c.c. of water. This ammonium chloride solution is most conveniently heated in a wash bottle; it should be boiling hot and made slightly alkaline with ammonia.

If the addition of ammonia to the casserole has produced a considerable precipitate of ferric hydrate, alumina hydrate, etc., the quantity of zinc hydrate that will be carried down with it will be too great to be disregarded, and it must be recovered by redissolving the precipitate and throwing it down again.[1] This is done most conveniently and with sufficient accuracy for ordinary technical purposes by transferring the first precipitate from the filter to the original decomposing vessel, employing a spatula and wash bottle for that purpose and using as little water as possible. The excess of water is evaporated off and the precipitate is then treated with the mixture of potassium chlorate and nitric acid as before, and subsequently with ammonia, ammonium chloride and hot water, the original procedure being followed a second time. The second precipitate having been filtered off and washed properly with the ammonium chloride solution, the filtrate is added to that which

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[1] E. Jensch has found that the retention of zinc by the precipitate produced by ammonia in the analysis of zinc ores is mainly due to the presence of much alumina in the ore. Repeated re-solution and re-precipitation, however, effectually overcome the evil. Four re-precipitations sufficed to give a precipitate of iron and alumina free from zinc in the worst instances (Zeits. f. angew. Chem., 1899, XX, 465-467). The quick method of re-precipitation described above appears to answer, however, all the purposes of ordinary technical analysis.

was first obtained. If the precipitate of ferric hydrate, etc., obtained in the first procedure be small, the quantity of zinc dragged down with it will be sufficiently minute to be ignored and the retreatment of the precipitate may be omitted. In the case of pure zinc ores, such as those of the Joplin district, the percentage of iron is so small that the second treatment is unnecessary.

In case there is no copper in the solution filtered off from the precipitate of ferric hydroxide, chlorhydric acid should be added to the point of neutralization, using a piece of litmus paper as a guide, and then 10 c.c. in excess. Pains should be taken to have that degree of acidity in all determinations made against the same standard. An excess of chlorhydric acid is necessary to prevent interference of lead, but too great an excess leads to inaccurate results. If the presence of copper is manifested by a blue coloration of the solution, the latter must be neutralized with chlorhydric acid and after addition of an excess, say 10 c.c. of the acid, the copper must be precipitated with sulphureted hydrogen, or by means of aluminum foil, thin sheet lead, or granulated lead. The last is the most convenient and is most commonly employed. From 20 to 40 g. are added to the contents of the flask, which should be at a temperature of about 70° C., and shaken vigorously until the copper in the solution is precipitated, after which it should be perfectly colorless.

The solution which should still be rather hot is now ready for titration. In performing the latter, it is a good plan to pour off about one-half of the solution and titrate rapidly until the end-point is passed, thus discovering approximately the percentage of zinc in the sample. The remainder of the solution is then added to the first portion and the titration is finished carefully, ordinarily by additions of two drops of ferrocyanide at a time. The addition of ferrocyanide should be stopped at the first appearance of a brown color and the reading of the burette should be taken. This reading is subject to the same corrections as were recited in connection with the standardization of the ferrocyanide solution.

Gold, silver, lead, copper, iron, manganese and the ordinary constituents of ores do not interfere with the above scheme. Cadmium behaves like zinc, and if it be not removed by a special process, the result of the determination will be too high. When cadmium is known to be present, it may be removed, together with the copper, by precipitation with sulphureted hydrogen, and the titration for zinc may be made upon the properly acidified filtrate without the removal of the excess of gas. There seems to be no simpler way to remove cadmium. If the copper be precipitated either by sulphureted hydrogen or aluminum foil, it may be redissolved in nitric acid and a rough determination of copper may be made from that solution. When copper is present in large quantity and is precipitated by sulphureted hydrogen, it is likely to carry down some zinc with it, except under nicely drawn conditions. Beringer recommends therefore that in the analysis of brass the copper be precipitated by electrolysis of a nitric acid solution before proceeding to the titration of the zinc.

Sulphide ores can ordinarily be decomposed by a simple digestion with the solution of potassium chlorate in nitric acid. In the case of slags different methods must be employed and it appears to be necessary to dehydrate the silica before finally taking the zinc into solution, inasmuch as gelatinous silica is likely to retain some zinc. Von Schulz and Low made the following notes as to details in the process:[1] Acids destroy the delicacy of the uranium test and for that reason a strong aqueous solution of the acetate is used. By having the zinc solution only faintly acid the brown coloration in the end-test becomes almost instantaneous and no previous test will develop a color. Under these conditions, however, lead is apt to interfere seriously and the presence of an excess of acid is therefore a necessity. When a strong solution of uranium acetate, not acidified, is used as indicator, the error caused by the excess of acid in the zinc solution amounts to only two drops of ferrocyanide, which may be allowed for, and the brown tinge develops so rapidly that the end-point is seldom passed by more than one test. When an ore contains but little copper, the granulated lead used to precipitate it frequently coheres in lumps that may hold zinc solution. These lumps are most easily broken up after a little of the ferrocyanide has been added. They appear to cause no appreciable error in the work. Lead shot or thin sheet lead may be used if preferred and may be cleaned with strong nitric acid and used repeatedly, but it is simple and more satisfactory to employ granulated lead and throw it away after use.

The method described above, although not free from defects, has proved well adapted for rapid and accurate work and has been in general use in the United States from the time of its first public description, nine years ago; no better method has since been offered. It is, of course, quite obvious that the method is not of universal application, but must be modified to suit the particular conditions. Sometimes a fusion may be necessary to effect complete decomposition, or perhaps a preliminary treatment with chlorhydric acid may be required, in which case, however, all chlorhydric acid must be expelled before beginning the regular treatment. The operator is expected to recognize such cases and apply the remedy. He is cautioned, however, to be very careful about modifying the method except from actual

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[1] Proc. Colo. Scientific Society, IV, 181.

necessity; most of the failures result either from inattention to details or the introduction of fancied improvements.

Von Schulz and Low Process of 1900.—The latest form of the von Schulz and Low method was described by A. H. Low in the *Journal of the American Chemical Society,* XXII, iv, 198, April, 1900, as follows:

"To standardize the solution of potassium ferrocyanide, made up with 22 g. of the salt per liter, weigh carefully about 100 mg. of pure zinc and dissolve in 6 c.c. of strong chlorhydric acid, using a 400 c.c. beaker. Then add about 10 g. of ammonium chloride and 200 c.c. of boiling water. Titrate with the ferrocyanide solution until a drop, when tested on a porcelain plate with a drop of a strong solution of uranium nitrate, shows a brown tinge.[1] About 20 c.c. of ferrocyanide will be required, and accordingly nearly that quantity may be run in rapidly before making a test, and then the titration should be finished carefully by testing after each additional drop. Instead of using a single drop of the zinc solution for the test, the reaction is much sharper if several drops are placed in a depression of the plate and tested with a single drop of a strong uranium solution. As this is near the end of the titration the quantity of zinc lost thereby is insignificant. As soon as a brown tinge is obtained, note the reading of the burette and then wait a minute or two and observe if one or more of the preceding tests do not also develop a tinge. The end-point is usually passed by a test or two and the burette reading must be corrected accordingly. A further correction must be made for the quantity of ferrocyanide required to produce a tinge under the same conditions when no zinc is present. This is only one or two drops. One cubic centimeter of the standard solution will equal about 0·005 g. of zinc, or about 1% when 0·5 g. of ore is taken for assay.

"To 0·5 g. of ore in a 250 c.c. pear-shaped flask, add about 2 g. of potassium nitrate and 5 c.c. of strong nitric acid. Heat until the acid is about half gone and then add 10 c.c. of a cold saturated solution of potassium chlorate in strong nitric acid and boil to complete dryness. It is usually necessary to manipulate the flask in a holder over a naked flame to avoid loss by bumping. The boiling may be conducted rapidly, and toward the end it is best to heat the entire flask so as to expel every trace of liquid. The potassium nitrate serves simply as a diluent of the dry residue and

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[1] Observe that Mr. Low now recommends the use of uranium nitrate instead of uranium acetate as indicator. Other chemists agree with him in that respect. A. C. Langmuir in a paper read before the New York Section of the American Chemical Society, March 18, 1901, stated that he had found uranium acetate to be an unreliable indicator in the presence of small quantities of chlorhydric acid; uranium nitrate in the form of a 3% or 4% solution is much more satisfactory.

insures the completeness of the subsequent extraction of the zinc. Cool sufficiently and add 30 c.c. of a prepared ammoniacal solution and heat to boiling. This solution is made by dissolving 200 g. of ammonium chloride in a mixture of 500 c.c. of strong ammonia water and 350 c.c. of water. Boil the contents of the flask very gently for about two minutes and then filter through a 9 cm. paper and wash with a hot solution of ammonium chloride containing about 100 g. of the salt and 50 c.c. of strong ammonia water to the liter. Collect the filtrate in a 400 c.c. beaker. The insoluble residue should be completely disintegrated and any ferric hydroxide present should appear of a fine sandy nature. Place a bit of litmus paper in the filtrate (not necessary if much copper is present) and neutralize carefully with chlorhydric acid, finally adding 6 c.c. of the strong acid in excess. Dilute to about 150 c.c. and add 50 c.c. of a cold saturated solution of hydrogen sulphide. Heat nearly to boiling and the solution is ready for titration. If more convenient, or apparently advisable, pass a current of hydrogen sulphide gas through the hot solution diluted to 200 c.c. Copper and cadmium, which are interfering metals, are thus precipitated. Unless in large amount they need not be filtered off. Practically no zinc is precipitated with the copper under these conditions and the discoloration of the liquid by even 10% of copper does not badly mask the uranium test.

" Titrate the hot solution as follows: Pour off about one-third and set it aside in a beaker. Titrate the remainder more or less rapidly, according as much or little zinc is indicated, until the end-point is passed, using the uranium indicator as in the standardization. Then add the greater part of the reserved portion and continue the titration with more caution until the end-point is again passed. Finally add the last of the reserved portion and finish the titration carefully, ordinarily two drops at a time. Make corrections of the final reading of the burette precisely as in the standardization. The true end-point is always slightly passed, and, after waiting a minute, it is usually sufficient to deduct for as many drops as show a brown tinge and one test additional.

"When precipitating with hydrogen sulphide it is a matter of considerable importance to have the solution of a definite degree of acidity. Cadmium and copper are to be precipitated, while it is better to retain lead in solution and not unnecessarily blacken the liquid with its sulphide. If there is enough acid to prevent the precipitation of its sulphide the lead will not interfere in the titration. On the other hand too much acid will prevent the precipitation of the cadmium. It will not come down from a boiling hot solution containing 5% of strong chlorhydric acid, but it is readily precipitated from a 3% solution, while lead is not, if the liquid be nearly boiling. It is therefore recommended to have an excess of 6 c.c. of strong chlorhydric acid in the final bulk of 200 c.c. of solution.

"Arsenic, when present in large amount, sometimes makes trouble by retaining iron in the ammoniacal solution. No attention need be paid to arsenic unless its presence in excess is thus indicated. In such a case begin anew and give the ore a preliminary treatment as follows: To 0·5 g. of ore in the flask add 10 c.c. of strong chlorhydric acid and 1 c.c. of bromine. Warm very gently for several minutes to decompose the ore without loss of bromine and then boil rapidly to complete dryness. The arsenic will thus be sufficiently expelled. Then add the potassium nitrate and nitric acid and proceed in the usual manner.

" In the case of ores that are free from cadmium, or where cadmium may be neglected, the copper may be readily precipitated without the use of hydrogen sulphide as follows: after neutralizing the ammoniacal filtrate from the insoluble residue, acidify with an excess of 10 c.c. of chlorhydric acid and add about 30 g. of granulated test lead. Heat nearly to boiling and stir the lead about until all the copper is precipitated. Then dilute to 200 c.c. and titrate as described, without removing the lead and precipitated copper."

Stone's Process for the Analysis of Manganiferous Zinc Ore.—G. C. Stone described a modification of the original von Schulz and Low method, which presents advantages in the analysis of manganiferous zinc ores, like those of New Jersey.[1] According to his method, sulphide ores are best dissolved in chlorhydric acid and potassium chlorate, care being taken to have sufficient acid present to insure keeping all the manganese in solution. Oxides, carbonates and silicates are dissolved in chlorhydric acid and boiled with the addition of potassium chlorate. Ores containing franklinite or rhodonite must first be fused with sodium carbonate and evaporated to dryness with chlorhydric acid to insure thorough decomposition, being then redissolved with chlorhydric acid in slight excess and boiled with potassium chlorate to oxidize the iron. Iron and alumina are precipitated with a solution of barium carbonate, which is prepared by warming the pure salt suspended in water with 2% or 3% of its weight of barium chloride; this converts the alkaline carbonate present into chloride, while the barium chloride remaining in the solution does not interfere with the process. The barium carbonate must be free from ammonia, and in effecting the solution it should be warmed for several hours. The solution from which the iron and alumina are precipitated should not contain a large quantity of free acid. The iron must all be in the ferric state.

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[1] Journal of the American Chemical Society, XVII, 473, June, 1895.

The thoroughly oxidized solution of the ore is washed into a 500 c.c. flask, cooled, and barium carbonate suspended in water is added until the precipitate curdles, an excess doing no harm. The precipitate is then filtered off and the solution is made up to 500 c.c. The solution should be filtered from the iron immediately, in order to avoid precipitation of zinc, and should be titrated as soon as filtered; if it be necessary to let the solution stand it must be made slightly acid, else some zinc will precipitate. Aliquot portions are taken for titration. One portion, which should contain between 0·01 and 0·04 g. of manganese, is diluted to about 200 c.c., heated nearly to boiling in a white porcelain dish, and titrated rapidly with the standard potassium permanganate solution, with very vigorous stirring. The greater part of the permanganate necessary should be added rapidly, or the manganese oxide is apt to stick to the sides of the dish, making it difficult to see the pink color of the solution at the end; therefore if the percentage of manganese in the sample be not approximately known it is best to make a rough preliminary titration, running in 1 or 2 c.c. at a time. In a second portion, made slightly acid with chlorhydric, the zinc and manganese are titrated together by standard potassium ferrocyanide solution. Inasmuch as manganous ferrocyanide is soluble in a large excess of chlorhydric acid, a large quantity of the latter is to be avoided, but 5 c.c. added to 100 c.c. of the solution do not cause any appreciable error. The solution is titrated cold. If manganese be present in appreciable quantity the color of the precipitate will darken as the ferrocyanide is run in and suddenly change to a light greenish yellow shortly before the end is reached. If lead be present in the solution, it must be sufficiently acid to prevent precipitation of that element by the ferrocyanide. A proportion of 5 c.c. of chlorhydric acid in 100 c.c. of solution is sufficient to prevent that and is not so much as to cause any appreciable error on account of the solubility of manganous ferrocyanide. A dilute solution of cobalt nitrate is employed as indicator. A drop of the cobalt solution is placed on a white porcelain plate and a drop of the solution to be tested is added so that the drops touch, but do not mix; a faint greenish line immediately shown at the junction of the drops marks the end reaction. The solution should not be warmer than the ordinary temperature of the laboratory.

The manganese is precipitated as $Mn_3Fe_2(CN)_{12}$, wherefore a quantity of potassium ferrocyanide that will precipitate four atoms of zinc will precipitate only three atoms of manganese. The calculation of results is illustrated by the following example: "The strength of the ferrocyanide solution was 1 c.c.=0·00606 g. zinc=0·00384 g. manganese. The strength of the permanganate solution was 1 c.c.=0·001 g. manganese; 2·5 g. of ore were dissolved and the solution was diluted to 500 c.c.; 50 c.c. of this solution were taken for the determination of manganese and 100 c.c. for the determination of zinc. Inasmuch as 18·45 c.c. (=7·38% Mn) were used in the first and 27·85 c.c. were used in the second titration, it is necessary to deduct 9·61 c.c. (for the 0·0369 g. of manganese present in the 100 c.c.) from 27·85 c.c., leaving 18·24 c.c. for the zinc, equal to 0·11053 g., or 22·11% Zn."

This method has been used with very satisfactory results in the laboratories of the New Jersey Zinc Company, the determinations of both zinc and manganese by it agreeing closely with those by gravimetric methods. In titrating manganese with potassium ferrocyanide solution, the uranium salts cannot be used as indicators because they react on the precipitated manganese ferrocyanide. Cobalt nitrate was found to be the best indicator. The solution of it should be dilute. In making the tests on the porcelain plate care should be taken that the drops do not mix, which would make it impossible to observe the reaction. If the color at the junction of the drops does not show at once, the end-point is not reached. Often when near the end it will show after standing a few seconds, but at the actual end-point it shows as soon as the drops touch. The ferrocyanide solution is best made up with about 30 g. per 1,000 c.c. It is standardized by titrating solutions containing about 0·1 g. of zinc, made slightly acid with chlorhydric, and kept at about the same volume as is used in the analysis. The quantity of ferrocyanide necessary to give a reaction with cobalt in this volume of acidulated water must be determined, and the amount so found deducted for each titration; it is about 0·7 c.c. for a volume of 140 c.c. The solution of potassium permanganate is made up with 1·99 g. per 1,000 c.c., which makes 1 c.c.=0·001 g. of manganese. It is standardized against iron in the usual manner, the manganese equivalent being found by multiplying the iron value by 0·294646.

Additional Notes Respecting the Ferrocyanide Method.—Among the other precautions to be observed in executing the ferrocyanide assay for zinc are to have approximately the same bulk of solution in all the titrations; to have the same quantity of HCl present; and to perform the titration at about the same temperature. If there be too large an excess of acid, or if the solution be too hot, a decomposition is likely to take place, resulting in the liberation of chlorine, which may be known by the solution becoming yellow or yellowish green. On the other hand, cold solutions should not be titrated; they give results that are too high. In a well-done assay the precipitate should always be white and the solution colorless or nearly so. The bluish color which the precipitate frequently manifests is due to the presence of traces of iron.

L. L. de Koninck and Eugene Prost, after an elaborate investigation of the volumetric determination of zinc by potassium ferrocyanide, came to the conclusion that the reaction is more regular and its termination more sharply marked if the ferrocyanide is always in excess with respect to zinc—e.g., if the zinc solution should be run into the ferrocyanide. This involves inverse titration, which is not very practicable, or else to titrate back. The latter method has the advantage of showing the approaching termination of the reaction by the diminution of the intensity of color produced by the indicator; for these reasons de Koninck and Prost give preference to it.[1]

E. H. Miller and E. J. Hall investigated the influence of various substances likely to be present when zinc is estimated by titration with potassium ferrocyanide and uranium acetate is used as indicator, such substances being added in successively increased quantity to constant known amounts of zinc solution which were titrated with ferrocyanide of a definite strength, uniform conditions of temperature and volume being carefully maintained. The following conclusions were drawn: (1) Salts, such as calcium chloride and sodium citrate, and acids, especially chlorhydric, disturb the end reaction by their solvent effect on uranium ferrocyanide. (2) Zinc ferrocyanide diminishes this effect in the case of chlorhydric acid; the correction established from blank tests is therefore slightly too great. (3) Ammonium chloride, not exceeding 17 parts, does not affect the accuracy of the method, but has some unknown action on zinc ferrocyanide. (4) Aluminum sulphate, in considerable quantity, renders the results unreliable. (5) Twelve parts of concentrated chlorhydric acid, sp. g. 1·2, suffice to prevent interference by lead. (6) Antimony gives high results, but small quantities of bismuth have no influence.[2]

SODIUM SULPHIDE METHOD.—In the technical laboratories of Belgium, France, Germany, Sardinia and Spain the determination of zinc in ores and furnace products is commonly made by titration with a standardized solution of sodium sulphide, the methods employed being more or less modifications of that which is known by the name of Schaffner. In the United States sodium sulphide titration is employed by a comparatively few chemists, who prefer it to the ferrocyanide method.

The estimation of zinc by titration with sodium sulphide depends upon the power of that reagent to throw down a white precipitate of zinc sulphide from an ammoniacal solution. Various indicators are employed to show when the zinc has been entirely precipitated and sodium sulphide is in excess. A solution of ferric chloride is frequently employed. Upon its

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[1] Chemical News, July 9 and 16, 1897.

[2] Columbia School of Mines Quarterly, 1900, XXI, iii, 267-272.

addition to the ammoniacal solution of zinc, flakes of ferric hydrate are produced, which retain their color until all the zinc has been precipitated, but blacken immediately in the presence of an excess of sodium sulphide. Instead of ferric chloride, solutions of nickel sulphate or lead acetate may be used, drops of those solutions being tested on a porcelain plate in the usual manner; an excess of sodium sulphide causes a black precipitate to be formed. Acetate of lead paper is also sometimes employed.

The solution of ferric chloride is prepared by dissolving 3 g. of iron wire in chlorhydric acid and a little nitric acid and diluting to one liter. The sodium sulphide solution is prepared by dissolving about 100 g. of crystallized sodium sulphide in 2,500 c.c. of water. The solution should be filtered or decanted in order to remove any black precipitate which may be formed in small quantity.

The sodium sulphide solution is standardized against metallic zinc or zinc oxide dissolved in chlorhydric acid containing a small quantity of nitric acid. The solution of zinc is diluted, in a flask, to about 500 c.c. and ammonia in excess is added, after which 5 c.c. of the ferric chloride solution is introduced. The solution of sodium sulphide is then run slowly from a burette into the cold ammoniacal solution, which should be agitated continuously, until the ferric hydrate is blackened.

In determining the zinc in ore, from 0·5 to 1 g. is weighed up and heated with chlorhydric acid and a little nitric acid, a large excess of acid being avoided. When decomposition is completed, the solution is diluted and ammonia and ammonium carbonate are added in excess. The solution is then heated gently for about 20 minutes, filtered into a flask and the residue on the filter well washed with hot ammoniacal water. To 5 c.c. of the ferric chloride solution ammonia is added in sufficient quantity to precipitate the iron, after which the whole is poured into the ammoniacal zinc solution; this procedure prevents the coagulation which would be likely to occur if the ferric chloride were added directly. When the solution is cold, the titration with sodium sulphide is performed, observing the precaution to keep the contents of the flask in continuous agitation.

When iron is present in the ore in only a small quantity the assay may be made without filtering off the hydroxide which is precipitated upon addition of the ammonia, but it is preferable to remove it and it is essential to do so if the quantity is considerable, since the precipitated ferric hydrate is likely to retain zinc oxide; in the latter case the precipitate should be redissolved and reprecipitated, the second solution being added to the first. If manganese is present in the ore complete precipitation is insured by the addition of a few drops of bromine water to the ammoniacal solution. If the presence of copper is manifested by the blue coloration of the solution, that metal must be removed by one of the conventional methods. If silver is present in sufficient quantity to vitiate the result it should be separated by filtration from the chlorhydric acid solution, before addition of the ammonia. Lead and cadmium do not interfere with the accuracy of the sodium sulphide assay; the former remains insoluble as sulphate or if taken into solution is precipitated by the ammonium carbonate, while cadmium is insoluble in the ammoniacal solution. This is an advantage of the sodium sulphide method over the ferrocyanide in the case of ores which contain cadmium, inasmuch as the special procedure for the elimination of the cadmium is unnecessary.

Another method of sodium sulphide titration was described by H. Nissenson and B. Neumann in *Chemiker Zeitung,* XIX, 1624, and the *Journal of the Society of Chemical Industry,* Jan. 31, 1896, p. 52. The following is an abstract of their paper:

"In the Schaffner process, the titration is made in an ammoniacal solution, and the indicator may be ferric hydroxide suspended in the solution, or ferric chloride (Streng), nickel chloride (Künzel), cobalt chloride (Deus), alkaline lead tartrate (F. Mohr), lead acetate (Fresenius), sodium nitroprusside (C. Mohr), thallium nitrate (Schröder), or lead carbonate (Schott). But of all these, only the first (freshly precipitated ferric hydroxide) and the last are generally used. In the analysis of blende, 1 g. of the ore (or 0·5 g., if it contain more than 25% Zn) is heated in a flask with 12 c.c. of strong chlorhydric acid, until all hydrogen sulphide has been driven off; the solution is then peroxidized with 3 c.c. of nitric acid, and after a short time 7 c.c. of sulphuric acid (1:2) are added, and the whole is evaporated until sulphuric acid fumes are observed. After cooling, the liquid is diluted and filtered. If cupriferous, the solution is boiled before filtration with 5 to 7 c.c. of sodium thiosulphate solution (1:10) until no more smell of sulphur dioxide can be detected. The precipitate which contains the silica, lead, and copper is filtered off, and the filtrate is peroxidized with bromine water, and then treated with 25 c.c. of ammonia (sp. gr. 0·925) for the separation of iron, manganese and alumina. After boiling and filtering, the precipitate is redissolved in chlorhydric acid (or in aqua regia, if highly manganiferous) and again precipitated.

"The mixed filtrates are made up to 500 c.c. and set aside for 12 to 18 hours for the removal of the great excess of ammonia. Simultaneously, a quantity of pure zinc, approximately equal to that in the ore under determination, is weighed accurately, dissolved in a similar quantity of chlorhydric and nitric acids, mixed with 25 c.c. of ammonia, made up to 500 c.c.

and set aside. Coda has shown that, to insure accuracy, it is not necessary to employ sulphuric acid for the solution of the check zinc.

"A shorter method, employed by the authors, consists in taking double the quantities above quoted all through, dissolving and proceeding as above, but boiling with sulphuric acid only for five or ten minutes (not to dryness) until red fumes cease to be evolved, making up to 500 c.c. after adding the ammonia, and filtering off only 250 c.c., which is then made up again to 500 c.c. for titration. The result may be a little low, but the error does not exceed 0·1%, while the titration is better effected, owing to the absence of great excess of ammonia.

"The success of the whole process depends upon attention to details in the titration. The sodium sulphide solution should contain 35 g. of the crystallized salt per liter, which is equivalent to about 0·01 g. Zn per cubic centimeter, but the titer rapidly alters on exposure to air, which necessitates the simultaneous performance of a check test with pure zinc. The burette may with advantage have a side-tube directly connected with the bottle containing the stock of standard solution, in order to minimize the action of the air when the burette is filled. Care must be taken that the color of the spot upon the lead paper, which marks the end of the reaction, shall be identical in the ore- and check-solutions, as in that case the variation in the sensitiveness of the eyes of certain operators in regard to the detection of the first darkening of the paper is eliminated. Two drops should be allowed to fall successively, after the lapse of a few seconds, upon the same spot upon the paper, the drops being taken from the solution by means of a narrow glass tube, which also serves as a stirrer. Toward the end of the operation, a test should be made after each fresh addition of 0·1 c.c. and very thorough stirring must be effected each time. A high temperature (even that of a hot summer day) and a great excess of ammonia alike interfere with the sharpness of the spotting reaction. The titration must, of course, be made in a room of which the air is quite free from hydrogen sulphide.

"When copper is present it is possible to make a determination in the above way on a part of the solution, deducting from the result the equivalent of copper found colorimetrically in another portion; but this is not recommended, as the presence of copper sulphide interferes with the sharpness of the lead-paper indications. For approximate and rapid work with cupriferous smelting products, the separation of copper may be avoided by adding to the ammoniacal solution, before titration, sufficient potassium cyanide solution to remove all but a trace of the blue color from the liquid. A trace of blue is left to insure that no excess of cyanide shall have been added, as this would interfere with the accuracy of the result, through the formation of a double cyanide of zinc.

"The sodium sulphide method if properly conducted will give results in exact concordance with those obtained gravimetrically."

H. Pellet found that the ammoniacal salts, produced when ammonia is employed in the estimation of zinc in plumbiferous minerals by the Schaffner method to neutralize the acid solution before titrating the zinc with sodium sulphide, retain in solution the lead present and so falsify the titration.[1] In order to avoid such error he substituted potash in that operation, leaving the liquid just sufficiently acid to prevent the deposition of the zinc. The solution is then poured into ammonia, agitated, and filtered, the filtrate being titrated with standardized sodium sulphide, one drop of ferric chloride (20%) solution being added as indicator. By this means the method retains its accuracy even in presence of 40% of lead. It is found that the volume of sodium sulphide solution required per unit of zinc increases slightly with the dilution of the liquid. An important point is to pour the neutralized solution into the ammonia, and thereby obviate the deposition of zinc oxide, which would otherwise have to be redissolved. In dissolving the mineral, Pellet prefers to oxidize the iron present by means of potassium chlorate instead of nitric acid.

E. G. Ballard recommends the use of a bright silver plate for determining the end-point in the titration of zinc with sodium sulphide solution. If the titration is done with a cold solution, a large excess of ammonia is to be avoided.[2]

OTHER METHODS.—For a description of other volumetric methods and the standard gravimetric methods the reader is referred to the various text-books of analytical chemistry. Some notes from recent chemical literature are subjoined.

Titration with Standard Acid.—Dementief has proposed to determine zinc by adding to an acid solution an excess of NaOH, sufficient to redissolve the $Zn(OH)_2$. The solution is then divided exactly in halves; one is titrated with standard acid and tropæolin OO indicator, which gives $Zn(OH)_2+NaOH$; the other half is titrated with standard acid and phenolphthalein, which gives only the NaOH. The difference is the measure of the $Zn\ (OH)_2$ present.[3]

Titration with Standard Alkali.—P. H. Walker has described a method for the volumetric determination of zinc which is a modification of that

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[1] Bull. Assoc. Belge des Chim., XI, iv, 126-130.

[2] Journ. Soc. Chem. Ind., May 31, 1897.

[3] Journ. Soc. Phys. Chem. Russ., XXV, iii, 222.

devised by Stolba for the determination of magnesium. To the zinc solution, which should contain ammonium chloride, a large excess of ammonia is added, followed by a large excess of sodium phosphate solution. Chlorhydric acid is then gradually added until, after stirring, the solution remains milky. It should then be heated to about 75° C., and the gradual addition of acid continued, with constant stirring, until nearly complete neutralization is attained. By this means the precipitate becomes crystalline; after standing five minutes it should be filtered off and washed with cold water until the washings show only a faint trace of chlorine. The filter paper and precipitate are then returned to the beaker in which the precipitation was carried out, an excess of standard acid is added, and the exact point of neutrality is determined by means of standard alkali, using methyl orange as indicator.

The reaction which takes place is represented by the following equation:

$$ZnNH_4PO_4+H_2SO_4=ZnSO_4+NH_4H_2PO_4.$$

Calculation from the above equation shows that 1 c.c. of normal acid corresponds with 32·7 mg. of zinc.

The method gives good results. Since the zinc ammonium phosphate is not precipitated in presence of a large excess of ammonia, any magnesium present, which will be precipitated, may be removed by filtration, and the filtrate neutralized to throw down the zinc. Fairly good results are obtained by this method also in the presence of iron, calcium and magnesium, but any manganese must be previously separated, best by means of nitric acid and potassium chlorate.[1]

Titration with Sodium Thiosulphate.—R. K. Meade has described a method for determining zinc by titration with a standard solution of sodium thiosulphate. Manganese is first thrown down as dioxide by means of potassium chlorate and nitric acid. After filtering, the iron and alumina are separated by double ammonia precipitations. Calcium and magnesium are removed by the addition of a large excess of sodium arsenate, and then the zinc is precipitated by adding nitric and acetic acids. The iodine liberated by digesting this precipitate in an acid solution of potassium iodide is titrated with standard thiosulphate and the amount of zinc calculated. The time required for four determinations, without any effort at speed, was about eight hours. The ore tested contained 30·18% Zn; the method described gave 30·00, 29·98, 30·04 and 29·98%.[2]

Pouget has proposed to pass sulphureted hydrogen through a zinc solu-

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[1] Journ. Am. Chem. Soc., 1901, XXIII, vii, 468-470.

[2] Ibid., 1900, XXII, vi, 353.

tion containing sodium acetate, precipitating the zinc as ZnS; after boiling to remove the excess of H_2S the sulphur of the ZnS, and consequently the zinc, is estimated by the addition of iodine solution and titration with standardized sodium thiosulphate.[1]

P. Knaps has also described a method for the determination of zinc by means of iodine solution. Zinc sulphide, in the acetic solution in which it is ordinarily precipitated, is converted by iodine into zinc iodide and sulphur. If, however, more than 0·05 g. of zinc sulphide to 200 c.c. of water be present, the separated sulphur protects particles of it from the action of the iodine. This can be prevented by forming a precipitate, such as barium sulphate, in the liquid prior to the introduction of the hydrogen sulphide. Thus, from 10 to 20 c.c. of a solution of 150 g. of barium chloride in a liter, and the same quantity of sodium sulphate solution (200 g. per liter), are introduced, the zinc precipitated, and the excess of hydrogen sulphide removed by boiling. An excess of standard iodine solution is then added, the liquid is shaken for one or two minutes, and the excess of iodine is titrated with standard sodium thiosulphate solution. Ammonium salts do not interfere, and the results are accurate even in the presence of considerable quantities of manganese salts.[2]

Electrolytic Assay.—H. Paweck described a method for the estimation of zinc electrolytically, employing a platinum anode and a cathode consisting of an amalgamated brass wire gauze disk, 6 cm. in diameter, suspended from the center by a 1 mm. wire. The electrolyte, about 200 c.c. in volume, may be either acid or alkaline; a current of 3·6 volts is used. Results are said to be fairly satisfactory as to accuracy.[3]

Gravimetric Method by Precipitation as Sulphide.—J. Meunier states that the difficulty experienced in collecting zinc sulphide for gravimetric determination is overcome in the following simple manner: The zinc solution, preferably slightly warmed, is precipitated with ammonia, just sufficient of the precipitant being cautiously added to redissolve the hydrate at first formed. A slow current of sulphureted hydrogen is then passed through the solution until a drop of it, on a white tile, gives a blackish coloration with another drop of a solution of ferrous sulphate. When this occurs the whole of the zinc will have been precipitated, and is in a suitable condition for collection. The passage of the gas is immediately stopped and the zinc sulphide collected, washed, and dried in the usual way. It will be found that the filtrate will at once run perfectly clear, and that the precipitate may be rapidly washed, especially if warm solutions be employed. The

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[1] Comptes Rendus, CXXIX, 45.

[2] Chem. Ztg., XXV, 539-540.

[3] Zeits. f. Elektrochem., 1898, XVIII, 221.

presence of large quantities of other salts does not affect the ease and rapidity with which the process may be conducted.[1]

DETERMINATION OF CADMIUM.

There are numerous good methods for the determination of cadmium, but as in the case of zinc there is none better for ordinary technical purposes than the ferrocyanide method, which is applicable to cadmium in the same way as to zinc. Inasmuch as cadmium occurs generally with zinc and it is desirable to estimate both elements, a convenient combination of methods can be made whereby both can be determined from the same sample. To do that the sample should be treated as described under the caption of the von Schulz and Low process of 1900. The cadmium and copper sulphides precipitated by means of hydrogen sulphide are filtered off. The filtrate is used for the determination of zinc in the regular manner. The precipitate is washed with a solution of hydrogen sulphide and then dissolved in a little hot, dilute chlorhydric acid. The solution should then be further diluted and the copper precipitated by means of test lead or aluminum foil in the conventional manner. If no copper be present that step will of course be omitted.

Inasmuch as zinc may be to some extent dragged down with the sulphides precipitated with sulphureted hydrogen, it is advisable to redissolve them in chlorhydric acid and precipitate them a second time. Or the chlorhydric acid solution may be heated to boiling and then be poured into an excess of cold sodium hydrate solution, whereby insoluble cadmium hydroxide is thrown down, while the zinc hydroxide is kept in solution by the excess of caustic soda. The cadmium precipitate is then filtered off, redissolved in chlorhydric acid and proceeded with in the usual manner.

The solution freed from copper is then ready for titration with the standard solution of potassium ferrocyanide, using uranium nitrate as indicator. The same solution may be used for cadmium as for zinc; the zinc equivalent in milligrams per 1 c.c. is to the cadmium equivalent as 65·4 : 112·3, those figures being respectively the atomic weights of zinc and cadmium. If many determinations of cadmium have to be made, however, it is preferable to prepare a special solution of potassium ferrocyanide, about 58% as strong as that used for zinc, and standardize it against a known weight of chemically pure, metallic cadmium.

Numerous methods for the determination of cadmium which are employed in European laboratories are described in *Das Cadmium,* a pamphlet

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[1] Comptes Rendus, CXXIV, xxi, 1151.

by E. Jensch, reprinted from *Sammlung Chemischer und Chemischtechnischer Vorträge,* III, vi, (1898). In general, these are gravimetric methods in which the cadmium is weighed as sulphide on a tared filter, or as oxide obtained by calcination of a final precipitate of hydroxide or carbonate.

E. H. Miller and R. W. Page have recently investigated several methods for the quantitative determination of cadmium.[1] They found the electrolytic method to be very accurate if a large excess of potassium cyanide and the presence of other salts be avoided. The carbonate method is the most troublesome and the least accurate. A very accurate method is the precipitation of cadmium from a neutral solution (cold) by a large excess of di-ammonium hydrogen phosphate and either weighing the precipitate as $CdNH_4PO_4,H_2O$ on a tared filter (after drying at 100° to 103° C.), or dissolving it in dilute nitric acid and igniting to pyrophosphate.

DETERMINATION OF LEAD.

The zinc smelter is frequently required to determine the percentage of lead in ore and spelter. The latter is commonly done gravimetrically, weighing the lead as sulphate as described in a subsequent section. The percentage of lead in ore is determined most conveniently by volumetric analysis, the permanganate, the molybdate and the ferrocyanide methods all being satisfactory with respect to speed, simplicity and the degree of accuracy required for technical purposes.

Ammonium Molybdate Method.—This method, which is due to H. H. Alexander, is based upon the fact that ammonium molybdate added to a hot solution of lead acetate will precipitate lead molybdate, $PbMoO_4$, which is insoluble in acetic acid, while an excess of the ammonium molybdate will give a yellow color with a freshly prepared solution of tannin. The standard solution of ammonium molybdate is prepared by dissolving 9 g. of the salt in 1,000 c.c. of water, affording a solution of which 1 c.c. is equal to about 0·01 g. of lead. If the solution be turbid, it can be clarified by adding a few drops of ammonia water. The tannin solution is prepared by dissolving 1 g. of tannin in 300 c.c. of water; it is used on a porcelain test plate. To standardize the molybdate solution, weigh out 0·3 g. of pure lead sulphate and dissolve it in hot ammonium acetate, acidify the solution with acetic acid and dilute with hot water to 250 c.c. Heat to boiling and then titrate with the molybdate solution until all the lead is thrown down as a white precipitate, which is ascertained by testing the solution from time to time with a drop of the tannin solution on a porcelain plate. So long as

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[1] School of Mines Quarterly, 1901, XXII, iv. 391-398.

the lead is in excess no color is produced, but as soon as the lead is precipitated the excess of the molybdate shows a yellow color with the tannin solution.

In determining the lead in an ore or furnace product by this method 1 g. of the sample is decomposed by heating in a casserole with 15 c.c. of strong nitric acid and 10 c.c. of strong sulphuric acid. When the nitric acid is completely expelled, which is indicated by the appearance of fumes of sulphuric anhydride, the casserole is removed from the heat and cooled. Its contents are then diluted with hot water, stirred thoroughly and boiled until all soluble sulphates are dissolved. Then filter, leaving as much of the precipitate in the casserole as possible and wash twice with hot, dilute sulphuric acid and once with cold water. Then add to the lead sulphate remaining in the casserole hot ammonium acetate, pour the hot solution on the filter, and allow it to run through into a clean beaker. This operation is repeated until the lead sulphate is completely dissolved. Then wash out the casserole thoroughly with hot water into the filter. Dilute the solution to 250 c.c. with hot water and heat to boiling. Then titrate with the ammonium molybdate.[1]

A determination can be made readily by means of the molybdate method in 30 minutes. It gives excellent results in the presence of the most common impurities, but fails when lime is present, owing to the precipitation of calcium molybdate. In the case of calcareous ores a combination of the permanganate method and the molybdate method may be made, the former being followed up to the point of precipitation of the lead by zinc. The precipitated lead should then be dissolved in dilute nitric acid, using as small quantity as possible, after which the solution should be made alkaline with ammonia, and then neutralized with acetic acid. The titration with ammonium molybdate may then be proceeded with.[2]

Potassium Permanganate Method.—In the employment of this method, which was devised by Mr. F. C. Knight, 1 g. of ore is treated with 15 c.c. of strong nitric acid and 15 c.c. of strong sulphuric acid in a 4 in. casserole, covered with a watch glass. Heat until decomposition is complete and fumes of sulphuric anhydride appear. Then remove the casserole and when it is cool add gradually about 50 c.c. of cold water, heat to boiling and filter immediately. Wash well with boiling water, slightly acidified with sulphuric acid, and finally with pure hot water. Rinse the insoluble residue into a beaker of about 200 c.c. capacity, using not to exceed 50 c.c. of water; add 5 c.c. of concentrated chlorhydric acid, cover with a watch glass and

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[1] H. H. Alexander, Eng. & Min. Journ., Apr. 1, 1893, p. 298.

[2] H. van F. Furman, Manual of Assaying, third edition, p. 142.

boil for five minutes. The sulphates of lead and lime pass into solution. If much silica and barium sulphate be present it is best to filter and wash well with boiling water. The filtration must be done rapidly. Small quantities of silica do not interfere, but large quantities prevent a subsequent precipitation of the lead in one spongy mass.

Dilute the solution with water to about 100 c.c., keeping it hot but not boiling. Add 2 g. of granulated zinc (free from lead), which will precipitate the lead in the solution as a metallic sponge. When the action of the acid on the zinc has apparently ceased add 0·5 g. more of zinc and allow to stand for five minutes. Then boil the solution and add 10 c.c. of concentrated chlorhydric acid, which dissolves the remainder of the zinc and leaves the lead sponge floating on the surface of the solution. Decant off the solution, wash the lead sponge with cold water and press it out flat with the finger. Then dissolve it in 1 c.c. of concentrated nitric acid and 20 c.c. of hot water, add a slight excess of sodium carbonate[1] and redissolve the precipitated lead carbonate by the addition of 5 c.c. of strong acetic acid. Add 20 c.c. of 95% alcohol, heat the solution to 65° C. and precipitate the lead with a saturated solution of pure crystallized oxalic acid. The lead comes down immediately as a dense, white, crystalline precipitate. Stir briskly until the precipitate settles, leaving a perfectly clear supernatant liquid. Filter and wash the precipitate three times with a hot mixture of alcohol and water (1:1) and then four times with hot water alone. In washing the precipitate it is advisable to use a fine jet, keeping the stream on the filter and not allowing it to flow on the glass, since otherwise the precipitate is likely to creep up and thereby occasion loss. When thoroughly washed the precipitate is rinsed into a flask or beaker with about 50 c.c. of hot water and 5 c.c. of concentrated sulphuric acid. The oxalic acid combined with the lead and set free by the sulphuric acid is then determined by titration with a standard solution of potassium permanganate, in the same way as for the determination of lime. The potassium permanganate solution should be prepared by dissolving 1·58 g. of $KMnO_4$ in 1,000 c.c. of water. One cubic centimeter of a solution of that strength is equivalent to about 0·05 g. of lead. The solution is standardized against oxalic acid, the equivalent in terms of lead being calculated by multiplying the oxalic equivalent by 1·6428.[2]

There are no metallic impurities likely to be encountered in zinc ores or furnace products which interfere with the accuracy of this method. A determination can be made in 35 to 40 minutes. The disadvantage of the

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[1] The solid salt is preferable to the solution.

[2] Proc. Colo. Scientific Society, IV, 215 to 223.

method is that the result is liable to be a little low on account of the incomplete precipitation of the lead as oxalate. According to Mr. Knight about 99·75% of the lead present is obtained. This inaccuracy is of minor importance in connection with the low percentages of lead that the zinc smelter finds in his products and otherwise the permanganate method is a very good one.

Potassium Ferrocyanide Method.—The estimation of lead by titration with a standard solution of potassium ferrocyanide is highly recommended by Furman, in his *Manual of Practical Assaying,* who states that 1 g. of the sample should be treated in the same manner as with Alexander's method up to the point where the lead is precipitated as sulphate. The precipitate of lead sulphate should be washed and then transferred into the flask or beaker with the use of the minimum quantity of water. Add 30 c.c. of a saturated solution of ammonium carbonate, heat quickly to boiling and boil at least one minute in order to decompose any calcium sulphate which may have formed. It is essential that calcium sulphate be converted into carbonate, since otherwise it would react upon the dissolved lead and cause low results. Filter and wash thoroughly with hot water containing a little ammonium carbonate. Dissolve the washed carbonate of lead in strong acetic acid, dilute to about 180 c.c. and titrate with the standard solution of potassium ferrocyanide in the same manner as for zinc. The ferrocyanide solution should contain 14 g. of the salt per liter, and when of that strength one cubic centimeter will be equivalent to about 0·01 g. of lead.

DETERMINATION OF IRON.

The methods of determining the percentage of iron in ores are so numerous that they cannot be described properly except in a special treatise on analytical chemistry. It will be sufficient here to refer only to the method of titration with potassium permanganate, which is the quickest and most easily performed of all the methods of determining iron and if properly carried out gives results which answer all the purposes of technical analysis. The potassium permanganate method is based on the measurement of the volume of the reagent required to oxidize the iron from the ferrous to the ferric state, a pink color being imparted to the solution the moment the conversion is complete. It is essential therefore after effecting a solution of the iron of a sample to reduce it completely to the ferrous condition before subjecting it to titration.

The standard solution of potassium permanganate is prepared by dissolving 5·9 g. of the pure crystallized salt in 1,000 c.c. of water. This solution should be made up at least 48 hours before standardizing, should be kept in a well-stoppered bottle and should be well shaken before use. The solution, if carefully kept (action of direct sunlight should be avoided), does not alter, but it is well to titrate it afresh occasionally. It is standardized against metallic iron, the latter being used generally in the form of fine soft iron wire which contains 99·6 to 99·7% Fe. About 200 mg. of the latter are weighed out and dissolved in 10 c.c. of dilute sulphuric acid by gently warming in a 250 c.c. flask. To the solution, diluted with about 100 c.c. of water, a small quantity of pure granulated zinc is added, which has the effect of reducing ferric sulphate to ferrous sulphate, the solution becoming colorless when the reduction is complete. The loss of color is not, however, a sufficiently accurate guide, since some of the ferric iron may remain when the solution is apparently devoid of color. The completeness of the reduction should be tested therefore by removing a drop of the solution on a glass rod to a porcelain plate and adding a drop of ammonium sulphocyanate solution; if the iron be completely reduced to the ferrous condition the drop will remain colorless, while if any ferric salt be present it will turn red, the depth of the color depending on the quantity of ferric iron present. The reduction being complete, the contents of the flask are further diluted by the addition of cold water and then decanted from the undissolved zinc into a large beaker, the flask and the zinc being well washed and the washings added to the original solution. The solution is then made up to about 700 c.c. and about 20 c.c. of dilute sulphuric acid are added. The titration with potassium permanganate is then proceeded with, and the iron equivalent of the standard solution is calculated from the number of cubic centimeters used. The solution containing the iron should not be permitted to stand too long before beginning the titration, since some of the iron is likely to be reoxidized by exposure to the air; all evolution of gas from the action of sulphuric acid on minute particles of zinc decanted over with the washings should have ceased, however, before beginning the titration. The end-point of the reaction is manifested when one drop of the permanganate produces a pink coloration in the solution under analysis which cannot be stirred out. The fact that the color may disappear after a time need cause no uneasiness, however, since that is one of the results of the natural conditions involved in the reactions.

The potassium permanganate solution may also be standardized against oxalic acid, 250 mg. of the latter being weighed out and dissolved in 100 c.c. of water. Then add 6 to 8 c.c. of pure concentrated sulphuric acid, heat to about 70° C. and run in the permanganate solution until a permanent color is obtained. The color will disappear very slowly at first, but after a small quantity of the permanganate solution has been added it will disappear rapidly. Owing to the smaller bulk of the oxalic acid solution as compared with the iron solution, when the standardization is performed by the former method, one or two drops less of the permanganate solution are required and allowance should be made for them. In standardizing against oxalic acid the equivalent of iron is calculated on the assumption that 250 mg. of oxalic acid=222 mg. of iron. Care must be taken that the oxalic acid employed be pure and perfectly dry. Instead of oxalic acid, the permanganate may be standardized against crystallized ammonium oxalate [$(NH_4)C_2O_4+H_2O$], which can be easily prepared in the pure state and keeps well; 142·08 parts of that salt correspond to 112 parts of iron.

To determine the percentage of iron in blende, dissolve 1 g. of ore in a small casserole with 4 c.c. of strong chlorhydric acid, 10 c.c. of strong nitric acid and 15 c.c. of dilute (1:1) sulphuric acid added in the order mentioned. Heat on a sand bath or iron plate until dense white fumes of sulphuric anhydride are evolved, and then for about three minutes longer in order to be sure of removing the last traces of nitric acid, which is essential. Then cool and add about 50 c.c. of water. Pour the solution into a flask, washing out the casserole and adding the washings to the solution in the flask. Add metallic zinc to reduce the ferric sulphate and then proceed in the same manner as in the standardization against metallic iron.

If the proper precautions are observed, iron in a chlorhydric acid solution may be determined by titration with potassium permanganate, although it is frequently asserted that this method is not accurate. Furman states, however, that if only a small quantity of chlorhydric acid be present and the solution is extremely dilute (700 c.c.) and cold and moreover contains a large excess of sulphuric acid (usually 20 c.c. of concentrated acid), the results obtained are as reliable as when sulphuric acid is used as the solvent. As a further precaution some chemists add a few cubic centimeters of a saturated solution of manganous sulphate before titration, but that is unnecessary if the above conditions are carried out; however, inasmuch as the addition of manganous sulphate can do no harm it is well to use it when in doubt or when a considerable quantity of chlorhydric acid has been employed.[1] Beringer has also shown that good results may be obtained with the use of a chlorhydric acid solution, if the acid be not greatly in excess.[2]

To determine the percentage of iron in oxidized ores dissolve 1 g. in a small casserole with 5 c.c. of chlorhydric acid. Dilute with water, filter and

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[1] Op. cit., p. 173.

[2] Textbook of Assaying, p. 200.

reduce the iron in the solution by means of metallic zinc. Then titrate as in standardizing against iron. Most ores will yield their iron by a simple boiling with chlorhydric acid. If the sample be a furnace product which perhaps cannot be decomposed in that manner, use the three acids as in dealing with a sulphide ore.

In case the ore is to be digested with chlorhydric acid and that solution is to be titrated, it is advisable to dissolve the iron for standardization in chlorhydric acid also in order that the conditions may be precisely the same. However, if the solutions contain only from 5 to 10% of free chlorhydric acid the results will be the same as those obtained from a sulphuric acid solution.

DETERMINATION OF LIME AND MAGNESIA.

Lime is most easily determined by precipitating it as oxalate from solutions, decomposing the calcium oxalate by dilute sulphuric acid and titrating the oxalic acid thus liberated with potassium permanganate. The permanganate solution may be the same as is used for the determination of iron and may be standardized in the same manner. If it be made up of such strength that 1 c.c.=0·01 g. of iron, 1 c.c. will be equal to 0·005 g. of lime. There is no convenient method for the volumetric determination of magnesia, and that is commonly estimated gravimetrically as described by Fresenius.

Lime.—In determining lime weigh up 1 g. of ore and digest with 6 or 7 c.c. of chlorhydric acid, adding a few drops of nitric acid to aid in the decomposition of any sulphides that may be present. If there be no sulphides the nitric acid may be omitted. In the case of sulphide ores it is best to decompose with a mixture of 4 c.c. of strong chlorhydric acid and 3 c.c. of strong nitric. The solution is then evaporated to dryness and heated to about 120° C. to drive off the free chlorhydric acid and render the silica insoluble, after which the soluble matter is taken up again by boiling with a few cubic centimeters of chlorhydric acid, diluted with water and filtered. If the ore be argillaceous and likely to contain lime as silicate the sample must first be fused with 5 or 6 g. of sodium carbonate, because calcium silicate is not completely decomposed by chlorhydric acid. The fused mass is then digested with water and chlorhydric acid in slight excess and the solution evaporated and taken up again as before described. In case barium sulphate and lead sulphate be absent, the insoluble residue may be washed, ignited and weighed as silica. Lead sulphate, which will be formed by the oxidation of the sulphide by nitric acid, may be dissolved by treatment with a warm solution of ammonium acetate.

The solution in which lime is to be determined must be free from lead, hich if present must be precipitated by sulphureted hydrogen or otherwise. ron and aluminum are then removed by precipitation with ammonia, and nally calcium is precipitated as oxalate by the addition of ammonium xalate or oxalic acid. The solution in which the precipitation is made hould not be much over 100 c.c. in bulk. If oxalic acid be used as the recipitant there should be a sufficient excess of ammonia present to insure hat the solution will be alkaline after the addition of the acid. If magnesia be present the ammonium oxalate should be in considerable excess n order to insure that all the magnesia will be converted into oxalate, hich is soluble. The contents of the beaker must then be heated to boiling or a few minutes and filtered immediately, whereby there will be no danger f the calcium oxalate running through the filter paper. If there be no agnesia present the precipitate is washed with boiling water until the ashings cease to give a reaction for oxalic acid. The filter paper with its ontents is then spread out on a watch glass somewhat larger than the paper, nd the precipitate is washed off with hot water from a wash-bottle with a ne jet into a beaker, the washing being finished with some dilute sulphuric cid used in the same manner.

If magnesia be present it is safest to redissolve the precipitate in chlorhydric acid, which is done conveniently by spreading the filter paper on a atch glass and washing the precipitate into a beaker as described above, nishing with chlorhydric acid instead of sulphuric. The chlorhydric acid hould be hot and dilute and as little of it should be used as possible. The hlorhydric acid solution should then be diluted with boiling water to about 0 c.c., and after being made alkaline with ammonia about 20 c.c. of ammonium oxalate solution should be added and the contents of the beaker eated nearly to boiling. The precipitate is then to be filtered off and ashed into a beaker as described above.

To the precipitate washed into the beaker with the aid of hot water and ilute sulphuric acid about 10 to 15 c.c. of sulphuric acid should be carefully dded and the solution heated to about 70° or 80° C., when it is ready for itration with the standard solution of potassium permanganate. The latter hould be run in with moderate rapidity and with constant stirring of the olution until a permanent pink tinge is produced, indicating the end-point f the reaction. The percentage of lime is then calculated from the number f cubic centimeters of the permanganate solution used. The results are ery accurate. The titration is very conveniently performed in an evaporatng-dish, the opaque white of which constitutes a good background in determining the end-point of the reaction.

Magnesia.—The filtrates from the precipitation of the lime as oxalate may be employed for the determination of magnesium, which is commonly done by precipitating that metal as ammonium-magnesium phosphate, which is converted into magnesium pyrophosphate ($Mg_2P_2O_7$) by ignition and weighed as such. The precipitation is effected by the addition of an excess of a 10% solution of hydrodisodium phosphate (Na_2HPO_4).[1] Both the solutions should be cold. After adding the phosphate solution the contents of the beaker should be stirred with a glass rod, care being taken not to touch the sides of the beaker, which would cause crystals of ammonium-magnesium phosphate to adhere thereto so firmly that it is difficult to remove them. It is advisable to set the beaker in a dish containing cold water and stir frequently, since a low temperature and agitation of the solution facilitate the precipitation. The solution must then be allowed to stand, without warming, for some time (up to 12 hours) to insure complete precipitation. When 1 g. of sample has been used for the analysis it can be assumed that 10 c.c. of the hydrodisodium phosphate solution of the strength specified above will be sufficient to precipitate all the magnesium if the latter be not in excess of 10%.

The solution from which magnesia is to be precipitated must be free from iron and alumina and the metals which are thrown down by sulphureted hydrogen, including zinc and manganese. The analytical scheme for substances containing those metals will, therefore, necessarily provide for their removal, which is of course best done before the precipitation of the lime. After the metals of the second group have been thrown down by means of sulphureted hydrogen, iron and aluminum are best precipitated as basic acetates. Manganese is then removed from the filtrate by oxidation with bromine water, after which the solution is acidified with acetic acid and the zinc precipitated with sulphureted hydrogen.[2]

The precipitate of ammonium-magnesium phosphate is collected on a small filter and washed with dilute ammonia (two or three parts of water to one part of ammonia of 0·96 sp. gr.) until the washings no longer show a precipitate by the addition of a few drops of silver nitrate solution (acidified with nitric acid) or only a very slight opalescence is manifested. The precipitate is then dried and transferred to a tared platinum crucible; the filter paper is burned on the lid of the crucible until white and its ash added to the contents of the crucible. The crucible must then be ignited strongly until its contents are white or nearly so. If the magnesium pyrophosphate is dark colored, moisten with a few drops of nitric acid, dry carefully and

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[1] One cubic centimeter of such a solution will precipitate 0·0112 g. of MgO. Magnesium pyrophosphate contains 36·036% MgO.

[2] Vide Fresenius, Quantitative Chemical Analysis, John Wiley & Son's edition of 1883, pp. 517, 519 and 755.

ignite again. The weight of the ignited precipitate multiplied by 0·36036 will give the weight of the equivalent in magnesia, from which the percentage in the ore can be calculated.

This method, if properly executed, gives very accurate results. The precipitate must be washed completely, but not overwashed, and the washing-water must always contain the prescribed quantity of ammonia. Instead of igniting and weighing the precipitate of pyrophosphate, it may be dissolved in chlorhydric acid, and after neutralization with ammonia and the subsequent addition of sodium acetate and acetic acid, it may be titrated with a standard solution of uranium acetate, using potassium ferrocyanide as indicator, but this method does not appear to offer any advantage.

DETERMINATION OF SULPHUR AND SULPHURIC ACID.

The accurate determination of sulphur, especially when it exists in the sample in the form of sulphide, is no easy matter. It is essential in all the methods to get the sulphur into solution as sulphate, which in many cases it is difficult to do; the effect of interfering impurities has to be carefully guarded against; and finally it is doubtful if a thoroughly good volumetric method has yet been devised, in the absence of which the slow and troublesome details of gravimetric analysis have to be performed. Even in the case of the shortest methods of technical analysis, therefore, with their inherent inaccuracies, the determination of sulphur is a comparatively slow and tedious process.

GRAVIMETRIC METHODS.—The gravimetric methods for the estimation of sulphur are based upon effecting a solution as sulphate to which barium chloride is added, throwing down barium sulphate which is filtered off and weighed as such. The decomposition of the sulphides is effected by fusion with alkaline nitrates or potassium hydrate (Fahlberg-Iles method); or by decomposition with acids and oxidizing agents. Among the latter methods, decomposition with nitric acid and potassium chlorate is most commonly employed; aqua regia is also frequently used, although it is apt to fail to effect complete conversion of the sulphur into sulphuric acid; while pyrites and blende may be decomposed by digestion at gentle heat with water and the gradual addition of bromine. If the sulphides have been prepared in the wet way, good bromine water is sufficient to oxidize them.

Fusion with Potassium Hydrate.—This method is doubtless the most accurate of the technical methods for the determination of sulphur in ores and furnace products; it also requires the most time. In this method, one or two sticks of potassium hydrate, which must be free from sulphur, are fused in a silver crucible, heated by a spirit lamp (gas must not be used). When the alkali is in quiet fusion, the lamp is removed and the sample (1 g.) of ore is introduced and the fusion is then continued five to 30 minutes, or until decomposition is complete. The fused mass, after cooling, is digested in a beaker with warm water, which is boiled after the crucible has been removed and is then filtered, the insoluble matter being washed thoroughly with boiling water. From 20 to 40 c.c. of bromine water are added to the filtrate, which is then heated to about 90° C. and acidified with chlorhydric acid. The solution must next be evaporated to dryness to render silica insoluble, which is filtered off after a redigestion with dilute chlorhydric acid in the usual manner. The filtrate from the silica is then boiled and the sulphur is precipitated by the addition of a hot solution of barium chloride. When the precipitation is effected with boiling hot solutions, the barium sulphate comes down very quickly and can be filtered off rapidly and without danger that it will run through the paper. After a thorough washing the precipitate is dried and transferred into a crucible by gently rolling the paper with the fingers, the paper being then burned on the cover of the crucible and its ash added to the remainder of the precipitate. After ignition the precipitate is weighed as barium sulphate, which should be perfectly white in color. Unless the precaution is taken to remove silica as above described the result will be inaccurate, but if that precaution be observed this method, which is universal in its application, is entirely satisfactory in all respects save the length of time required.

Digestion with Nitric Acid.—This method is not so accurate as that involving the fusion with potassium hydrate, but is much more rapid and is consequently used commonly in technical laboratories. As described by Furman (op. cit., p. 90) it is performed as follows:

Digest 1 g. of the sample in a flask of 200 c.c. capacity with 3 to 4 g. of potassium chlorate (in the case of a heavy sulphide ore more potassium chlorate should be used) and 7 c.c. of nitric acid, about 3 c.c. of the acid being added at first and the remainder at short intervals, 1 c.c. at a time. When all the acid has been added heat to boiling and drive off all but 2 c.c. The solution should not show any globules of yellow sulphur, which would indicate imperfect oxidation.

After the flask has cooled, add about 50 c.c. of water and then an excess of saturated solution of sodium carbonate, which will precipitate the lead, iron, etc., and decompose the sulphates of lead and calcium which have been formed. Boil for 30 minutes to one hour, adding water to keep the bulk of the solution about the same. Then pass through a fluted filter into a beaker, wash until no traces of sulphuric acid are shown, acidify the filtrate with chlorhydric acid, boil to expel the carbonic acid and then precipitate with a solution of barium chloride, proceeding in the same manner as described in the case of fusion with potassium hydrate. If the ore contains barium sulphate the latter will remain undecomposed with the precipitate of mixed carbonates.

The presence of nitric acid or nitrates is objectionable in a solution which is subsequently to be precipitated with barium chloride, as is also an excess of chlorhydric acid. Consequently in analyzing blende and pyrites some chemists digest 0·5 g. of the ore with 10 c.c. of a mixture of three volumes of nitric acid and one volume of chlorhydric, heating gently at intervals; then evaporate to dryness, treat with 5 c.c. of chlorhydric acid and evaporate again. Take up with 1 c.c. of chlorhydric acid and 100 c.c. of hot water, filter and wash. Heat the filtrate, make slightly alkaline with ammonia, filter off the precipitated ferric hydrate, wash with hot water, evaporate to about 200 c.c. if necessary, make slightly acid with HCl and add the barium chloride solution.

VOLUMETRIC METHODS.—For the volumetric estimation of sulphur the latter has to be oxidized and dissolved as a sulphate in the same way as for gravimetric analysis. The aim is to substitute a simple titration of the solution for the tedious filtration of barium sulphate and ignition of the precipitate. Numerous processes have been devised, but none has been generally accepted as satisfactory. This subject was discussed at length by M. Felix Marboutin in *Moniteur Scientifique* of September, 1897, to which reference should be made by those who are particularly interested in it. Description will be limited in this treatise to two processes which seem to be most particularly adapted to the requirements of technical analysis.

Furman's Process.—H. van F. Furman has proposed a volumetric method for the determination of sulphur which is based on Alexander's method for the determination of lead by titration with a standard solution of ammonium molybdate.[1] In Alexander's method the lead is precipitated as sulphate, wherefore the percentage of sulphur can be calculated from the percentage of lead. The same solution of ammonium molybdate may be used as is employed for the estimation of lead, or a special solution of a strength which will enable the percentage of sulphur to be read directly from the burette may be prepared. In any case the solution will be standardized against metallic lead or lead sulphate in the same manner as for Alexander's method. The equivalent in sulphur may be calculated on the basis that lead sulphate contains 10·56% S.

If the sulphur of the sample under analysis has been brought into solution

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[1] Op. cit., p. 92.

by fusion with caustic potash and digestion with hot water, hydrogen peroxide is added to the filtrate to oxidize the potassium sulphide. The solution is then heated and acidified with a *slight* excess of nitric acid. To the hot solution add an excess of a solution of lead nitrate and filter, retaining as much as possible of the lead sulphate in the beaker. Wash by decantation with cold water until a reaction for lead is no longer obtained. Then dissolve the lead sulphate in hot ammonium acetate, acidify with acetic acid and titrate with the standard ammonium molybdate solution.

In case the sulphur has been brought into solution by digestion with nitric acid and potassium chlorate, acidify the filtrate from the precipitated carbonates with a slight excess of nitric acid, boil out the carbonic acid, precipitate the sulphuric acid with lead nitrate and proceed as above.

The sulphur may also be put into solution by fusing 1 g. of the sample with 10 g. of mixed sodium carbonate and potassium nitrate. Dissolve the fused mass, when cold, in hot water. Filter, acidify the filtrate with nitric acid, boil to drive out carbonic acid, precipitate the sulphuric acid with lead nitrate and proceed as above.

In all cases the reagents used should be carefully examined for the presence of sulphur.

Andrews' Process.—Max Reuter considers[1] that of all methods proposed for the estimation of combined sulphuric acid, Andrews' is the best, and if carried out exactly as described its accuracy leaves nothing to be desired; moreover, it is far more rapid than any gravimetric method and only one standard solution—viz., decinormal sodium thiosulphate—is required, which is an advantage over most of the other volumetric methods.

The principle of this method is as follows: From a boiling solution of the sample to be tested in which the sulphur has been put in the form of an alkali sulphate and should be present to the extent of about 0·07 g. of SO_3, the sulphur is precipitated by an excess (150 c.c.) of barium chromate dissolved in chlorhydric acid. The excess of the acid is neutralized by boiling with ammonia or with pure powdered chalk (preferably the latter) until there is a neutral reaction, the precipitate being removed by filtration and washed with hot water. After cooling, the filtrate which contains the alkaline chromate corresponding to the sulphate in the solution under examination is acidulated with 5 c.c. (not more) of strong chlorhydric acid, after which 20 c.c. of a 10% solution of potassium iodide are added and the liquid is allowed to remain five minutes in a covered beaker to insure complete reduction of chromic acid, during which time a current of carbonic

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[1] Chem. Ztg., 1898, XXII, 357.

acid is passed through to avoid atmospheric oxidation of the iodohydric acid set free. Finally it is diluted to 1 or 1·5 liters and titrated quickly with the standard thiosulphate solution, iodine being liberated by the free chromic acid. The reactions are represented by the following equations:

$$2Na_2SO_4+BaCrO_4=Na_2CrO_4+BaSO_4.$$
$$2Na_2CrO_4+2HCl=2NaCl+H_2O+Na_2Cr_2O_7.$$
$$Na_2Cr_2O_7+14HCl+6KI=2CrCl_3+6KCl+3I_2+7H_2O.$$

It results therefore that $2SO_3$ corresponds to 6I; or $I=\frac{1}{3}\ SO_3$. The following solutions are required for the analysis: (1) chlorhydric acid solution of 3 to 4 g. of pure barium chromate, which is prepared by precipitating potassium chromate with barium chloride and dissolving the washed precipitate in 30 c.c. of concentrated chlorhydric acid, after which the solution is diluted with water up to one liter; change in the degree of concentration ought to be avoided; and (2) a decinormal solution of sodium thiosulphate.

The use of only 5 c.c. of chlorhydric acid before the titration prevents the reappearance of the blue coloration after the titration is finished, which occurs if as much as 20 c.c. of acid be used. If all the indicated precautions are observed the results are very close. A dozen analyses can be made easily in an afternoon. Andrews' process was described originally in the *American Chemical Journal,* 1880, II, 567.

SPECIAL ANALYTICAL METHODS.

DETERMINATION OF ZINC IN ALLOYS.—A. C. Langmuir in the *Journal of the American Chemical Society,* XXI, ii, February, 1899, described the following method for the determination of zinc in alloys containing copper, tin, lead, iron and zinc: Dissolve in nitric acid, evaporate to dryness, and take up the residue with nitric acid if tin be present. Filter and separate copper and lead simultaneously by electrolysis. The solution should contain 5 to 10 c.c. of concentrated nitric acid in 150 to 200 c.c. of water. After removing and washing the electrodes carrying copper and lead, the solution is evaporated to dryness in a weighed platinum dish and the residue ignited and weighed. The residue is then treated with chlorhydric acid, and the iron, which is usually small in amount, is precipitated by ammonia. The precipitate is filtered off, ignited and weighed, its weight being deducted from that of the residue (zinc oxide) weighed in the platinum dish. If nickel be present the weight of the combined oxides may be taken and the nickel determined subsequently, the weight of the zinc being then found by difference.

ANALYSIS OF SPELTER.—In making a chemical examination of a sample of spelter the chief consideration is the percentages of various impurities which are likely to be present, wherefore it is customary to determine them directly, the percentage of zinc being then estimated by difference. The impurities which are generally to be looked for are lead, iron and cadmium; less frequently, arsenic, tin, and copper. In making an analysis of spelter it is advisable to weigh out a comparatively large quantity, say 10 or 20 g., so that results may be reported to at least three places of decimals.

Lead in spelter is conveniently determined by dissolving 10 g. of the sample in 100 c.c. of dilute chlorhydric acid (1:2), without heating, and decanting off the solution before the zinc is entirely taken up. The remainder is then dissolved in 2 c.c. of dilute nitric acid and the solution is evaporated until most of the acid is removed, after which it is diluted with water to about 30 c.c. and filtered off. The lead is then precipitated as sulphate by the addition of 10 c.c. of dilute sulphuric acid. The precipitate of lead sulphate may be filtered off and weighed as such, or it may be redissolved and titrated with ammonium molybdate. The filtrate from the lead sulphate, including the original solution decanted off, may be used for the determination of iron, the latter being precipitated as hydroxide by oxidizing the solution with a little bromine water and then adding 40 c.c. of ammonia water of 0·925 sp. gr. A systematic determination of all of the common impurities of spelter is provided for in the following scheme.

Tin.—The weighed sample is dissolved in dilute nitric acid, boiled and allowed to settle if there be formed a white insoluble substance. In the case of the latter it should be filtered off, washed, dried and weighed, this being oxide of tin.

Lead.—To the filtrate or the original clear solution add ammonia water and ammonium carbonate, boil, filter off the precipitate, wash with hot water, digest the precipitate with dilute sulphuric acid, filter, wash and weigh as lead sulphate, from which the percentage of lead may be calculated.

Iron.—Make alkaline with ammonia the filtrate from the lead sulphate and pass hydrogen sulphide through it. Filter off the precipitate and dissolve in chlorhydric acid, oxidize with nitric acid and precipitate again with ammonia. Wash, ignite and weigh as ferric oxide. Instead of weighing the ferric oxide, the percentage of iron may be determined by titration with potassium permanganate in the usual manner.

Arsenic.—To the filtrate from the sulphide of iron add chlorhydric acid in slight excess, filter off and wash the precipitate; rinse it back into the beaker, dissolve in nitric acid, filter off the sulphur and add ammonia in excess and magnesia mixture. Filter off the ammonium-magnesium arsenate and wash with dilute ammonia. Dry, ignite with nitric acid and weigh as magnesium pyrarsenate, from which the percentage of arsenic may be calculated, on the basis that magnesium pyrarsenate ($Mg_2As_2O_7$) contains 48·4% As. The magnesia mixture is prepared by dissolving 22 g. of magnesia in about 250 c.c. of dilute chlorhydric acid, then adding 5 g. more of magnesia, boiling and filtering; to the filtrate, 300 g. of ammonium chloride and 250 c.c. of strong ammonia are added, and the solution is then diluted with water to 2,000 c.c.

Copper.—To the filtrate from the ammonia and ammonium carbonate add sulphuric acid in small excess and treat with hydrogen sulphide. Filter off the precipitate, wash and rinse it into a beaker; then boil with dilute sulphuric acid and filter, saving the filtrate. Dry, burn the filter paper with the precipitate, treat with a drop or two of nitric acid, ignite and weigh as copper oxide (CuO).

Cadmium.—To the filtrate from the sulphide of copper add ammonia so as nearly to neutralize the excess of acid and treat with hydrogen sulphide. Filter off the precipitate, using a weighed filter, wash first with an acid solution of hydrogen sulphide and afterward with water, dry at 100° C. and weigh as cadmium sulphide; if free sulphur is suspected to be present, wash with carbon bisulphide before drying.[1]

Zinc.—Although the percentage of zinc in pure spelters is best determined by difference as described above, according to A. C. Langmuir in the *Journal of the American Chemical Society,* XXI, ii, February, 1899, in the case of impure spelters it is sometimes more convenient and equally accurate to determine the zinc directly. A convenient method is the following: Dissolve 0·5 g. of the sample in nitric acid and separate the lead electrolytically in a solution containing 10% nitric acid. A platinum cylinder should be used in order to collect the lead peroxide. The solution is then evaporated to dryness in a weighed platinum dish, and the residue ignited and weighed. Then dissolve in a little chlorhydric acid, reduce with two or three grams of granulated zinc, add sulphuric acid to complete the reaction, dilute to one liter with ice water and titrate with potassium permanganate. The weight of the iron found is then deducted from the weight of the residue in the dish. If cadmium, arsenic or other impurities be present in the spelter in more than traces, the above method is inapplicable unless modified to include treatment by hydrogen sulphide.

Doctor Langmuir in a paper read before the New York Section of the American Chemical Society, March 18, 1901, described a short method for the analysis of spelter, substantially as follows: "Cover the weighed

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[1] C. and J. J. Beringer, Textbook of Assaying, p. 224.

quantity of spelter with water, add a few drops of platinic chloride, and then sufficient sulphuric acid to start a moderate rate of solution. The zinc dissolves, leaving lead in the metallic state. Filter, dissolve the lead in nitric acid on the filter and convert to sulphate, as usual. The solution of zinc may then be at once titrated for iron by permanganate solution, after which cadmium is precipitated by hydrogen sulphide."

VALUATION OF ZINC-DUST.—The commercial value of this material depends upon the percentage of metallic zinc which it contains. Inasmuch as it invariably contains zinc oxide also, the ordinary methods for the estimation of zinc are inapplicable, since they give only the total zinc. There are numerous methods described for the determination of metallic zinc in this material, most of which depend upon the measurement of the hydrogen evolved by a treatment of the sample with acid, comparing the quantity with that given off by a corresponding weight of pure zinc. Other methods depend upon the measurement of the reducing power of the metallic zinc in the dust. One of the best methods for the valuation of zinc dust is considered to be that which has been described by Klemp.[1]

Klemp's method depends on the power of zinc to reduce potassium iodate to iodide in an alkaline solution. When the solution is afterward acidified, iodine is liberated and is distilled off from the mixture into potassium iodide solution, and titrated with sodium thiosulphate. From 0·5 to 1 g. of the sample is weighed out into a 200 c.c. stoppered flask. For every 0·1 g. of zinc present, 10 c.c. of a solution containing 370 g. of potassium hydroxide, or 300 g. of sodium hydroxide per liter, and 3 c.c. of a solution containing 15·25 g. of potassium iodide per 300 c.c. are measured into a beaker, and then poured into the flask. Some glass beads are added, and the flask stoppered and shaken for five minutes in the cold, no advantage being gained by heating it. Its contents are then washed into a 250 or 500 c.c. flask, and made up to the mark with water, from which 100 c.c. are transferred into the retort of a Topf's apparatus, dilute sulphuric acid is added, and the apparatus filled with carbon dioxide. A solution of potassium iodide is placed in the receiver, and the retort heated, at first gently, and then more strongly till the contents are perfectly colorless. The stream of carbon dioxide is continued throughout the distillation, which generally occupies about 20 minutes. The solution of iodine in potassium iodide is then transferred from the receiver into a flask, and standard solution of sodium thiosulphate is added in slight excess; the excess is titrated with weak standard iodine solution, starch being used as an indicator. After correcting for the excess, the quantity of zinc is calculated from the

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[1] Zts. f. analyt. Chemie, XXIX, 253.

quantity of thiosulphate used. Each 0·2 g. of iodine found corresponds to 0·25644 g. of zinc. Iron and lead do not interfere materially with the method, though results are a little lower in the presence of those metals. The results by this method agree well with those obtained by that of Fresenius, but are generally higher than those obtained by Drewson's and lower than those obtained by Topf's method (vide *Journ. Soc. Chem. Ind.*, IX, 968). For a volumetric method, depending on the amount of hydrogen evolved by the action of an acid on zinc-dust, vide *Journ. Soc. Chem. Ind.*, V, 145.

Another method consists in the treatment of the sample with a definite quantity of a standard solution of potassium chromate or bichromate, an excess of which is employed, and then adding an excess of dilute sulphuric acid. The reaction is expressed as follows:

$$3Zn+2CrO_3+6H_2SO_4=3ZnSO_4+Cr_2(SO_4)_3+6H_2O.$$

When the solution of the zinc is complete, the excess of chromate employed is determined by titration with a standard solution of ferrous sulphate.

The following method is due to A. Fraenkel: About 1 g. of the sample is weighed into a dry, well-stoppered flask of 200 c.c. capacity, and treated with 100 c.c. of a semi-normal solution of potassium bichromate and 10 c.c. of dilute sulphuric acid (1:3). The mixture is shaken for five minutes, then 10 c.c. more of the acid are added, and the mixture is again shaken for from 10 to 15 minutes. The zinc by this time will have entirely dissolved. The solution is transferred to a half-liter flask and diluted to the mark, 50 c.c. of it are taken and after the addition of 10 c.c. of potassium iodide solution (1:10) and 5 c.c. of sulphuric acid is titrated with decinormal sodium thiosulphate solution.[1]

A simple method described by A. R. Wahl consists in treating 0·5 g. of the sample with 25 c.c. of cold water in a 250 c.c. stoppered flask, and after suspending the dust by a thorough shaking 7 g. of ferric sulphate are added. The following reaction takes place:

$$Fe_2(SO_4)_3+Zn=ZnSO_4+2FeSO_4.$$

There is a gentle evolution of heat, and after shaking the flask about 15 minutes the zinc will be completely dissolved, the residue consisting of impurities. Strong sulphuric acid (about 25 c.c.) is then added and the flask is filled with water up to 250 c.c., of which 50 c.c. are taken and further

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[1] Mitt. des k. k. techn. Gewerbe Museums in Wien, 1900, X, 161.

diluted to 100 c.c. and then titrated with standard potassium permanganate, which measures the quantity of ferrous salt reduced by the zinc and therefore the percentage of the latter in the sample.[1]

DETERMINATION OF SULPHATES AND SULPHIDES.—It is frequently desirable to determine the relative proportion of sulphur which exists as sulphide and various sulphates especially in roasted ore. An estimation which is sufficiently correct for most technical purposes can be made in a simple manner. The neutral sulphate of zinc, iron and most of the other metals, including magnesium, which are likely to be met with in zinkiferous ores are readily soluble in cold water, wherefore they may be removed by a simple leaching with water. It is preferable not to boil the solution, since thereby basic sulphates are likely to be precipitated. After removal of the soluble sulphates, the sulphuric anhydride existing in the form of lead sulphate and the basic sulphates can be extracted by boiling with a solution of sodium acetate; or in case there be no necessity to determine separately the soluble sulphates, the sample may be treated directly with sodium acetate.

In examining a sample of roasted blende to determine the proportion of sulphur existing as sulphide and sulphates, boil 1 g. in a 200 c.c. flask with 50 c.c. of a 5% solution of sodium acetate; allow to settle and decant the solution through a filter; repeat the treatment with sodium acetate and then boil twice with distilled water. The sulphur occurring as sulphates will then be in the filtrate, while the sulphur as sulphide will be in the residue remaining on the filter. The former may be precipitated with barium chloride in the usual manner, care being taken to remove interfering impurities. The residue on the filter may be washed into a flask and treated with nitric acid and potassium chlorate to effect decomposition of the sulphides in the usual manner.

Steinbeck's method for the determination of basic sulphates consists of the addition to the residue, remaining after removal of the sulphates soluble in water, of three times its weight of sodium bicarbonate and enough water to dissolve most of the soda. This is allowed to stand for 24 hours with occasional shaking. The solution is then filtered off, acidified with chlorhydric acid, boiled to remove the carbon dioxide and finally precipitated with barium chloride. Sodium bicarbonate in solution reacts with all the basic sulphates and with lead sulphate, while it has no effect upon the sulphides. Calcium sulphate is decomposed by it, but barium sulphate is not.

DETERMINATION OF SULPHUROUS ACID IN ROASTING-FURNACE GAS.—In works where the sulphurous gas from the roasting furnaces is utilized for the manufacture of sulphuric acid it is important to control the process by

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[1] Journ. Soc. Chem. Ind., XVI, 15.

determining the percentage of sulphurous anhydride by volume in the gas. The following method is described by Mr. F. J. Falding:[1] The sample of the gas is most conveniently taken through a 1-in. pipe projecting into the flue leading to the Glover tower. The 1-in. pipe should be built into the wall of the flue. On the outside it is reduced to ¼ in., the complete arrangement being shown in the accompanying engraving. The rubber tube should be of sufficient length to permit considerable freedom of movement. It connects with a bent $^3/_{16}$-in. glass tube, of which one leg passes through the rubber stopper of an 8-oz. bottle of white glass. Another bent glass tube with equal legs is inserted through the other hole of the stopper and connected by a ¼-in. tube with a 4-qt. bottle as shown. The rubber tube leading from the 4-qt. bottle terminates with a piece of glass tubing drawn down to about $^1/_{16}$ to ⅛ in., which must be long enough to reach about 2 in. inside of a 250 c.c. graduated jar.

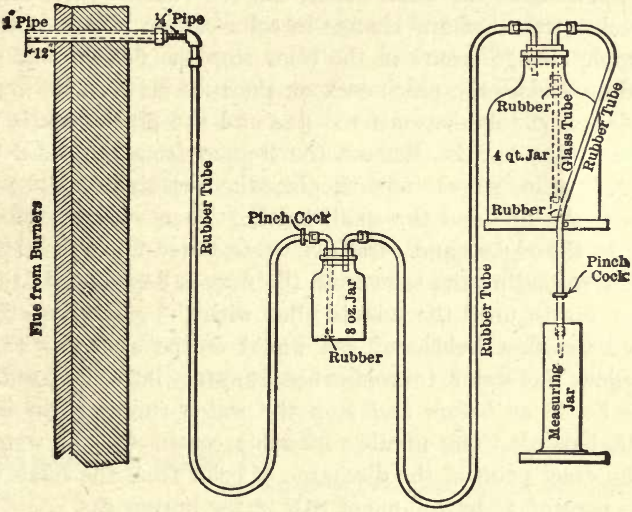

FIG. 8.—APPARATUS FOR THE DETERMINATION OF SULPHUROUS ACID.

In order to make the determination a solution of starch and a decinormal solution of iodine are required. These should be prepared as directed in Lunge and Hurter's *Alkali Maker's Handbook,* second edition, p. 171. The iodine solution will keep well in a dark, cool place. The starch solution is more likely to spoil, even when saturated with salt, and should be renewed more frequently.

In operation the large bottle is filled with water, care being taken to make

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[1] The Mineral Industry, VII, 698.

the stopper perfectly tight. The siphon is started by a slight suction through the nozzle and the pinch-cock is then closed. The 8-oz. bottle is filled about one-fourth full of clean water, slightly warmed in winter, into which about a teaspoonful of starch solution is poured. Then by means of a pipette 10 c.c. of iodine solution is added to the contents of the 8-oz. bottle. The rubber stopper is then replaced tightly and the pinch-cock between the flue and the small bottle is closed. The pinch-cock at the nozzle is then opened, allowing the water to waste. When the water ceases to run, proving the tightness of the stoppers and connections throughout the apparatus, the pinch-cock between the flue and the small bottle is opened. Then take the small bottle in the left hand, keeping the right hand on the pinch-cock at the nozzle, and shake the small bottle, not too violently, holding it to the light in such a way that any change in color can be readily noted. When a considerable change occurs in the color stop the flow of water with the right hand, and close the pinch-cock on the tube between the flue and the small bottle. The tube between the flue and the pinch-cock is now filled with the gas to be tested. Remove the stopper from the small bottle and add 10 c.c. of the iodine solution. Replace the cork tightly. Open the pinch-cock between the flue and the small bottle. Then with the pinch-cock at the nozzle in the right hand carefully waste water until the liquid in the glass tube, terminating the tube from the flue, is depressed to the bottom; or, in other words, until the tube is filled with the gas to its extreme end. Just before the first bubble of gas would escape and pass through the solution allow the water to commence running into the graduated jar. Shake the bottle as before and stop the water running the instant the color is discharged. The number of cubic centimeters of water the jar holds at the exact point of the discharge of color from the small bottle represents the percentage by volume of SO_2 in the burner gas.

Before making further tests the small bottle should be emptied and fresh water and starch solution used. Any intelligent workman can be taught to make the test, and after a little practice can perform a test in less than three minutes. He may either be provided with a percentage table or instructed to keep his gas between 128 and 138 c.c., or as may be required. The following table is useful for reference:

c. c. water in jar.....	82	86	90	95	100	106	113	120	128	138	148	160	175	192	212
% SO_2 by volume....	12·0	11·5	11·0	10·5	10·0	9·5	9·0	8·5	8·0	7·5	7·0	7·5	6·0	5·5	5·0

VI

PROPERTIES OF ZINC AND ITS ALLOYS.

Zinc is a bluish-white metal of shining luster, which according to the periodic arrangement of the elements by Mendeléef is classified in group II, corresponding to the symbol RO, the series in the order of atomic weights consisting of beryllium (9·1), magnesium (24·36), calcium (40), zinc (65·4), strontium (87·68), cadmium (112·3), barium (137·43), erbium (166) and mercury (200). Of these elements, magnesium, zinc and cadmium have remarkable points of resemblance in their chemical and physical properties, while three others, viz., calcium, strontium and barium (alternating with magnesium, zinc and cadmium in the periodic arrangement) present an equally striking similarity. Beryllium and mercury are both volatile metals. Erbium has not yet been isolated in its elemental form, and its position in Mendeléef's classification is doubtful. An arrangement of these elements in a form which more clearly shows their connection, was given by Professor T. W. Richards in the *Proceedings of the American Academy of Arts and Sciences,* March 9, 1898, and the *American Chemical Journal,* 1898, XX, vii, as follows:

$$\mathrm{H}(1\cdot0075)\left\{\mathrm{Be}(9\cdot1)\left\{\mathrm{Mg}(24\cdot36)\left\{\begin{array}{l}\mathrm{Ca}(40\cdot0) : \mathrm{Sr}(87\cdot68) : \mathrm{Ba}(137\cdot43) : \ldots\ldots\ldots \\ \mathrm{Zn}(65\cdot4) : \mathrm{Cd}(112\cdot3) : \ldots\ldots\ldots\ldots : \mathrm{Hg}(200)\end{array}\right.\right.\right.$$

Atomic Weight.—The atomic weight of zinc was formerly supposed to be 65 (vide Remsen, Eliot & Storer et al.) and that figure is commonly used at present, but later investigation has shown it to be about 65·4. F. W. Clarke adopts 65·41; T. W. Richards, 65·40; and the revising commission of the German Chemical Society, 65·4. Inasmuch as the solutions employed in the volumetric estimation of zinc are commonly standardized against zinc oxide, while in gravimetric analysis the metal is weighed as ZnO, it makes a slight difference whether the atomic weight of zinc be assumed as 65 or 65·4.[1] The atomic volume of zinc is 9·1. In its chemical relations it is

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[1] For example: If the atomic weights of oxygen and zinc be reckoned 16 and 65·4 respectively, the percentage of zinc in zinc oxide is 80·344; if the atomic weights be reckoned 16 and 65, the percentage is 80·247. If one gram of a sample yields

bivalent, combining with two atoms of chlorine and one atom of oxygen, forming chlorides and oxides which are analogous to those of magnesium and cadmium.

PHYSICAL PROPERTIES.—*The specific gravity* of cast zinc is 6·861 according to Brisson, 6·9154 according to Karsten, and 7·149 according to Matthiessen. According to Bolley the specific gravity of the metal when poured at or near its melting point and cooled slowly, is 7·145, and according to Rammelsberg, 7·128; when poured at red heat and cooled slowly, 7·120 according to Bolley, and 7·101 according to Rammelsberg; when poured at the melting point and cooled quickly, 7·158 (Bolley) and 7·147 (Rammelsberg); poured at red heat and cooled quickly, 7·109 (Bolley) and 7·037 (Rammelsberg). By rolling, the specific gravity of zinc is increased to 7·2 or 7·3.

Melting, Boiling and Ignition Points.—Zinc melts at 410° C. according to Wurtz, at 412° according to Daniell, at 415° according to Roberts-Austen, at 419° C. according to Heycock and Neville, and at 433° according to Person. Becquerel gives its boiling point as 891° C.; Violle, 929·6°; F. Meier and Crafts, 940° C.; Roberts-Austen, 940°; Thum, 1,000°; Deville and Troost, 1,040°; Komorek, 1,050°. These differences are due probably to inaccuracies in the determinations, the older observers not having the advantages of recent pyrometry. At all events the boiling point of zinc is lower than the melting point of silver (which is given as 970° C. by Holman, Lawrence and Barr, in the *Technology Quarterly*, 1896, IX, 24 to 39), since silver does not melt in zinc vapor.[1] According to Barus[2] the boiling point of zinc is raised 1·5° C. for each centimeter of atmospheric pressure above 760 mm. The fluid density of zinc is 6·550 by Mallet's method, and 6·480 by oncosimeter. The specific gravity of zinc vapor is 2·36 (V. Meyer,

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750 mg. of ZnO by weight or by titration, the percentage of zinc in the ore according to the former constants is 60·258; according to the latter, 60·185; this is a difference of 1·46 lb. of metal in 2,000 lb. of ore of that grade.

[1] Since the above was written the boiling point of zinc has been determined accurately by D. Berthelot (vide Comptes Rendus, CXXXI, vi, 380 to 382, and Journ. Soc. Chem. Ind., Oct. 31, 1900). The molten metal was contained in a deep, narrow porcelain crucible, surrounded by a nickel wire, which when traversed by an electric current formed the source of heat, the whole apparatus being packed in magnesia and asbestos to prevent radiation. The temperature was measured by means of a thermo-couple contained in a porcelain Rose's tube, sealed at the end and inserted through the cover of the crucible, which was fitted besides with tubes by which a current of nitrogen could be passed through. The temperature was maintained 2° or 3° below the boiling point of the metal until the latter was heated uniformly; it was then raised slowly to the boiling point, at which it remained steady in spite of further increase in the electric current. Five determinations gave results varying from 913° to 927° C., thus checking closely the figure of Violle and completely discrediting the old figures of 1,000° C. and upward.

[2] Die Physikalische Behandlung und die Messung hoher Temperaturen, p. 42.

Berichte der deutschen Chemischen Gesellschaft, 1886, XIX, 3295). One liter of zinc vapor weighs 32·7 criths[1] (Frankland and Japp, *Inorganic Chemistry,* London, 1884, p. 582). Upon cooling below the boiling point zinc vapor condenses to liquid, the more easily the less the vapor is mixed with other gas; if the vapor be cooled, however, below the melting point of the metal it is condensed directly in the solid form as a fine blue powder. This property plays an important part in the practical metallurgy of zinc. In melting zinc, the increase in volume in the change from the cold, solid state to the liquid is 11·10%. Between 0° and 100° C. the specific heat of zinc is 0·09555 (Regnault); other authorities give it as 0·0927 from 0° to 100° C. and 0·1015 from 100° to 300° C.

Zinc burns in the air at a temperature as low as 500° C. (according to Daniell at 505°) with a bright, greenish-white flame and the formation of oxide, ZnO. Such fine shavings as are employed for the precipitation of gold from cyanide solutions may be ignited with a lucifer match and burn almost as freely as straw. Zinc in an extremely finely divided form, such as commercial "zinc dust," is subject to such rapid oxidation that it is liable to spontaneous combustion and even to explosion; it is classed by fire underwriters as a dangerous substance. When heated in chlorine, or the vapors of bromine or iodine, zinc burns brilliantly, with the formation of chloride, bromide or iodide respectively.

Crystallization.—Zinc crystallizes in rhombohedral forms of the hexagonal system and may be obtained in those forms by heating to the melting point, but not higher, and pouring upon a non-conducting, cold surface. Dana notes that it also appears to crystallize in the isometric system, at least in various alloys. Ordinarily, however, the metal has a coarsely laminar texture, sometimes granular. The fracture is granular or coarsely crystalline, dependent upon the temperature of casting, being coarse when the metal is heated nearly to redness before pouring and fine when the temperature of the liquid metal is not much above its melting point. Zinc heated to 160° C. emits when bent a cry like tin, but not so loud, which is supposed to be due to the sliding of the crystal faces over one another.

Ductility, Malleability and Hardness.—At ordinary temperatures zinc is brittle, especially when impure,[2] but between 100° C. and 150° C. it becomes

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[1] The crith is the weight of one liter of hydrogen at 0° C. and 760 mm. atmospheric pressure=0·0896 g.

[2] Impure commercial zinc may frequently be broken in the direction of its cleavage faces by repated blows with a hammer. According to some authorities pure zinc always yields ductile ingots. A clue to some of these anomalous facts is found in the observations of Gustav Rose and others that much depends upon the system of crystallization. "Supposing a mass of molten zinc to freeze into, say, cubes, the ingot will be ductile; an ingot of, say, rhombohedra, on the other hand is almost bound to be brittle, because the crystals are oriented in

so malleable and ductile that it may be rolled into sheets and drawn into wire, and after cooling retains those properties, which important discovery was made by Hobson and Sylvester in 1805.[1] But at 205° C., it becomes so brittle again that it may be powdered in a mortar. When cast at a temperature near the melting point it is more malleable than when cast at a higher temperature, and is also less acted upon by acids (Percy). In malleability zinc ranks between lead and iron; in ductility between copper and tin.[2] In hardness it stands between copper and tin of the common metals, but more exactly between silver and platinum, being 2·5 on Moh's scale,[3] 6 on Turner's sclerometer, and 1,077 on Bottone's scale, in which the diamond is 3,010.[4] It is difficult to file.

Thermal and Electrical Conductivity.—The thermal conductivity of zinc is variable (or has not been determined accurately) ranging from 19 (Wiedemann) to 64·1 (Calvert and Johnson), silver being 100.[5] Its electrical conductivity is 16·92 mercury at 0° C. being unity. Rating silver at 100, the electrical conductivity of zinc is 24·06 according to Becquerell, 27·39 according to Matthiessen, and 29·90 according to Weiller. The electrical resistance of a wire 1 mm. in diameter and 1 m. in length is 0·0724 ohm, increasing with temperature. According to Roberts-Austen[6] the coefficient of linear expansion of zinc is 0·0000291; according to Fizeau it is 0·002905 for 100° from 0° C. upward; according to Calvert and Johnson it is 0·002193 for hammered zinc; according to the British Board of Trade units it is 0·002532.

The tensile strength of zinc varies greatly according to the mode of preparation, ranging from 2,700 lb. per sq. in. for cast metal to 17,700 for an annealed rod. Roberts-Austen gives its ultimate tensile strength (kind of metal not stated) as 7,000 to 8,000 lb. per sq. in.[7] against 4,600 lb. for cast tin and 19,000 for cast copper. According to Wertheim, a permanent elongation of 0·5 mm. per meter, of a bar 1 mm. square, took place with tensions of 0·75, 1·00 and 3·20 kg., according as to whether the bar was drawn, annealed or cast; he found also that the coefficient of rupture of a wire 1 mm. in diameter was 1·5 kg. with cast metal and 12·50 with

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a lawless fashion, and, as they cannot be expected to contract at the same rate in all directions, we must be prepared for a brittle ingot" (Enc. Brit., 9th edition, XXIV, 786).

[1] Gilbert's Ann., XXIV, 104.

[2] The ductility of commercial zinc is increased by subjecting it to regular and high pressure.

[3] Moh's scale of hardness for metals is as follows: lead, 1; tin, 1·07; zinc, 2·5; copper, 3·0; gun metal, 3·3; bronze containing 12% Sn, 3·5; bronze containing 18% Sn, 3·7; iron wire, 3·8 to 3·9; sewing needles, 5 to 5·5.

[4] According to Hugueny the hardness of zinc is 0·83 that of copper.

[5] According to Roberts-Austen the thermal conductivity of zinc is 28·1.

[6] Introduction to the Study of Metallurgy, revised edition, p. 72.

[7] Op. cit., 27.

drawn and annealed. According to Karmasch the absolute strength of cast zinc is 197·5 kg. per sq. cm., and of sheet and wire 1,315 to 1,560 per sq. cm. Trautwine determined that a prism of cast zinc, 1 in. square and 4 in. high was compressed 1/400 of its height by 2,000 lb., 1/200 by 4,000 lb., 1/100 by 6,000 lb., 1/38 by 10,000 lb., and 1/15 by 20,000 lb., while under 40,000 lb. it yielded rapidly and broke into pieces.

CHEMICAL PROPERTIES.—In dry air zinc retains its luster, but in damp it becomes covered with a thin, grayish-white coat of a basic carbonate, which protects it from further oxidation. This property makes it valuable for outdoor uses. Pettenkoffer found that a sheet of zinc exposed as part of a roof at Munich, Bavaria, for 27 years was oxidized only to a depth of 0·01 mm. Zinc is electro-positive to all other common metals, except magnesium, and precipitates all the ductile metals from their solutions with the exception of magnesium, iron and nickel, but is itself precipitated by magnesium. It dissolves readily in nitric acid, but when pure is almost unaffected by other acids, dilute or strong. Impure commercial zinc, however, is easily soluble in dilute sulphuric acid, and in dilute or strong chlorhydric. Similarly pure zinc is unaffected by boiling water, while the latter is decomposed by ordinary metal with the evolution of hydrogen, zinc hydrate being formed, but as the metal becomes coated with an envelope of the soluble hydrate action gradually ceases, wherefore even impure zinc may be preserved indefinitely in pure water. When the zinc is in a state of extremely fine division, as in zinc gray (zinc dust), water is decomposed by it even at ordinary temperatures. Sea water attacks zinc much more rapidly than pure water. The difference in the behavior of pure and impure zinc was first pointed out by A. de la Rive in 1830, and has been discussed by many writers, especially Pullinger in the *Journal of the Chemical Society of London,* LVII, 815, and Werren in *Berichte der deutschen Chemischen Gesellschaft,* XXIV, 1785. According to the latter, pure zinc upon being put into acid is immediately enveloped by a coating of hydrogen, which protects it; upon boiling this coating is torn away and the zinc dissolves. The solubility of zinc in sulphuric acid is promoted by the addition of chromic acid or hydrogen peroxide. If a piece of platinum be brought in contact with the zinc, the latter dissolves quickly. Zinc dissolves readily in cold nitric acid, because the latter oxidizes the hydrogen evolved.

Aqueous solutions of the caustic alkalies dissolve zinc with the evolution of hydrogen, but much more slowly than acids do. The action is more energetic when the zinc is in contact with iron, forming a galvanic couple. Zinc is thus easily dissolved in potash lye when contained in an iron vessel. The zinc shavings used for the precipitation of gold in the cyanide process are dissolved quickly by the caustic alkali present in those solutions. In the cyanide process, commercial advantage is taken of the power of zinc to displace other metals, in this case gold and silver, from their solutions.

At red heat zinc is attacked by carbonic acid with the formation of carbon monoxide and zinc oxide. Sulphur also unites with it at red heat, forming zinc sulphide, but the combination is incomplete, even when both the zinc and the sulphur are finely divided and intimately mixed, because the particles of zinc are protected from further action by the infusible envelope of zinc sulphide which is first formed. By rapid heating with sulphide of mercury, as well as sulphide of calcium, zinc is completely changed to sulphide.[1] In smelting zinc with oxide of lead the latter is reduced to metal and zinc oxide is formed. In smelting with carbonates of the alkalies zinc oxide is also formed, carbonic dioxide being disengaged, but with sulphates of the alkalies a mixture of zinc oxide and zinc sulphate is formed, and sulphurous anhydride is set free.

IMPURITIES OCCURRING IN ZINC AND THEIR EFFECT.—The impurities which occur in commercial zinc, often amounting to as much as 2% of the latter, are lead, iron, cadmium, copper, carbon, silicon, arsenic, antimony, sulphur, tin, silver, thallium, indium and gallium. The most common of these are lead, iron and cadmium, while the five elements last mentioned are rare. Silver has been detected in zinc from Brixlegg in Tyrol, and tin in New Jersey metal.[2] Freiberg zinc has contained 0·0393 to 0·0524% indium. Thallium is present in many specimens of blende and calamine from Theux and Nouvelle Montagne, Belgium, and appears in the metal distilled from them. It has also been detected in the mother liquors of the zinc sulphate works near Goslar in the Lower Harz.

Lead.—Lead is found in most makes of spelter, being invariably distilled to a considerable extent along with the zinc when it occurs in the ore.[3] An excess of lead can be separated, however, since the capacity of zinc for holding it is limited, varying with the temperature; it is the greater, the higher

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[1] Schnabel, Handbuch der Metallhüttenkunde, II, 5.

[2] Kolbech has isolated crystals of cassiterite in the blende from Freiberg.

[3] The extent to which lead is distilled over with the zinc depends upon the percentage existing in the ore and the temperature and other conditions under which the operation is managed. Under certain conditions an ore not very high in lead may yield a spelter containing a good deal of lead, while under other conditions an ore comparatively high in lead may afford a really good grade of spelter. At a certain works in Europe an ore assaying 14% Pb yields a spelter with only 1% Pb, while Picard and Sulman claim to obtain a spelter containing only 0·5% Pb by distillation of ore assaying 24% Pb. On the other hand, the ore distilled in Upper Silesia, which is very much lower in lead, yields a spelter holding 2 or 3% Pb.

the temperature. According to Roessler and Edelmann, zinc will hold 1·7% Pb at its melting temperature and 5·6% at 650° C.[1] Any excess of lead beyond those proportions will sink unalloyed to the bottom of the pot. A moderate tenor in lead makes zinc malleable and ductile, and consequently is desirable in metal which is to be rolled, but with an increasing proportion it becomes tender. Consequently the percentage of lead in zinc intended for rolling must be limited, else the sheets will tear and crack under the rolls. A tenor of 1·5% Pb will permit the rolling of the zinc without cracking the sheets, but will unfit them for some purposes. The weakness and softness of the sheets increase as the percentage of lead increases. Zinc with 3% Pb may still be rolled, but it is very weak. The presence of a considerable quantity of lead unfits zinc for sheet for graphical purposes, which requires smooth, glossy surfaces such as are afforded by a good grade of zinc, while metal containing much lead gives a gray, rough surface;[2] lead is also objectionable in zinc for making the better grades of brass, but a moderate percentage does not unfit it for use in making inferior grades.

Iron.—Iron may be present in zinc to the extent of several per cent, but the tenor seldom exceeds 0·2%. Up to that figure, according to Karsten, the presence of iron does not affect importantly the properties of zinc, but the general tendency of the impurity is to make the metal less fluid, less malleable, less strong, and harder and more brittle.[3] Spelter may contain 0·125% Fe and still be a good metal for rolling to sheet, but with more than that percentage its bad effects become apparent, and between 0·20 and 0·25% become very marked and interfere very seriously with the rolling of the metal. Commonly the percentage of iron in commercial zinc is between 0·01 and 0·05%, but according to Jensch (in *Zts. f. angew. Chem.*, 1890, p. 13) spelter produced from the dust collected from Upper Silesian iron furnaces, containing 17 to 22% Zn and 23 to 25·5% FeO, has a tenor of 0·71% Fe. That zinc can hold a still greater percentage of iron is shown by the hard product obtained in the refining of crude zinc, which segregates

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[1] In ordinary commercial refining by gravity separation of the lead from crude molten spelter, the lead content of the latter is reduced to about 1%. According to Spring and Romanoff zinc will hold 25·5% Pb at 900° C., 7% at 650°, and will still retain 1·5% at 419°.

[2] To ascertain whether zinc is suitable for graphical purposes, Augurer recommends brightening a small spot with a finely ground scraper and examining the fresh surface with a microscope. The zinc appears as ashy, star-like scales, while the lead shows as bluish flecks. The microscopical appearances of pure zinc and zinc contaminated with lead are described by H. Behrens in Das mikroscopische Gefüge der Metalle und Legirungen, published by L. Voss, Hamburg and Leipsic, 1894, p. 56.

[3] The American brass trade has become very particular as to the iron content of spelter and now imposes a limit of about 0·05% Fe as the maximum permissible in Western metal to be used for brass manufacture.

between the lead and zinc, while in galvanizing iron an alloy of zinc with 4% Fe separates in the kettles. Herapath found 7% Fe in zinc taken directly from the condensers of a distilling furnace.

Cadmium.—Cadmium seldom occurs in zinc except in insignificant quantities, which have no injurious influence; if the ore contains a comparatively high percentage of cadmium, the proportion in the spelter will still be low, since cadmium is a more volatile metal than zinc and for the more part fails to be condensed. If present in larger quantity than ordinarily the tendency is to make the spelter more brittle, giving it a fine-grained fracture, but it is said that zinc with 15% Cd may still be rolled. If the spelter is to be used for the manufacture of zinc white, the presence of cadmium is objectionable, since either cadmium oxide or sulphide may give the product a yellowish tinge.

Copper and Tin.—Copper makes zinc harder and more brittle, even if only 0·5% be present, so that in rolling it cracks at the edge and the sheets cannot be folded without breaking. Tin also makes it harder and more brittle. According to Karsten, zinc containing 1% Sn is rendered so brittle, at a temperature at which it would otherwise be pliant, that when rolled it cracks at the edges of the sheets. Both copper and tin are of rare occurrence in spelter. Missouri brands have shown 0·0013 to 0·1123% Cu; Lehigh Zinc Co., Penn., 0·530% Cu (an old analysis); Reckehütte and Georgshütte, Upper Silesia, 0·0002% Cu; Freiberg, Saxony, 0·07% Sn; while tin has also been noted in New Jersey and Welsh spelter. Zinc remelted from roofing scrap may contain more tin, due to the solder.[1]

Rare Metals.—Of the uncommon metallic impurities the following maxima have been noted: Silver, 0·0017% in spelter from Upper Silesia; thallium, 1·40% in a specimen smelted from scrap;[2] indium, 0·0524% in spelter from Freiberg, Saxony; magnesium, 0·46%; aluminum, 0·17%; antimony, 0·0249% in zinc from Missouri. Traces of manganese and bismuth have been reported.

Arsenic and Antimony.—Arsenic if present in large quantity makes the spelter brittle and difficult to melt. Arsenic combines with zinc in various proportions, forming alloys, at moderate temperatures. The presence of as much as 0·0603% has been noted in Missouri zinc. Any arsenic at all unfits zinc for the development of hydrogen where arseniureted gas would be objectionable. Zinc which is to be used for precipitating gold from cyanide solution should be free from arsenic, because of the danger in treating the

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[1] Fohr. Berg- u. Hüttenm. Ztg., 1888, p. 28.

[2] This was a very remarkable crude zinc reported by Kosmann in Chem. Ztg., 1886, 1·40% thallium, 7·19% As, 2% Fe and some ZnO. No. 50. It contained 7·15% Pb, 0·99% Cd,

precipitate with acid, fatalities having occurred in the United States from arseniureted hydrogen developed from impure zinc in that manner. The effect of antimony on zinc is said to be similar to that of arsenic, but probably has not been much studied.

Sulphur and Carbon.—Sulphur occurs in numerous spelters up to 0·0741%. Its effect has not been studied, but so far as known it does not seem to be deleterious.[1] According to Funk, the sulphur contained in spelter is not chemically combined, but exists as ZnS carried over from the charge in distillation.[2] Funk states also that the carbon, which has been found in certain zinc to the amount of 0·1775%, is mechanically carried over, and not chemically combined; and considers the odor of the gases generated by action of acids on zinc to be due principally to hydrogen sulphide and not to hydrocarbons. Rodwell showed that the black flocks which remained after dissolving commercial zinc in acid consisted of carbon, lead sulphate and a trace of iron.[3] The influence of carbon on zinc has not been studied, but as in the case of sulphur, so far as known it is neither harmful nor advantageous.

Chlorine to the amount of 0·2 to 0·3% was found by Kunzel in spelter made from the crusts from a Belgian works.[4] Although this metal contained lead and iron only in traces and to the eye was of good quality, it could not be rolled. Jensch has also called attention to the presence of chlorine in zinc. Silicon has been found in Missouri zinc up to 0·1374%.

Oxygen occurs frequently in zinc in the form of zinc oxide, which may commingle with the metal collected in the condensers or in the subsequent handling. When present in considerable quantity it produces a pasty metal (burned zinc) which gives castings without sharp edges, brittle, and difficult to work with chisel and file. Such burned zinc is apt to be produced by any imperfection in the distillation process which causes the zinc vapor to be oxidized, such as a deficiency of carbon in the charge and carrying the distillation too far, whereby the atmosphere of carbonic monoxide in the retort is expelled by furnace gases penetrating the walls and the remaining zinc vapor is oxidized, the oxide going over into the condenser.[5]

ZINC ALLOYS.—Zinc forms alloys with most of the common metals at temperatures sufficiently high to insure fusion. These alloys are usually white,

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[1] It is considered by some, however, that even a small amount of sulphur may seriously affect the ductility of the spelter, inasmuch as the sulphur unites with iron when the hot zinc comes in contact with that metal and thereby the zinc is rendered cold short. By coating the molds with beeswax and chalk the reaction between the sulphur and iron may be largely prevented. Zinc will be contaminated with iron from the tools and molds even when it is quite free from sulphur.

[2] Zts. f. anorgan. Chem., 1896, p. 49.

[3] Chemical News, January, 1861, No. 57.

[4] Berg-u. Hüttenm. Ztg., 1874, p. 6.

[5] Kerl, The Mineral Industry, V, 613.

crystalline, brittle compounds, and are of little importance with the exception of the copper-zinc series and the newly investigated aluminum-zinc series.

Aluminum and Zinc.—An alloy of zinc and aluminum which possesses interesting and valuable properties has been described by Professor Durand, of Cornell University. It consists of two-thirds aluminum and one-third zinc. This alloy is said to be the equal of good cast iron in strength and superior to it in elastic limit. Its color is white. It takes a fine, smooth finish and does not readily oxidize. It melts at a dull red heat or slightly below, probably at about 800° to 900° F., and is very fluid, running freely to the extremities of the mold and filling perfectly small or thin parts; in that respect it is said to be superior to brass, but it is brittle and hence unsuited to pieces which require the toughness possessed by brass. The tensile strength of the alloy was found to be approximately 22,000 lb. per sq. in., and its specific gravity 3·3.[1]

Doctor Joseph W. Richards confirms the general correctness of Professor Durand's statements and adds that experience in making the alloy has led to considerable improvements in the results obtained.[2] There is now no difficulty in producing castings of the alloy showing a tensile strength of 40,000 lb. This alloy resembles closely in its characteristics a high-carbon steel, being extremely rigid, slightly elastic and breaking short with a fine-grained fracture. It works well under tools, in turning or boring, not requiring lubrication. It is the hardest and strongest of the available alloys of zinc and aluminum, takes a high polish and retains its color well, but it is not so resistant to shock as are the other alloys containing less zinc. Doctor Richards reports the specific gravity of this alloy as 3·8 and states that a contraction of 17% takes place during the alloying of its ingredients, which observation suggests the cause of its great strength.

Although there will be numerous uses for the strong, rigid alloy above described, Doctor Richards considers that the alloy consisting of three parts of aluminum and one part of zinc will be the most valuable of the zinc-aluminum series. The latter alloy, containing 25% Zn, is softer than the other one, which contains 33⅓% Zn. Its elastic limit is about the same, with a slight elongation before breaking; its tensile strength is 35,000 lb. per sq. in. It is not malleable, but on the other hand it is not brittle inasmuch as it bends slightly before breaking. The latter property is a valuable one, since it enables a casting to be straightened to a certain extent by hammering. Remarkably clean and sharp castings can be made, when

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[1] The Mineral Industry, VI, 29.

[2] Eng. & Min. Journ., Nov. 30, 1901, p. 715.

experience has been attained as to the proper gating of the mold and the exact temperature of casting. In the case of both of these alloys overheating in the crucible must be avoided; also the use of iron stirring implements, because oxide and dross do not separate easily out of the metals and may be poured into the mold causing injury to the casting. The specific gravity of the alloy containing 25% Zn is 3·4. A contraction of 14% takes place during the alloying.

The alloy of 75% Al and 25% Zn, when properly made from pure metals, is equal to the best brass in the lathe, under the drill, and in not clogging the file. It casts soundly, takes a high polish and has as fine a color as the best aluminum. It is not so hard and short as the alloy containing 33⅓% Zn, nor is it quite so strong, but it has supplanted the latter for most purposes because of its better working qualities and greater reliability under shock. It is now used for the manufacture of scale beams, surveying and astronomical instruments, and light machine parts. Its use is increasing rapidly.

Below 25% Zn, the strength and hardness of the zinc-aluminum alloys decrease rapidly. The alloy with 15% Zn has in castings an elastic limit of 16,000 lb. per sq. in., a tensile strength of 22,330 lb. and an elongation of 6% in two inches. It can be rolled and drawn into wire if frequently annealed. All of the alloys of zinc and aluminum with less than 15% Zn can be forged, rolled or drawn. They gradually become softer and weaker and require lubrication of the tools during working.

The alloys of zinc and aluminum which are high in zinc appear to be destitute of specially valuable mechanical properties. Even the alloy consisting of 50% Al and 50% Zn, which has a specific gravity of 4, falls under that category.[1]

Antimony and Zinc.—With antimony zinc unites readily in all proportions, forming alloys which are brittle and fusible, and exhibit a close-grained, dark-gray fracture when much antimony is present. An alloy consisting of equal parts antimony and zinc is of a bright, sky-blue color, and has the peculiar property of writing upon glass. Even a small proportion of antimony renders zinc bluish, and the presence of more than 0·5% may be detected in that manner. Zinc-antimony alloys have found a limited application in thermopiles.

Bismuth and Zinc.—Zinc unites with bismuth when both are melted, but upon cooling two distinct layers are formed, the upper containing 2·4% Bi

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[1] The above notes as to the properties of the zinc-aluminum alloys are abstracted from the paper by Doctor Richards previously referred to.

and 97·6% Zn, and the lower 85·7 to 91·4% Bi and 14·3 to 8·6% Zn. In this respect bismuth behaves with zinc very much as lead does.

Copper and Zinc.—Zinc and copper unite in all proportions, forming alloys, which under the name of brass, are of widespread industrial application. These compounds vary much in their properties according to their composition. Those containing upward of 80% Cu are reddish-yellow to red; with less than 48% Cu they are gray to white; in the intermediate stages they are shades of yellow. The hardness of these alloys is usually greater and the melting point lower than those calculated as the mean of their constituents. The alloys with more than 62% Cu can only be rolled cold; those with about 60% Cu can be rolled hot or cold, but are less ductile and tough than the higher grade; those with about 50% Cu cannot be rolled, either hot or cold; with less than 50% Cu they lose their yellow color and become brittle, while their fusibility increases. The chief application of the alloy containing about 50% Cu is for solder for copper and brass.

Since the ductility of the copper-zinc alloys increases with the percentage of copper, the best kinds, such as red brass, gilding metal, and percussion cap metal are made to contain 80 to 96% Cu. Dutch metal contains 80 to 85% Cu; Prince's metal, 75% Cu; best English sheet brass, 70 to 72% Cu; ordinary, 66·67% Cu; common, 63% Cu, and yellow metal, 60% Cu. The last, though somewhat deficient in ductility and toughness, possesses the great advantage that it can be rolled either hot or cold.

Brass is made by melting together copper and zinc in the proper proportion. Laurie has shown[1] that there is only one definite alloy of copper and zinc, it corresponding approximately to the formula $CuZn_2$, containing about 33% Cu and 67% Zn, and that all other copper-zinc alloys may be considered as solidified solutions of this definite alloy in an excess of one of its constituents. The definite alloy itself is hard and brittle, and of little or no practical value.

Gold and Zinc.—With gold, zinc forms a series of alloys which are pale yellow when the gold is in excess, and become greenish as the proportion of zinc increases. For jewelry part of the copper is often replaced by brass, the zinc in the latter giving the gold a more desirable color, but the presence of much zinc makes the alloy hard and difficult to work.

Iron and Zinc.—With iron zinc forms a series of alloys which are generally hard, white and brittle. A fine-grained alloy containing about 95% Zn and 5% Fe, which is darker in color than pure zinc, is produced in the process of galvanizing. This is known as "hardhead" and "hard zinc." Alloys richer in iron are obtained by liquating hard zinc and fusing

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[1] Journal of the Chemical Society of London, LIII, 106, and LV, 677.

the friable residue; alloys with as much as 30% Fe being produced in that manner. They are also prepared by immersing clean iron wire or sheet in zinc heated to near its boiling point. These alloys find industrial application in the manufacture of "delta metal" and similar compounds, which are brasses with the addition of from 0·25 to 4% Fe. The iron imparts hardness, toughness and tenacity, and the alloy can be rolled hot or cold.

Lead and Zinc.—With lead, zinc behaves in much the same manner as with bismuth, uniting readily when both are molten, but separating into two layers upon cooling. The upper consists chiefly of zinc with about 1·5% Pb, and the lower is lead with 1·2% Zn. Ordinary commercial zinc is an alloy containing about 1% Pb, which enables the zinc to receive a good polish, and increases its malleability. The presence of that amount of lead, therefore, is desirable in zinc for rolling into sheets, but an excess of lead is undesirable. Lead is also undesirable in zinc which is to be used for the manufacture of brass, especially the superior grades.

Mercury and Zinc.—With mercury zinc readily unites, forming a series of white brittle amalgams which become pasty when mercury is in excess. Zinc amalgams are easily obtained by adding zinc to mercury heated nearly to its boiling point. Zinc plates for galvanic batteries are coated with mercury by first cleaning the surface of the zinc with dilute sulphuric acid and then rubbing the mercury over the clean surface. An amalgam of zinc and tin is used on the rubbers of frictional electric machines.[1]

Silver and Zinc.—Zinc unites with silver at high temperatures, forming a series of white alloys. Those with 5, 10 or 20% Zn are ductile and can be rolled like silver. They are more fusible than the corresponding copper-silver alloys. At low temperatures zinc and silver do not unite readily.

Tin and Zinc.—With tin, zinc readily unites in all proportions, the resulting alloys being generally harder than tin but softer than zinc. Their color is in all cases white, and their fracture crystalline, though their appearance varies with the mode of preparation. The alloys of tin and zinc are not uniform in composition, since the tin tends to separate and collect at the bottom on cooling. Tin-zinc alloys are used to a slight extent for castings for ornamental purposes. A detailed description of these alloys, based on the work of Guettier and Rudberg, is given in Hiorns' *Mixed Metals,* pp. 249 to 253.

An addition of tin to a mixture of zinc and lead causes those metals to alloy to an extent to which they will not by themselves. If tin, lead and zinc are melted together and left at rest in a fused condition no separation

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[1] Thorpe, Dictionary of Applied Chemistry; as to the literature of zinc amalgams vide Journ. Soc. Chem. Ind., IX, p. 512.

takes place if the proportion of tin exceeds a certain amount; but if the quantity of tin be less than that, the mixture separates into two layers, each layer consisting of a ternary alloy of the three metals.

Other Binary Alloys.—Zinc alloys with magnesium, nickel, cobalt, tellurium and sodium. According to Vautin[1] the zinc used for precipitating gold from cyanide solutions may be advantageously alloyed with 1% of sodium.

Complex Alloys.—Besides the binary alloys, zinc enters into the composition of a considerable number of alloys containing three or more metals, which are more or less useful in the arts. The following are the most important of these:

	Cu.	Zn.	Sn.	Pb.	Ni.	Sb.	Fe.
Brass *a*	63·00	34·00					
	72·00	27·00					
" *b*	70·29	29·26	0·17	0·28			
Muntz's Metal *c*	60·00	40·00					
	62·00	38·00					
Sterro's Metal *d*	55·33	41·80					4·66
Aich's Metal *e*	60·00	38·12					1·50
Mosaic Gold	65·00	35·00					
Pinchbeck	83·33	16·76					
Mannheim Gold	80·00	20·00					
	88·00	12·00					
Bronze *f*	95·00	1·00	4·00				
" *g*	82·70	1·80		4·70			
	71·40	6·00		5·90			
" *h*	74·00	10·00	1·00	15·00			
" *i*	80·00	7·00	13·00				
	90·00	7·00	3·00				
Delta Metal	55·10	43·47		0·37			1·08
Silicon Bronze *m*	97·12	1·12	1·14				
Packfong *n*	43·80	40·60			15·60		
English German Silver	61·30	19·10			19·10		
Sheffield " "	57·00	19·00			24·00		
Berlin Argentan	52·00	22·00			26 00		
Ashberry Metal	.....	2·80	77·80			19·40	
Anti-Friction Metal	5·00	85·00				10·00	
Babbitt's Metal	4·00	69·00	19·00	5·00		3·00	
Tombac (English)	86·38	13·61					
" (Viennese)	97·80	2·20					
Watchmakers' Alloy *o*	58·86	40·22		1·90			

a Typical brass. *b* Wire; always brittle if lead reaches 2%; tin may vary from 0·1 to 0·5%. *c* Used for sheathing ships. *d* An Austrian metal used for ordnance. *e* This and Sterro's metal are remarkable for their great strength, viz., 85,080 lb. per sq. in.; another analysis of Aich's metal is Cu 60·20%, Zn 38·10%, Fe 1·60%, total 99·90%. *f* British bronze coinage. *g* Japanese art bronze. *h* Chinese art bronze. *i* Used for bearings for heavy axles. *m* Used for telephone wire. *n* A Chinese alloy. *o* A gold-like alloy used by watchmakers.

Part of the tin in gun metal and in bearing metal is frequently replaced by zinc, the density and wearing properties of those alloys being thereby increased. For the same reason the bronze coinage of Great Britain contains

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[1] Journ. Soc. Chem. Ind., 1891, p. 96.

1% Zn. Zinc is also present in some varities of aluminum bronze, manganese bronze and other similar alloys. The "Biddery ware," manufactured in India, usually contains about 90% Zn with copper, lead and tin in different proportions.

Zinc, copper and nickel form a nickel bronze of great strength, and alloys of that nature have been proposed for making high-pressure steam fittings. They are little subject to corrosion and in some respects are considered preferable to steel castings. According to Sergius Kern (*Chemical News*, 1900) the alloy composed of 70% Cu, 12·5% Zn and 17·5% Ni has a tensile strength of 26 tons per sq. in., and an elongation of 23% in two inches, while the alloy with 70% Cu, 10% Zn and 20% Ni has a strength of 36 tons and elongation of 14 to 17%.

Wilder's metal coating is an alloy consisting of 84% Zn, 14% Sn, 1·5% Pb and 0·5% Al, which is intended to take the place of ordinary spelter in galvanizing. The tin reduces the melting point of the alloy below that of pure spelter. The function of the lead is to give more fluidity to the alloy. The aluminum is claimed to improve the appearance of the coating.

A manganese bronze composed of 53% Cu, 42% Zn, 3·75% Mn and 1·25% Al, is said to make a very strong and tough alloy, suitable for propeller wheels, gears, etc., and for mining screens, being not attacked by acid mine waters.

The property of zinc to form alloys with gold and silver has already been referred to. Mention has been made, also, in a previous chapter, of its power to rob molten lead of those metals when alloyed therewith, and the application of that property in the desilverization of lead bullion. At the same time the zinc completes the refining of the lead by alloying with, and thus removing, the last traces of copper, tellurium and other impurities.

Zinc can be separated from alloys of which it is a constituent by heating to a temperature above its boiling point. This process is applied technically on a large scale in the refining of desilverized lead. After the gold and silver have been removed from the softened lead by the addition of zinc and skimming off the crusts, the lead remains saturated with zinc, the percentage varying according to the temperature but ranging usually from 0·6 to 1%. In order to produce commercial lead the product of the desilverizing kettles is run into a reverberatory furnace wherein the zinc is burned off.

VII

CHEMISTRY OF THE COMPOUNDS OF ZINC.

The chemical combinations of zinc which play an important part in the ordinary method of recovering the metal are especially the sulphide, the sulphates (neutral and basic), the oxide, the carbonates and the silicates. In the methods which have come into use, however, for the treatment of zinky mixed ores and the numerous processes which have been proposed for that purpose, many of the other combinations of zinc are produced, and in view of the great interest in the development of such processes it is useful to summarize rather fully the properties of the chemical compounds of the metal, especially the more important of them.

Sulphide.—Zinc sulphide (ZnS) occurs in nature as the mineral blende. It may be produced artificially by (1) heating zinc oxide with sulphur; (2) heating zinc oxide in a stream of hydrogen sulphide; (3) heating zinc shavings with mercury sulphide; (4) heating finely divided zinc with alkaline polysulphides; (5) heating zinc sulphate and coal to white heat; method, namely by precipitation, zinc sulphide is a fine, white, amorphous sulphide, sodium sulphide or potassium sulphide. Prepared by the last method, namely by precipitation, zinc sulphide is a fine, white amorphous powder, in which form it is useful as a pigment; the reaction finds commercial application in the manufacture of lithophone. Lithophone is a double precipitate of zinc sulphide and barium sulphate. The sulphides of the alkaline earths, like the sulphides of the alkalies, throw down zinc sulphide, but in the case of solutions of zinc sulphate, the sulphates of the earths, which are formed in the reaction, being themselves insoluble, go down together with the zinc. Zinc sulphide occurs in nature as a white mineral (vide Chapter VIII) and white may be considered the true color of the compound.

Zinc sulphide is infusible and at moderate temperatures non-volatile, but according to Percy it volatilizes at high temperatures. Heated to redness in the air it burns with the evolution of sulphurous anhydride and the formation of zinc oxide, neutral zinc sulphate ($ZnSO_4$), tetrabasic sulphate

($3ZnO,ZnSO_4$) and perhaps other basic sulphates according to the conditions under which the oxidation takes place. If the sulphates be formed they may be decomposed with evolution of sulphurous anhydride, sulphuric anhydride and oxygen by further raising the temperature, so that zinc oxide alone may be formed from the sulphide by prolonged roasting at the proper temperature. Zinc sulphide may be also converted into zinc oxide by heating in an atmosphere of steam, whereby hydrogen sulphide will be given off. Heating in a mixed atmosphere of air and steam, hydrogen sulphide will still be produced, but there appears to be an increased tendency to the formation of zinc sulphates. According to Schnabel[1] the desulphurization of zinc sulphide by means of steam is incomplete and requires a raise of temperature to white heat.

The behavior of zinc sulphide in the blast smelting furnace does not concern the ordinary method of zinc smelting at all and is of comparatively little importance in the treatment of mixed sulphide ores. In smelting with other sulphides, zinc to a certain extent enters the matte, making it more infusible and decreasing its specific gravity, which are both undesirable effects. Indeed, zinc is in every respect an objectionable element in blast furnace smelting; and zinc as sulphide is worse than zinc as oxide. Zinc sulphide is sometimes found in slags from the lead blast furnace, in which it is probably held mechanically. For further particulars as to this subject reference should be made to Professor H. O. Hofman's excellent treatise on the *Metallurgy of Lead,* and to papers by Doctor M. W. Iles in recent numbers of the *School of Mines Quarterly.*

Percy reported that in heating zinc sulphide with carbon, or in carbon-lined crucibles, it was completely volatilized; except if it were ferruginous a residue of iron sulphide, free from zinc, remained. It was not stated in what form the zinc was volatilized, whether as ZnS or as Zn, and the phenomenon requires further investigation to determine the conditions, chemical and pyrometric, under which it takes place. According to Berthier,[2] zinc sulphide heated with carbon and lime yields metallic zinc and calcium sulphide, but the reaction is incomplete and dependent upon the temperature. In heating 6·32 g. of $CaCO_3$ with 6·03 g. of ZnS to a high temperature more than five-sixths of the zinc was volatilized, and the residue weighing 4·6 g. contained only a little ZnS. Percy heated 35 g. of blende with 35 g. of lime to white glow in a lime crucible (set inside a graphite crucible) and obtained a brown, porous, partially fused mass weighing 27 g., which contained some calcium polysulphide, soluble in hot water, while the remainder was soluble in HCl, hydrogen sulphide being given off and

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[1] Handbuch der Metallhüttenkunde, II, 10.

[2] Tr. de Essais, II, 570.

zinc being found in the solution. Iron also decomposes zinc sulphide at bright red heat, forming zinc vapor and ferrous sulphide, which reaction is possibly of considerable importance in the commercial distillation of zinc ore. The behavior of zinc sulphide with carbon, lime and iron at high temperatures is of especial interest, inasmuch as experience has demonstrated the need of eliminating the sulphur from the ore as completely as possible, involving an unusually careful and laborious roasting, because whatever zinc remains combined with sulphur is supposed to be lost in the process of distillation, being wasted in the residues withdrawn from the retorts. Carbon is necessarily always present in the charge for distillation; lime occurs frequently; and metallic iron is often formed in the process. The fact that sulphur does actually hold back zinc in the charge leads to the inference that although iron and lime may decompose zinc sulphide under certain conditions, they do not do so, or at the most their effect is insignificant, under the conditions which prevail in the retort during the distillation of a charge. Opinions differ, however, as to this; in reality the nature of the complete chain of reactions which take place in the retort is only imperfectly understood.

Zinc sulphide and zinc oxide also react under some conditions. According to Berthier, they unite in various proportions under the influence of heat, forming fusible oxysulphurets. Percy thought on the other hand that zinc oxide was reduced by zinc sulphide at high temperature, forming zinc and sulphurous anhydride, by a reaction analogous to that which takes place between cuprous oxide and sulphide. Percy's experiments were inconclusive, however, and according to Schnabel (op. cit.) attempts to effect the reduction of zinc sulphide by zinc oxide and carbon have failed on account of the incomplete reaction between the oxide and sulphide. The subject was brought up not long ago by a patent granted to Christopher James, of Swansea, for a process based upon this reaction. If the reaction occurs at all it must be at a temperature higher than that which prevails in roasting zinc blende in practice, since otherwise a larger amount of zinc would be lost by volatilization than is actually the case. In roasting blende in practice a temperature of 900° C. and upward is attained, but with proper furnaces and well conducted operations the loss of zinc is insignificant.

Zinc sulphide and lead oxide when mixed in the proper proportions and heated, according to Berthier,[1] react with the formation of zinc oxide, lead and sulphurous anhydride. In heating a mixture of 24·08 g. of blende with 55·78 g. of litharge there were obtained 29·2 g. of impure lead, which contained 1·8% S and 0·8% Zn. Over the lead there was a mass of zinc and

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[1] Ibid., I, 403.

lead sulphides and oxides. In order that the zinc oxide produced should be dissolved, an excess of 25% its weight in litharge was requisite, under which condition a resinous brown, glassy slag was obtained. Metallic lead has little effect on zinc sulphide, if any at all.

Antimony does not decompose zinc sulphide. Tin decomposes it partially at bright red heat. Copper decomposes it at white heat with the formation of copper sulphide. Copper protoxide also decomposes it, but incompletely, forming a regulus of the appearance of copper and copper sulphide. These investigations are due to Percy.

Carbonic acid has no effect on zinc sulphide, not even at red heat, but at that temperature the carbonates of the alkalies react with it, forming zinc oxide, zinc sulphate, and alkaline sulphides. Zinc sulphide heated with potassium nitrate yields zinc oxide and potassium sulphate; possibly a similar reaction occurs between zinc sulphide and sodium nitrate.

According to Berthier,[1] hydrogen has no action on zinc sulphide, but Morse states[2] that when the latter is heated in a current of hydrogen it appears to sublime, which is explained by the theory that the sulphide of zinc is reduced by an excess of hydrogen with the formation of hydrogen sulphide, and at a lower temperature the volatilized zinc recombines with the sulphur of the H_2S.

Zinc sulphide is insoluble in water, but it is decomposed and dissolved by dilute mineral acids. Nitric acid is the best solvent. When heated with concentrated sulphuric acid, zinc sulphate is formed together with sulphurous anhydride and free sulphur. Chlorine vapor or water attacks zinc sulphide feebly, but at moderately high temperatures chlorine gas acts upon it more energetically with the formation of zinc chloride and sulphur monochloride, according to the reaction

$$2ZnS+6Cl=2ZnCl_2+S_2Cl_2.$$

Above 600° C. the decomposition becomes more active and according to E. A. Ashcroft[3] it is then represented by the equation

$$ZnS+2Cl=ZnCl_2+S.$$

At the temperature of 600° C. and over the monochloride of sulphur probably could not exist.

Zinc sulphide is decomposed by an aqueous solution of $PbCl_2$ with the

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[1] Annales des Mines, XI, 46.

[2] Chem. Ztg., 1889, p. 179.

[3] In a paper on Sulphide Ore Treatment by the Phoenix Process, read before the Institution of Mining and Metallurgy, London, June 19, 1901.

formation of zinc chloride and lead sulphide. On the other hand lead sulphide is decomposed by molten zinc chloride.[1]

SULPHITES.—With sulphurous acid zinc forms two sulphites, namely an acid and a neutral salt, known respectively as the bisulphite and the monosulphite. There are also an hyposulphite and a thiosulphite of zinc, which are unstable compounds formed with hyposulphurous acid (H_2SO_2) and thiosulphuric acid ($H_2S_2O_3$).

The neutral sulphite of zinc ($ZnSO_3$) is formed by the action of H_2SO_3 on zinc or zinc oxide, thus:

$$H_2SO_3+Zn=2H+ZnSO_3,$$
$$H_2SO_3+ZnO=H_2O+ZnSO_3.$$

In the case of the former reaction, the hydrogen which is set free reacts with sulphurous acid, thus:

$$H_2SO_3+2H=H_2SO_2+H_2O.$$

The hyposulphurous acid combines with zinc as hyposulphite, which is altered first to thiosulphite and then to sulphite.

Zinc oxide dissolved in H_2SO_3 furnishes small crystals, but little soluble in water and insoluble in alcohol, which correspond to the symbol $ZnSO_3+2H_2O$, or to $2ZnSO_3+5H_2O$. This compound is precipitated from its aqueous solution by alcohol, but it dissolves readily in an excess of H_2SO_3. Exposed to the air zinc sulphite changes to sulphate. At 200° C. zinc sulphite loses its sulphurous anhydride, zinc oxide remaining. At a higher temperature the desulphurization takes place rapidly and completely; the escape of the gas causes the precipitate to become very porous and light, swelling enormously in bulk.

Zinc sulphite is barely soluble in water, but in the presence of sulphurous acid it forms a soluble acid salt, which is represented probably by the symbol, $H_2ZnS_2O_6=ZnSO_3,H_2SO_3$, which may be considered a solution of the monosulphite in an excess of sulphurous acid. Whichever be the correct view, sulphurous acid can be driven off by boiling the solution and insoluble zinc monosulphite precipitated thereby. Zinc hydroxide is precipitated from a solution of zinc sulphite by means of milk of lime, calcium bisulphite being formed and going into solution.

The properties of zinc sulphite are of importance in several processes proposed for the treatment of mixed sulphide ores, and for the neutralization of sulphurous fumes.

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[1] S. Ganelin, United States patent No. 593,415, Nov. 9, 1897.

Sulphates.—Zinc forms numerous sulphates, in general by the action of sulphuric acid on the metal, its oxide and carbonates, or by roasting the sulphide, the nature of the salt obtained varying with the conditions of its production.

The simple or neutral sulphate of zinc is represented by the symbol $ZnSO_4 = ZnO,SO_3$. When obtained by crystallization from a solution it is combined with varying proportions of water according to the temperature at which it is crystallized. The ordinary zinc sulphate of commerce is represented by the symbol $ZnSO_4 + 7H_2O$.

$ZnSO_4 + 7H_2O$ is obtained by crystallization from solutions below 30° C., forming as orthorhombic prisms, isomorphous with magnesium sulphate, and of 2·036 sp. gr. It is insoluble in absolute alcohol, but is extremely soluble in water. The aqueous solution has an acid reaction and a styptic, metallic taste. According to Poggiale[1] the solubility of $ZnSO_4$ and $ZnSO_4,7H_2O$ in 100 parts of water is as follows:

Temperature	$ZnSO_4,7H_2O$	$ZnSO_4$	Temperature	$ZnSO_4,7H_2O$	$ZnSO_4$
0° C.	115·22	43·02	60	313·48	74·20
10	138·21	48·36	70	369·36	79·25
20	161·49	53·13	80	442·62	84·60
30	190·90	58·40	90	532·02	89·78
40	224·05	63·52	100	653·59	95·03
50	263·84	68·75			

Gerlach[2] and Schiff[3] give the following data as to specific gravity in connection with which I have interpolated the columns of percentage of $ZnSO_4$ and the corresponding approximate readings on the Beaumé and Twaddell scales:

$ZnSO_4,7H_2O$ Per cent.	$ZnSO_4$ Per cent.	Gerlach's determinatfons at 15° C.				Schiff's determinations of sp. gr. at 20·5% C.
		Sp. gr.	Degrees Beaumé.		Degrees Twaddell.	
			United States	Europe.		
5	2·8	1·0288	4·1	4·0	5·8	1·0289
10	5·6	1·0593	8·2	8·0	11·8	1·0588
15	8·4	1·0905	12·1	12·0	18·2	1·0899
20	11·2	1·1236	16·0	15·9	24·8	1·1222
25	14·0	1·1574	19·8	19·6	31·5	1·1560
30	16·8	1·1933	23·4	23·2	39·5	1·1914
35	19·6	1·2315	27·2	27·0	46·2	1·2285
40	22·5	1·2709	30·9	30·7	54·2	1·2674
45	25·3	1·3100	34·3	34·2	62·0	1·3083
50	28·1	1·3532	37·8	37·6	70·6	1·3511
55	30·9	1·3986	41·2	41·0	79·4	1·3964
60	33·7	1·4451	44·7	44·4	89·0	1·4439

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[1] Ann. de Chimie et Physique (3), VIII, 467. [2] Fresenius' Zts. f. analyt. Chem., VIII, 260.
[3] Liebig's Annalen der Chemie, CX, 72.

The aqueous solution of zinc sulphate is decomposed by the electric current, metallic zinc being deposited at the cathode while the acid radical is set free at the anode.

Zinc sulphate crystallized with seven molecules of water effloresces on exposure to the atmosphere; it loses six molecules of its water of crystallization at 100° C., and the remaining molecule at 200°. It is, however, difficult to dehydrate the salt completely without driving off sulphuric anhydride.

$ZnSO_4+6H_2O$ is produced by crystallization at 30° C., forming in clinorhombic crystals. It is also produced by driving off one molecule of water from the compound previously described.

$ZnSO_4+5H_2O$ is obtained by crystallization between 40° and 50° C.; also by heating the septihydrated salt with alcohol of 0·856 sp. gr.

$ZnSO_4+4H_2O$ is produced together with the septihydrated salt when an acid and concentrated solution is made to crystallize at 0°, forming as opaque rhombohedrons, which are unaltered by exposure to air.

$ZnSO_4+2H_2O$ is deposited by the addition of concentrated H_2SO_4 to a boiling solution of $ZnSO_4$; it is a crystalline powder. It is also produced by boiling the septihydrated salt with absolute alcohol.

$ZnSO_4+H_2O$ is produced by heating the septihydrated salt to 100° C. It is also deposited in crystalline grains from a saturated solution at 100° C. According to Graham this salt does not lose its water until 238° C.

$ZnSO_4$, the anhydrous salt, is a white, brittle substance of 3·4 sp. gr. It absorbs moisture from the air, transforming itself into $ZnSO_4+7H_2O$, and in combining with water disengages heat. It is formed by roasting zinc sulphide at a low temperature. At a higher temperature it is decomposed with the formation of a basic sulphate, oxide, sulphuric anhydride, sulphurous anhydride and oxygen. Calcined in a current of hydrogen it leaves a residue of zinc oxysulphuret.

The decomposition of $ZnSO_4$ by a simple calcination is a tedious operation, requiring a high temperature, and it has been proposed to facilitate it by calcining with a mixture of carbon, when under certain conditions of temperature the reaction $ZnSO_4+C=ZnO+SO_2+CO$ takes place. Such a calcination is of course limited practically below the temperature at which ZnO is reduced by carbon. In fact, this reaction takes place only at dull red heat.[1] If zinc sulphate mixed with carbon be raised quickly to white heat the sulphate is reduced to sulphide. Zinc sulphate is also decomposed by roasting in a closed vessel with the equivalent quantity of zinc

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[1] P. Mahler, Annales des Mines, 1885, VII, 512.

sulphide, according to the reaction $ZnS+3ZnSO_4=4ZnO+4SO_2$. This reaction was employed on a large scale in the Parnell process.

The following experiments made by me several years ago are not without interest, although the temperatures were not determined accurately; the calcinations were performed in the muffle of an ordinary assay furnace:

I. Ten grams of anhydrous zinc sulphate were calcined at orange heat (estimated to be 900° C.) for one hour. The product contained 2·68% S. Roasting for another hour at the same temperature eliminated all the sulphur.

II. Ten grams of $ZnSO_4$ were calcined at dull red heat (estimated to be 700° C.) for two hours. The product contained 14·78% S.

III. Ten grams of $ZnSO_4$ mixed with one gram of charcoal were roasted contemporaneously in the same muffle and for the same length of time as in experiment II. The product contained 9·58% S.

IV. Four grams of ZnS and the equivalent of $ZnSO_4$ were heated for 1 h. 10 m. in a covered crucible, first at cherry-red, which was raised gradually to orange heat. The product contained 0·76% S.

$ZnSO_4,ZnO$: The bibasic sulphate of zinc is obtained by digesting a solution of $ZnSO_4$ with an equivalent quantity of ZnO or ZnO,H_2O (prepared by precipitation from an equal quantity of $ZnSO_4$ solution). The solution of this salt, the bibasic sulphate, can not be crystallized, but can be decomposed by prolonged boiling.

$ZnSO_4,3ZnO$: Slow evaporation of a solution of the bibasic sulphate, as well as an addition of water to it, produces the tetrabasic sulphate, which presents itself as flexible, quadrangular, needle-shape crystals, retaining 10 molecules of water. The tetrabasic sulphate is produced also by the incomplete precipitation of a solution of the neutral sulphate by means of potash and dissolving the precipitate in boiling water. Upon cooling, the tetrabasic sulphate is deposited as small, unctuous crystals, retaining $2H_2O$. The tetrabasic sulphate is produced, moreover, by the prolonged digestion of the neutral sulphate with an excess of zinc, or or zinc oxide, which affords the salt with $10H_2O$ in opaque laminae or needles. The same salt can be obtained by partially decomposing the neutral sulphate by heat and taking up the residue by boiling water. The tetrabasic sulphate with $10H_2O$ dessicates slowly, forming a powder which is unaltered by the air. It loses $8H_2O$ between 100° and 125° C. and retains $2H_2O$ (Schindler), but according to Kühn the salt dried above 100° contains $4H_2O$ and the air dried salt, $8H_2O$. According to Büscher a compound with $7H_2O$ is obtained when 20 parts of borax in aqueous solution is added to a solution containing 60 parts of $ZnSO_4$ at 50° to 60° C.; the precipitate is free from boric acid.

The tetrabasic sulphate of zinc is supposed to be the form of basic sulphate, which is produced chiefly in roasting the neutral sulphate at the temperature at which the latter decomposes. At still higher temperature the tetrabasic sulphate itself is decomposed, forming ZnO, SO_3,SO_2 and O.

Tetrabasic sulphate of zinc is insoluble in cold water and but little soluble in boiling water.

$ZnSO_4,5ZnO+10H_2O$: The hydrated hexabasic sulphate of zinc is a white powder, which is formed by the action of water on the sulphate of zinc and ammonium ($ZnSO_4[NH_3]_2+H_2O$). It loses its water at 100°, and regains subsequently only about a third of it in the air.

$ZnSO_4,7ZnO+2H_2O$: The hydrated octobasic sulphate of zinc is precipitated by the addition of a large quantity of water to the bibasic salt. It forms a voluminous precipitate, which after drying is very light. It is insoluble in water. Digested with a solution of the neutral sulphate it forms tetrabasic sulphate. The octobasic sulphate with $2H_2O$ is also obtained by boiling a solution of neutral sulphate supersaturated with ammonia.

$ZnSO_4,H_2SO_4+8H_2O$: Von Kobell reported[1] the existence of an acid sulphate of zinc, obtained accidentally, as limpid, clinorhombic prisms, soluble in boiling water, but only slightly in cold water.

OXIDES.—Zinc is commonly considered as forming only one oxide, namely ZnO, but recently Robert C. Schuepphaus (in conjunction with E. Lungwitz) has pointed out the possibility of the existence of a lower, or at all events another, oxide. The known existence of a suboxide of cadmium, Cd_2O, points by analogy to the possibility of the existence of a similar suboxide of zinc. The possibility that there is a dioxide, ZnO_2, has also been pointed out.

Zinc oxide (ZnO) exists in nature as the mineral zinkite, or red zinc ore, which is found in considerable quantity in New Jersey, but is not elsewhere abundant. It may be prepared artificially (1) by oxidizing metallic zinc; (2) by roasting zinc sulphide, zinc sulphite and the various sulphates; and (3) by heating the carbonates, nitrates, hydrates and various other salts. In the previous chapter the conditions under which metallic zinc is oxidized by the atmosphere were described. The oxidation of the metal is also effected by heating with various salts which can supply oxygen, such as potassium chlorate and nitrate and arsenic acid.

Zinc oxide is normally a white powder of 5·5 to 5·7 sp. gr., which upon heating acquires a canary yellow color and upon cooling again regains its former appearance. It undergoes no chemical change in this procedure and is regarded as a permanent compound, for which reason and the ease of

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[1] Journ. f. prakt. Chem., XXVIII, 492.

obtaining it pure it is commonly used as the standard for the solutions employed in determining zinc in analytical chemistry.

In its pyrometallurgical behavior zinc oxide is infusible and is commonly regarded as non-volatile, but that idea is incorrect. Zinc oxide is certainly non-volatile at moderate temperatures and to only a slight extent at those which are attained in the roasting furnace (where the maximum is about 1000° C.) but according to experiments of Stahlschmidt[1] it is notably volatile at the melting point of silver (970° C., Holman), about 15% more at the melting point of copper (1054° C., Roberts-Austen) and rapidly at white heat. Roasted blende is not volatile at the melting point of silver, but is considerably volatile at that of copper.

Zinc oxide is reduced at high temperatures by various substances, among them carbon and carbon monoxide and upon that reaction is based the most important process in the practical metallurgy of zinc. The reduction of zinc oxide by carbon and carbon monoxide is express by the formulae:

$$(1)\ ZnO+C=Zn+CO \text{ and } (2)\ ZnO+CO=Zn+CO_2$$

The precise nature of the reaction which actually takes place in the reduction and distillation on a large scale, whether it be according to equation No. 1 or equation No. 2, or both, is unknown. Whatever it be, it is found necessary in practice to have a large excess of carbon in the retort. That excess of carbon among other purposes serves to reduce to monoxide such carbon dioxide as may be formed, which otherwise might act oxidizingly on the zinc (the reduction of carbonic dioxide by zinc taking place at red heat), while any zinc oxide which may perchance have been formed is reduced again by carbon or carbon monoxide, either or both. The net result, irrespective of what actually occurs, is zinc vapor and carbon monoxide, which alone, or practically alone, issue from the retort, when there is an excess of carbon present. Schnabel remarks[2] that in the case of a mixture of carbon monoxide and dioxide the amount of zinc oxidized by the latter is dependent upon its proportion in the mixture and the temperature.

The reduction of zinc oxide by carbon or carbon monoxide begins according to recent experiments by Robert C. Schuepphaus and E. Lungwitz[3] at 910° C, a statement which is confirmed by Hempel[4] who says that reduction begins under the boiling point of zinc (920°). It is completed at about 1,300° C., a temperature which is commonly attained in practical distillation. The comparatively low temperature at which the reduction begins,

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[1] Berg-u. Hüttenm. Ztg., 1875, p. 69.

[2] Handbuch der Metallhüttenkunde, II, 7.

[3] Journ. Soc. Chem. Ind., Nov. 30, 1899, p. 987.

[4] Berg-u. Hüttenm. Ztg., 1893, Nos. 41 and 42.

approaching closely that which is often reached in the roasting furnace indicates the danger of loss of zinc in the latter through reduction by carbonaceous matter in the gases of combustion; if the roasting be done in muffle furnaces the operation is of course free from danger in this respect. As would be naturally expected the temperature and facility of the complete reduction of zinc oxide is affected by the physical condition of that substance. Very finely divided, pure oxide, such as is prepared by calcining precipitated zinc carbonate is reduced quickly and completely at yellow heat. Zinc oxide from ores and in a coarser state of subdivision, on the other hand, requires a full white heat and a longer time. It is commonly accepted that zinc oxide obtained by roasting zinc blende is more difficultly reduced than that obtained by calcining zinc carbonate (smithsonite and hydrozinkite), but although that conclusion is drawn from and supported by the results of practice, I am unaware that it has been the subject of accurate pyrometrical investigation; the difference in reducibility is probably due to the physical condition of the ores rather than to inherent differences in the chemical properties of the oxides obtained in the various ways.

Hydrogen reduces zinc oxide at red heat but the steam produced by that reaction may under certain circumstances reoxidize the zinc to a more or less extent. Schnabel (op. cit., p. 7) quotes experiments by Deville[1] and Dick,[2] which indicated that if small quantities of hydrogen were passed quickly over glowing zinc oxide, chiefly zinc was obtained; while almost all the zinc was reoxidized if the current of hydrogen was slow. The oxidizing action of steam on zinc may be dependent upon both the proportion it bears to the hydrogen present, and the temperature. Deville was of the opinion that the latter played the more important part, since with a strong current of hydrogen a decrease in temperature took place, which was not the case with a weaker current. At the reduced temperature, thus caused, the steam cannot act oxidizingly on the zinc, as it can at the higher temperature.

Metallic iron is said by Percy to reduce zinc oxide at high temperatures. Metallic iron is frequently formed in the retort during distillation of a charge of zinc ore. The behavior of zinc oxide with zinc sulphide has been referred to in a previous paragraph. Sulphur itself when heated with zinc oxide produces zinc sulphide and sulphurous anhydride.

Zinc oxide fuses with carbonates of the alkali metals, forming colorless, transparent substances,[3] when the proportion of ZnO to the carbonate does not exceed 1:4. The chemical nature of these compounds is not described. They are probably zincates, compounds of zinc with potassium and sodium

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[1] Annales de Chimie et de Physique (3), XLIII, 479.

[2] Percy, Metallurgy.

[3] Berthier, Tr. de Essais, II, 567.

of that character being formed under certain circumstances in hydrometallurgy. Although zinc oxide is usually a base it sometimes plays the part of an acid, forming zincates with alumina, the alkalies, and the alkaline earths. In this respect zinc presents an analogy to beryllium (a member of the same group of elements according to the periodic classification) which with the alkalies, alkaline earths, etc., forms beryllates. The weak basic character of zinc is shown, moreover, in its tendency to form basic salts (such as basic sulphates, carbonates, etc.) in which it also resembles the chemical behavior of beryllium and magnesium. The strong bases do not readily form basic salts, but on the contrary form strong acid salts. Thus potassium and sodium form acid carbonates; calcium forms preferably an acid carbonate, but it is unstable; beryllium and magnesium form only extremely unstable compounds with carbonic acid, their basic properties not being sufficiently strong to hold them in combination with the weak acid except at a low temperature. The similarly weak union between zinc and carbonic acid is of practical importance in the calcination of calamine ores (smithsonite and hydrozinkite) which process is in all respects analogous to that of lime burning. The zincate of aluminum, or zinc spinel, occurs in nature as the mineral gahnite, and is also formed in the retort during distillation. Its properties are described in a subsequent paragraph. Zinc oxide also unites with the strong base litharge, lead oxide, forming a fluid, pale yellow slag when heated with eight times its weight of litharge, and a less fluid compound with a smaller proportion of litharge, down to the point where the mixture becomes infusible.[1]

Zinc oxide combines with silica, forming silicates, which are described under that caption.

With respect to the hydrometallurgical properties of zinc oxide, the limits of this treatise will permit of reference only to those which are of practical importance in the treatment of mixed ores and the production of the commercial salts of zinc. Zinc oxide is soluble in numerous acids, especially sulphuric, chlorhydric and nitric, with which it forms respectively sulphate, chloride and nitrate. It is soluble also in caustic soda and potash, forming zincates, and in ammonium carbonate and ammonia. Sulphurous acid takes it up as bisulphite. Ferric chloride and ferric sulphate dissolve it as chloride and sulphate respectively, an equivalent quantity of ferric oxide being precipitated. In water zinc oxide is quite insoluble. It is dissolved by acetic acid as acetate. Zinc oxide suspended in cold water is attacked by chlorine gas passed into the emulsion, zinc oxychloride and

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[1] Berthier, op. cit., I, 515.

hypochlorite being first formed and the zinc going finally into solution according to the equation:

$$2ZnO+4Cl=ZnCl_2+ZnCl_2O_2$$

According to E. A. Ashcroft[1] zinc oxide is converted directly to chloride by chlorine gas at a high temperature (upward of 600° C.) according to the reaction:

$$ZnO+2Cl=ZnCl_2+O$$

Hydroxide.—The hydroxide, or hydrate of zinc ($ZnO,H_2O=Zn[OH]_2$), is produced as a white amorphous substance by precipitation from solutions of zinc salts by caustic soda or caustic potash, avoiding an excess of alkali, since the precipitate is thereby redissolved, but if once dried it becomes less soluble in alkali. It is also precipitated by the hydrates of lime and magnesia. Zinc oxide does not unite directly with water. Hydrate of zinc is also produced by the galvanic action between zinc and iron, brass, or lead in an ammoniacal solution of zinc oxide. Zinc hydrate is easily decomposed by heat into the anhydrous oxide and water. It is soluble in acids and solutions of ammonium salts and the other substances which are solvents for zinc oxide.

Carbonates.—Zinc forms with carbonic acid a long series of carbonates and hydrocarbonates, certain of which exist in nature as the minerals smithsonite and hydrozinkite. In general they are prepared artificially by addition of sodium or potassium carbonate to a neutral solution of a salt of zinc, usually the sulphate. Because of the weak basic character of zinc as an element the basic carbonates are the more easily formed. The neutral carbonate of zinc cannot be obtained by precipitation with Na_2CO_3 or K_2CO_3 under ordinary circumstances; when either of those salts is added to a neutral solution of a salt of zinc there is a disengagement of carbonic acid and the precipitate formed is a basic hydrocarbonate. This hydrocarbonate is soluble in an aqueous solution of carbonic acid; upon exposure to the air a granular powder deposits from the solution, which has been taken for neutral carbonate of zinc by some chemists, but by others has been held to be a basic carbonate. According to Thorpe[2] neutral zinc carbonate, $ZnCO_3$, can be produced artificially by adding acid carbonate of sodium to a solution of zinc sulphate, while Roscoe and Schorlemmer state that it is produced by precipitation from $ZnSO_4$ with an excess of hydrogen potassium car-

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[1] In a paper on Sulphide Ore Treatment, read before the Institution of Mining and Metallurgy, London, June 19, 1901.

[2] Dictionary of Applied Chemistry, p. 1057.

bonate ($KHCO_3$), the neutral carbonate of sodium (Na_2CO_3) throwing down a basic carbonate of zinc, which is the more basic the higher the temperature of precipitation and the more dilute the solution.

Zinc carbonate, $ZnCO_3$, is produced when the solution of a zinc salt is precipitated by an alkaline carbonate under pressure. As obtained by heating a solution of $ZnCl_2$ to 150-160° C. and precipitating with calcium carbonate or sodium bicarbonate it is a white microcrystalline powder. The neutral hydrated carbonate ($2ZnCO_3,H_2O$) is an amorphous powder obtained by digesting a basic carbonate of zinc with ammonium bicarbonate. The anhydrous carbonate of zinc is soluble in carbonic acid water. A solution of CO_2 at five atmospheres, taking up 1/189 its own weight of $ZnCO_3$. On exposure to the air the solution becomes turbid, and on boiling a precipitate of zinc hydrocarbonate comes down.

The basic carbonates and basic hydrocarbonates of zinc are the more important compounds of zinc and carbonic acid. They are formed in the manner previously described. Their precise composition depends upon the conditions under which they are precipitated. All of these compounds lose their water and their carbonic acid at 300° C.[1] There is a long list of them and the following is probably only partially complete:

$2ZnCO_3,Zn(OH)_2$: This is the precipitate formed by adding an excess of alkaline bicarbonate to a solution of $ZnSO_4$ (Rose, *Poggend. Ann.* LXXV, 107; *Ann. de Chimie et de Physique* (3) XLII, 106).

$ZnCO_3,Zn(OH)_2+2H_2O$: This is a thin, white powder obtained by precipitating cold a solution of $ZnSO_4$ with sodium sesquicarbonate, washing with water and drying in the air (Boussingault, *Ann. de Chimie et de Physique* (2) XXIX, 284). When the precipitation takes place from hot solutions, the precipitate holds only one molecule of H_2O (Schindler).

$2ZnCO_3,3Zn(OH)_2$: Produced by the precipitation cold of a zinc solution by means of a neutral alkaline carbonate. A certain quantity of zinc carbonate remains dissolved by the carbonic acid set free in the solution, but is brought down by boiling. The precipitate generally entrains some alkali. If a boiling solution of $ZnSO_4$ be poured into a boiling solution of an alkaline carbonate the precipitate comes down as a light powder resembling magnesia; if the boiling be continued for some time, the precipitate is made free from alkali and the precipitation is complete. If the precipitation is effected by ammonium carbonate the precipitate is crystalline. This hydrocarbonate is soluble in 2,000 to 3,000 parts of cold water, separating out upon boiling, and also in solutions of ammonium salts, from which it

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[1] According to Rose, who states that zinc carbonates slowly decompose at this temperature.

disengages ammonia by boiling. According to Bensdorff it is this hydrocarbonate which forms the coating on sheet zinc exposed to air under a film of water. When dried at 100° C., this compound $[2ZnCO_3,3Zn(OH)_2]$ becomes $4ZnCO_3,7Zn(OH)_2+H_2O$.

$3ZnCO_3,5Zn(OH)_2+H_2O$: According to Lefort (*Journ. Pharm.* (2) XI, 329) the precipitates produced by alkaline carbonates from zinc solutions, hot or cold, have this composition. Schindler considers that the precipitate described above as $2ZnCO_3,3Zn(OH)_2$ is more accurately represented by the formula $3ZnCO_3,5Zn(OH)_2+H_2O$.

$ZnCO_3,5Zn(OH)_2+H_2O$: This is the natural carbonate—i.e., the mineral zinconise or hydrozinkite. Lefort attributes the same composition to precipitates brought down cold by alkaline bicarbonates; while H. Rose ascribed it to that obtained from neutral carbonates and very dilute zinc solutions, hot or cold.

$ZnCO_3,3ZnO+2H_2O$: This is the precipitate produced by sodium carbonate and the tetrabasic sulphate of zinc.

$ZnCO_3,7ZnO+2H_2O$: Obtained from sodium carbonate and the octobasic sulphate of zinc.

CHROMATE.—Zinc chromate, $ZnCrO_4$, is made by precipitation of a solution of zinc sulphate with neutral potassium chromate. If the solution is alkaline, zinc hydroxide is precipitated also; hence the method needs much care. Potassium bichromate cannot be used because of the ready solubility of zinc chromate in free acid. Zinc chromate can be prepared, however, by boiling zinc oxide with potassium bichromate. Zinc chromate is used as a pigment, having a light lemon color, which is permanent. It is not affected by sulphur, and can be mixed with other pigments. It is very soluble in mineral acids, and is decomposed by caustic alkalies.

SILICATES.—Zinc combines with silica in various proportions. The formula, $Zn_2SiO_4(=2ZnO,SiO_2)$, represents the singulo-silicate, in which form it occurs in nature as the minerals willemite and hemimorphite. The other silicates of zinc correspond to the usual formulæ—i.e., ZnO,SiO_2=the bisilicate, etc. These silicates are prepared artificially by heating zinc oxide with silica at high temperature.

All of the zinc silicates are difficultly fusible, the more so the higher their tenor in silica. According to Percy the bisilicate is infusible at the most intense white heat, while the singulo- and lower silicates melt at that temperature, forming more or less translucent slags of white-yellow and green-yellow color. The natural silicate smelts at the same temperature to an opaque, stony slag of grayish-green color.

The properties of zinc silicate are of importance in the practical metallurgy of zinc inasmuch as willemite and hemimorphite are common ores, while losses may occur through the formation of slag in the retorts in which zinc enters as silicate. The amount of zinc absorbed by such slags is rarely of much consequence, however, and the trouble from slags in zinc smelting is experienced chiefly from the other elements which enter into their composition. Zinc silicate is reduced by carbon.

ALUMINATE.—The aluminate of zinc ($Al_2O_3,ZnO=Al_2O_4Zn$) occurs in nature as the mineral gahnite. It may be produced artificially, as colorless, octahedral crystals, which are harder than quartz, by fusing a mixture of zinc oxide, alumina, and boric acid. Its specific gravity is 4·58. By heating an intimate mixture of ZnO and Al_2O_3, in the proportion of 1:6, Percy obtained a sintered, gray, stony mass, which scratched flint glass. The aluminate of zinc is frequently formed in the walls of the retorts, which are thereby colored a deep blue.

FERRATE.—Ferrate of zinc ($ZnFe_2O_4=ZnO,Fe_2O_3$) occurs as black and brilliant octahedral microscopic crystals in an analogous manner to the aluminate. Also if zinc oxide and ferric oxide be heated to redness, and the product be treated with insufficient chlorhydric acid to dissolve all the iron, a residue is obtained which contains zinc ferrate. This compound is one that should receive more study. There is reason to believe that it is formed during the roasting of ferruginous blendes, and if that be so it is probably a matter of considerable consequence in connection with hydrometallurgical processes for the extraction of zinc.

Zinc ferrate is slightly magnetic and has a specific gravity of 5·132. Apparently it is reduced by carbon. The mineral franklinite is a complex mangano-ferrate of zinc, iron and manganese.

CHLORIDES.—Zinc chloride ($ZnCl_2$), also known as zinc butter (French, beurre de zinc), is a compound of zinc which is used extensively in the arts and is the soluble form into which the zinc in ores is put for extraction by certain electrolytic and other processes. It is a white, deliquescent, wax-like substance of sp. gr. 2·75. It is a powerful caustic, being distinguished by its property of burning deeply and not merely superficially like many others. A concentrated solution of zinc chloride converts starch, cellulose, and a great many other organic substances into soluble compounds; hence the impossibility of filtering a strong solution of zinc chloride through paper.

Zinc chloride is formed when zinc oxide is heated to redness in chlorine gas, whereby the chlorine combines with the zinc and oxygen is liberated. Zinc chloride is also produced by the combustion of zinc in chlorine, wherein the metal burns brilliantly; by the action of chlorine on moist zinc at ordinary temperatures; by the distillation of two parts of mercuric chloride and one part of zinc; by the distillation of a mixture of anhydrous sulphate of zinc and sodium or calcium chloride; or from one part zinc oxide and two parts ammonium chloride; or by evaporating the solution of zinc oxide in chlorhydric acid and subliming the hydrous chloride ($ZnCl_2,H_2O$) thereby obtained; or by the addition of zinc to the fused chloride of an electronegative metal. Sodium chloride added to a solution of zinc sulphate gives zinc chloride and sodium sulphate, which may be separated by refrigeration.

Zinc chloride melts at about 262° C. to a brownish liquid, which boils at 708°-719° C., and volatilizes without decomposition at red heat. At a high temperature it is decomposed by aluminum with the formation of zinc and aluminum chloride. It is soluble in absolute alcohol and ether and is very soluble in water. The aqueous solution is decomposed by the electric current with the liberation of zinc at the cathode and chlorine at the anode. Molten zinc chloride is also decomposed electrolytically with the same results. Numerous processes for the extraction of zinc from ores have been based on these properties, although only one or two of them has come into practical use.

Zinc chloride has a remarkably affinity for water, so much so that it is used in the laboratory as an absorbent for water in the same manner as sulphuric acid is. It is, indeed, difficult to obtain the chloride free from water. When produced by crystallization from a concentrated solution it is obtained as $ZnCl_2+H_2O$. If on the other hand the solution be evaporated there is always some decomposition into basic chlorides and hydroxide, the following reactions taking place:[1]

$$ZnCl_2+H_2O=ZnO,HCl+HCl$$
$$ZnO,HCl+H_2O=ZnO,H_2O+HCl.$$

It is extremely difficult to dehydrate zinc chloride, without driving off chlorhydric acid and forming basic chloride. In boiling down a zinc chloride solution and fusing the salt in the ordinary manner from 3 to 5% of the zinc is oxidized and the equivalent of chlorhydric acid is driven off, while 3 to 5% of water always remains with the fused chloride even at high temperature; evaporation under a vacuum, however, is said to remove all the water besides preventing the formation of basic chloride.[2]

Basic chlorides, or oxychlorides, of zinc are also formed by dissolving zinc oxide or metallic zinc in a concentrated solution of zinc chloride; moreover in diluting an aqueous solution of zinc chloride to a certain degree there is a partial decomposition and formation of oxychloride. When zinc hydrox-

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[1] Remsen, Inorganic Chemistry, p. 613.

[2] Ashcroft, loc. cit.

ide is precipitated from a chloride solution by means of milk of lime or milk of magnesia, some oxychloride goes down with it. In the chloridizing roasting of zinc ore with salt for the formation of zinc chloride, a certain proportion of oxychloride, which is non-volatile at the sublimation temperature of the simple chloride, is formed. These oxychlorides are, however, volatile at a very high temperature, say 1,200° C., and possibly are to some extent decomposed into zinc oxide and chlorine. Three oxychlorides of zinc have been definitely described:

(1) $ZnCl_2,3ZnO$: dried at 38° C. it retains four molecules of water, two of which it loses at 100° C. It is soluble in acids and alkalies, but only slightly in water.

(2) $ZnCl_2,6ZnO$: this is formed by the action of water on the ammoniacal chlorides, $ZnCl_2,2NH_3$ and $ZnCl_2,4NH_3$; it is precipitated also by adding ammonia to a solution of zinc chloride in such a manner as to dissolve a part of the precipitate. At ordinary temperatures it holds $10H_2O$, and at 82° C., $6H_2O$; on calcination it loses water and some zinc chloride, and becomes a more basic oxychloride. It is insoluble in water.

(3) $ZnCl_2,9ZnO$: this is an insoluble white powder which remains when the residue from the evaporation of a solution of zinc chloride to sirup consistency is taken up with water; it is also produced by the addition of enough potash to a solution of zinc chloride to give an alkaline reaction; in the first case it holds back $3H_2O$, and in the second $14H_2O$.

The action of ammonium chloride upon molten zinc, which is taken advantage of in the process of galvanizing is represented by the equation:

$$Zn+2NH_4Cl=ZnCl_2+2NH_3+2H$$

The zinc chloride formed dissolves oxide from the surface of the metal forming zinc oxychloride.

Zinc chloride cannot be melted safely in clay or earthenware pots, which being porous to the chloride are subject to disintegration when heated. When free from lead and other chlorides and also from water, however, zinc chloride can be safely fused in iron pots and handled with iron tools; which it does not attack.

There are several ammoniacal chlorides of zinc. On adding ammonia to a concentrated, warm solution of zinc chloride until the precipitate formed is redissolved, there are separated, in cooling, crystals of the composition $ZnCl_2,4NH_3,H_2O$. Other compounds of this class are designated $ZnCl_2,2NH_3$ and $ZnCl_2,5NH_3,H_2O$.

Zinc chloride readily forms double chlorides analogous to those formed by magnesium chloride. The following combinations have been identified:

1. $ZnCl_2,4NH_4Cl$
2. $ZnCl_2,3NH_4Cl$
3. $ZnCl_2,2NH_4Cl+H_2O$
4. $ZnCl_2,NH_4Cl+2H_2O$
5. $ZnCl_2,2KCl$
6. $ZnCl_2,2NaCl+3H_2O$
7. $ZnCl_2,MgCl_2+6H_2O$
8. $ZnCl_2,BaCl_2+4H_2O$

The substance, $ZnCl_2,2NH_4Cl+H_2O$, is formed by mixing a solution of zinc in chlorhydric acid with a solution of ammonium chloride. It is used in soldering, serving to clean the surface of the metal by the action of the zinc chloride on the oxides.

According to Kremers[1] the specific gravity of solutions of simple zinc chloride ($ZnCl_2$) varies as follows:

Spec. Grav.	% $ZnCl_2$	Spec. Grav.	% $ZnCl_2$
1·1275	13·8	1·3869	37·5
1·2466	25·8	1·5551	49·2

Bromide.—Zinc bromide ($ZnBr_2$) is produced by the combustion of zinc in bromine vapor, and by the solution of zinc oxide in bromhydric acid. On evaporating this solution a crystalline, deliquescent mixture of hydrous bromide of zinc and zinc oxide is obtained from which by heating strongly zinc bromide may be sublimed. It is soluble in ether and alcohol and very soluble in water. Pure zinc bromide has a specific gravity of 3·643 at 10° C. It melts at 394° C. and boils between 650° and 700°.

Ammoniacal zinc bromide ($ZnBr_2,2NH_3$) crystallizes out of a solution of zinc bromide in warm ammonia water on cooling. It is decomposed by water and by heat.

Iodides.—Zinc iodide (ZnI_2) is produced by fusing zinc and iodine, or by action of iodohydric acid on zinc or zinc oxide. It is very soluble in water and on calcination is decomposed into iodine and zinc oxide. Zinc iodide has a specific gravity of 4·696. It melts at 446° C. and boils at 624° C.

The ammoniacal iodides, double iodides and oxyiodides of zinc are analogous to the bromine and chlorine compounds. The following have been identified.

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[1] Poggendorff s Annalen der Physik und Chemie CV, 360.

1. $ZnI_2,5NH_3$, formed by action of ammonia gas on the anhydrous iodide; decomposed by water.
2. $ZnI_2,4NH_3$; decomposed by water.
3. $ZnI_2,2NH_4I$.
4. $ZnI_2,2KI$.
5. $ZnI_2,2NaI+3H_2O$.
6. $2ZnI_2,BaI_2$.

FLUORIDES.—Zinc fluoride ($ZnFl_2$) is produced by precipitation with potassium fluoride from solutions of zinc salts; and by digestion of zinc oxide with fluorhydric acid. It is only slightly soluble in water, more soluble in dilute acids, especially in fluorhydric, and very soluble in ammonia. Its specific gravity is 4·84 at 15° C. There is a double fluoride of zinc and potassium, $ZnFl_2,2KFl$.

Heat of Formation of Various Compounds of Zinc. . .

The heat of formation of various compounds of zinc, according to Thomsen's thermochemical investigations[1] are given in the subjoined tables. All of Thomsen's experiments were made at about 18° C. The unit of heat referred to in these tables is the quantity required to raise the temperature of one gram of water 1° C. When it is said that the heat of formation of any compound is a certain number of units it is meant that such a quantity of heat is developed in the production of a quantity of the substance equal to its molecular weight in grams; e.g., the heat of formation of ZnO from Zn and O being 85,430 units and the molecular weight of ZnO being 65·4+16=81·4, there are developed 85,430 gram calories in the production of 81·4 g. of zinc oxide.

$Zn+O=ZnO$	85.430	Cal.
$Zn+O+H_2O=ZnO,H_2O$	82,680	"
$Zn+2Br=ZnBr_2$	75,930	"
$Zn+2Cl=ZnCl_2$	97.210	"
$Zn+2I=ZnI_2$	49,230	"
$Zn+2O+SO_2+7H_2O=ZnSO_4+7H_2O$	181,660	"
$Zn+2O+SO_2=ZnSO_4$	158,990	"

The solution of the zinc salts in water is attended by development of heat or the reverse.

$ZnCl_2$	dissolved in water evolves	15,630	Cal.
$ZnSO_4+7H_2O$	" " " absorbs	4.240	"
$ZnSO_4$	" " " evolves	18.500	"

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[1] Thermochemische Untersuchungen, III, 275 et seq.

The heat of formation in aqueous solutions is given in the following table:

$Zn+2Cl=ZnCl_2$	112,840	Cal.
$Zn+2HCl=ZnCl_2+2H$	34,200	"
$Zn+O+SO_3=ZnSO_4$	106,090	"
$ZnO+SO_3=ZnSO_4$	20,660	"
$ZnO,H_2O+SO_3=ZnSO_4+H_2O$	23,410	"
$ZnO,H_2O+2HCl=ZnCl_2+H_2O$	19,880	"
$ZnO,H_2O=2C_2H_4O_2=Zn(C_2H_3O_2)_2+2H_2O$	18,030	"
$Zn+2O+SO_2=ZnSO_4$	167,470	"
$Zn+2Br=ZnBr_2$	90,960	"
$Zn+2I=ZnI_2$	60,540	"

The heat of formation and the heat of decomposition of any substance are the same; i.e., in order to effect the decomposition the same quantity of heat must be supplied as was evolved in its formation. Thus the heat of formation of zinc chloride being 97,210 calories, its decomposition requires 97,210 calories.

VIII.

THE ORES OF ZINC.

Zinc ores are widely distributed throughout the world, workable deposits occurring in nearly every country of Europe and the North of Africa, and in various parts of the United States, while there are others in Australia, Canada and Mexico, and less well explored regions, which are not yet available to a very large extent on account of their inaccessibility. The principal ores of zinc are the sulphide, the carbonates, the silicates, the compound of zinc and manganese oxides (franklinite) and the simple oxide (zinkite), which rank in importance in the order mentioned. The carbonates and silicates are commonly referred to by the general term "calamine," which was formerly the only class of zinc ore used for the production of spelter, but the exhaustion of the easily worked surface deposits brought the undecomposed sulphide ore into the market, and during the last twenty years its importance has been steadily increasing. At the present time, however, a large part of the zinc produced in Europe is still derived from calamine, but in the United States blende is, and has been for many years, by far the more important of the two ores. The name "calamine" is used here, as uniformly throughout this treatise to indicate the class of zinc ores comprising both the carbonates and both the silicates, in which sense it is commonly employed in metallurgy, although this does not correspond with the mineralogical nomenclature generally accepted in the United States. A different custom prevails, however, in England, and owing to the consequent uncertainty that must necessarily attend the use of the word to designate a single mineral species it seems best to discard it for that purpose. This was discussed by me in a paper read before the American Institute of Mining Engineers in March, 1895, as follows:

"The hydrous carbonate is known mineralogically as hydrozinkite, zinconise, or zinc-bloom; the anhydrous silicate is recognized as willemite. With respect to the anhydrous carbonate and the hydrous silicate there is a confusion of name which is of old standing. Attempts to clear it away were long ago made by the mineralogists with the result that there is now a more or less national uniformity of nomenclature; but there is still an international disagreement, sometimes very perplexing and always leading to inexactness in expression.

"The name *calamine,* together with *Galmei* of the Germans, is commonly supposed to be derived from *καδμεια,* which was used by the Greeks to designate the peculiar kind of ore employed with copper in their brass-making, and also the accretions which formed in the brass-founder's furnaces. Agricola, however, says that it is from *calamus,* a reed, in allusion to the appearance of the material, *cadmia fornacum,* which collected on the furnace walls. But whatever the derivation of the word, it was used until within 100 years to include all the oxidized ores and compounds of zinc, both natural and artificial. Indeed, the difference between the carbonates and the silicates does not seem to have been suspected before 1780, when Bergmann published an account of certain experiments upon them; and it was not until 1803 that their true composition was made known by Smithson, and all doubts as to their being distinct mineral species were cleared away.

"The naming of these minerals is described by Dana in his *System of Mineralogy* (1892), pp. 548-549. In 1807, Brongniart called the silicate *calamine,* leaving for the other mineral its chemical name, *zinc carbonaté,* by which it continued to be known until, in 1832, Beudant called it *smithsonite.* In 1852, Brooke and Miller, with no good reason, reversed these names, and thus led to the confusion which still exists. On account of this confusion, Kenngott, in 1853, introduced for the silicate the name *hemimorphite,* which has not been generally accepted.

"At the present time American usage follows Dana, calling the anhydrous carbonate smithsonite and the hydrous silicate calamine. English mineralogists, on the contrary, generally employ calamine to designate the anhydrous carbonate, referring to the hydrous silicate as *electric calamine.* The application of the name smithsonite to the hydrous silicate by Brooke and Miller had a certain following in their time, but no longer obtains. On the Continent of Europe, however, the equivalent names, *calamine* and *Galmei,* are used in common parlance, especially in the zinc industry, to include the four mineral varieties, carbonates and silicates, hydrous and anhydrous. In Germany, many mineralogists use the nomenclature adopted by Dana, but most writers on technical subjects employ *Galmei* as a class name only, designating the silicates as *Kieselgalmei,* and the anhydrous carbonate as *edler Galmei, Smithsonite,* or *Zinkspath* (zincspar). French writers avoid confusion by using the chemical terms *zinc carbonaté* and *zinc silicaté,* although in France, as in Germany, *calamine (Galmei)* is employed by mineralogists as a purely scientific name for a distinct species—the hydrous silicate. The general meaning that the word calamine has on the Continent is undoubtedly a survival of the custom of the time when no difference in the oxidized ores of zinc was recognized, together with the fact that metallurgically they belong to the same class."

In the following summary of the properties and mineralogical characteristics of the various ores of zinc Dana's *System of Mineralogy,* sixth edition, has been drawn upon freely.

Blende.—Zinc sulphide, ZnS (also known as sphalerite, and by miners as black jack, German blende, French zinc sulfuré). This mineral when pure contains Zn 67·15% and S 32·85%, but it is usually contaminated with iron, manganese or cadmium, and rarely by mercury, lead and tin; traces of indium, gallium and thallium are found in blendes from certain localities. Blende is often argentiferous; less often auriferous. When quite pure, blende is white or nearly colorless, but commonly it appears yellow, brown, black, and also red and green, owing to impurities. Its streak is usually brown, but may be light yellow and white. Pure blende is transparent to translucent, but the ordinary varieties are opaque. It crystallizes in tetrahedral forms of the isometric system, but the crystals are frequently highly complex and distorted. Good crystals of blende are rather rare, however, and it is found commonly in crypto-crystalline to amorphous forms, the latter sometimes as a powder. It also occurs in foliated and fibrous forms. Its fracture is conchoidal; hardness, 3·5 to 4; specific gravity, 3·9 to 4·1; luster, resinous to adamantine. It is rather brittle, but not so much so as galena.

Blende is easily identified with the aid of the blow-pipe. In the open tube it gives off sulphurous fumes. In the reducing flame on charcoal it gives a coating of zinc oxide, which is yellow while hot and white after cooling; if, however, the mineral contains cadmium a reddish brown coating of cadmium oxide will be formed first. Moistened with cobalt nitrate solution the zinc coating gives a green color when heated in the oxidizing flame. Heated with soda on charcoal in the reducing flame (after a preliminary roasting) a strong green, zinc flame is emitted. Blende is difficultly fusible. It dissolves in chlorhydric acid with evolution of hydrogen sulphide. The mineral with which it is most likely to be confused is galena, which certain of the lustrous black cryptocrystalline varieties resemble strongly. The characteristic brown streak of blende and the black streak of galena, however, form an easy and infallible means of distinction.

Several varieties of blende are distinguished according to the presence of other sulphides, isomorphous with the zinc sulphide.

(a) *Ordinary,* containing little or no iron. Its color is usually white to yellowish brown, but is sometimes black. The red, or reddish brown, transparent crystallized kinds are sometimes called ruby blende or ruby zinc. Snow white blende of crystalline form has been found at Nordmark, Sweden, and at Franklin Furnace, N. J. A soft, white, amorphous deposit, forming a powdery mass at least 4 ft. thick and 30 ft. in length was once found at Galena, Kan.[1] Ordinary blende is most commonly of a brownish color and resinous appearance, whence it is frequently referred to by the miners as resin-jack. The Joplin blende is chiefly of this variety.

(b) *Marmatite,* ferriferous blende, containing 10% or more of iron, existing as monosulphide. The proportion of FeS to ZnS ranges as high as 1:2. Ferriferous blende is always dark brown to black in color. Its luster is sometimes dull; sometimes very bright, almost metallic. A large part of the blende which occurs in connection with galena and pyrite in the Rocky Mountains is of this variety. The percentage of iron in its composition is very variable, as to which the intensity of the black coloration is not a reliable guide, a very small proportion of iron sometimes producing a deep black color. Ferriferous blende differs from the ordinary variety in being magnetic, though only feebly so; its magnetic susceptibility increases with the percentage of iron in its composition.

(c) *Przibramite,* cadmiferous blende containing up to 5% Cd present as sulphide. Cadmiferous blende is usually reddish in color. Although cadmiferous blende with as much as 5% Cd has been described, such occurrence is rare, the cadmium tenor of commercial blende being very much lower. Edmund Jensch, who made an extremely elaborate investigation of this subject, reported that of the ore which came under his notice the kind richest in cadmium was a lot of 208 tons of black blende shipped from Abo, Finland, in 1890, which contained 0·46% Cd and 34·43% Zn.[2] Assuming that the cadmium was held entirely by the blende, the tenor of the latter would have been somewhat under 1% Cd.

(d) *Wurtzite,* a ferriferous blende corresponding to the formula, $6ZnS+FeS$, which has been found at Oruro, Bolivia, and at Przibram, Bohemia. It occurs as hexagonal crystals of 3·9 to 4·1 sp. gr. and 3·5 to 4 on the scale of hardness. Its color is brownish black and streak light brown. This variety is also sometimes cadmiferous.

The metallic mineral most commonly associated with blende is galena. The deposition of both these minerals seems to be favored by limestone, in connection with which rock the most important deposits are found.

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[1] American Journal of Science, 1890, XL, 160.

[2] Das Cadmium, sein Vorkommen, seine Darstellung und Verwendung, in Sammlung Chemischer und Chemisch-technischer Vorträge, III. vi. 201 to 232.

Voltzite.[1]—This is an oxysulphide of zinc, corresponding to the symbol, $ZnO+4ZnS$, which has been found in the form of incrustations at Pontgibaud, France, and Joachimsthal, Austria. It is a yellow mineral of 3·7 sp. gr. and 4·5 on the scale of hardness.

Goslarite.—Zinc sulphate, $ZnSO_4+7H_2O$, containing zinc oxide 28·2%, sulphuric anhydride 27·9%, and water 43·9%, results from the decomposition of blende, but owing to its easy solubility in water is rather rare in occurrence. It is a brittle mineral, of hardness 2 to 2·5, sp. g. 1·9 to 2·1, vitreous luster, white, reddish, yellowish and bluish in color, transparent to translucent, with an astringent, metallic and nauseous taste. In the closed tube it yields water and on charcoal gives the reactions for zinc, besides forming a sulphide which when moistened tarnishes silver.

Ferrogoslarite, in which part of the zinc sulphate is replaced by ferrous sulphate, occurs in some of the mines of Missouri and Kansas, where it forms mammillary or stalactitic incrustations of a light yellow to brown color. It is also common in the drainage of the mines of that region.

Smithsonite.—Zinc carbonate $ZnCO_3$ (also known as zinc spar, and by miners as dry bone, German Galmei, Edler Galmei, Kohlengalmei or Zinkspath, French zinc carbonaté). This mineral when pure contains carbonic dioxide 35·2%, and zinc oxide 64·8% (Zn, 52%). However, it is usually contaminated by iron, manganese or cadmium carbonates. Pure smithsonite is a brittle mineral of hardness 5, sp. gr. 4·3 to 4·45, uneven to imperfectly conchoidal fracture, vitreous luster, inclining to pearly, white streak, and color white, often grayish, greenish, brownish white and sometimes green, blue and brown. It is subtransparent to translucent. It crystallizes according to the rhombohedral system, but rarely occurs well crystallized, being found rather in granular and earthy forms.

Several varieties of smithsonite are distinguished according to the presence of foreign elements, catalogued as follows:

(a) *Ordinary,* classified as (1) crystallized, (2) botryoidal and stalactitic, closely resembling similar forms of hemimorphite or zinc silicate, (3) granular to compact masses, and (4) earthy, impure, occurring in nodular and cavernous masses, varying from grayish white to dark gray, brown brownish red and brownish black in color, and often with drusy surfaces in the cavities; the last variety is the "dry-bone" of American miners, which term also includes some hemimorphite.

(b) *Monheimite* or zinc-iron spar, ferriferous smithsonite, which often contains over 20% of iron carbonate.

(c) *Maganiferous smithsonite,* containing 5% or more of $MnCO_3$.

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[1] This spelling is adopted in the Century Dictionary instead of the older form "voltzine."

(d) *Cadmiferous smithsonite,* containing up to 5% of cadmium carbonate. In Arkansas and Missouri smithsonite is found of bright orange yellow color, due to greenockite, or cadmium sulphide; this is known locally as "turkey fat ore." At Wiesloch, in Baden, a yellow variety containing as much as 3% Cd used to be found.

These varieties are not often well defined, usually merging into one another by indefinite gradations. Thus there are many smithsonites which are both ferriferous and cadmiferous. The Silesian zinc carbonate ore is notably of this mixed type. As in the case of blende, the percentage of cadmium which is found in commercial smithsonite is very much less than would be inferred from the statements of the mineralogists. In an investigation of 10 years Jensch failed to find more than 0·3% Cd in any Silesian ore, while the average of all his analyses was only about 0·1%.

Before the blowpipe smithsonite is infusible, but when moistened with cobalt nitrate solution and heated in the oxidizing flame on charcoal it gives the characteristic reactions for zinc. In the closed tube it loses its carbonic anhydride. It is soluble in chlorhydric acid with effervescence, which reaction easily distinguishes it from all other zinc ores except hydrozinkite. The latter, however, gives off water in the closed tube, which smithsonite does not do.

FRANKLINITE.—A manganoferrate of iron, manganese and zinc [$(Fe,Zn,Mn)O, (Fe,Mn)_2O_3$]. This mineral, which is found in large quantities at Stirling Hill and Franklin Furnace, N. J., and is of rare occurrence elsewhere, contains 21% Zn when it corresponds to the formula here given; its composition is rather irregular, however, the relative quantities of the different metals varying rather widely, while conforming to the general formula of the spinel group. It is a brittle mineral of conchoidal to uneven fracture, hardness 5·5 to 6·5 and sp. gr. 5·07 to 5·22. It is opaque, with metallic luster, sometimes dull, and iron-black color. It is slightly magnetic, which property enables its separation from the willemite and zinkite, with which it occurs, by the powerful effect of the Wetherill magnetic machines.

It is easily identified by its reactions before the blowpipe. In the oxidizing flame with borax it gives a reddish amethystine bead, due to manganese, and in the reducing flame this becomes bottle green, due to iron. Fused with soda it gives a bluish green manganate, and when heated on charcoal a faint coating of zinc oxide, which is more marked when a mixture with borax and soda is used. Franklinite is soluble in chlorhydric acid. It is infusible before the blowpipe. Franklinite crystallizes in the isometric system, affecting the octahedral habit, but it usually occurs massive, granular, coarse or fine to compact.

ZINKITE.[1]—Zinc oxide, ZnO (known also as red zinc ore, German Rothzinkerz, French zinc oxyde). Zinkite when pure contains 80·25% Zn and 19·75% O. Manganese oxide is sometimes present. This mineral is of rare occurrence except at Stirling Hill and Franklin Furnace, N. J., where it is found in considerable quantity in connection with willemite and franklinite. It is a brittle mineral, with sub-conchoidal fracture, hardness 4 to 4·5, and sp. gr. 5·43 to 5·7. It is translucent to sub-translucent, has a sub-adamantine luster, and gives an orange yellow streak. Its color is deep red, also orange yellow, the former being the commoner. In crystallization it affects a hemimorphic form of the hexagonal system, but natural crystals are rare and it usually occurs foliated, massive, or granular.

Before the blowpipe it is infusible. Heated in the closed tube it blackens, but on cooling resumes the original color. On charcoal it responds to the characteristic tests for zinc. It is soluble in chlorhydric, nitric and sulphuric acids. With fluxes, on the platinum wire, it frequently gives a reaction for manganese, because of contaminating traces of that element.

HYDROZINKITE.—Zinc hydrocarbonate, $3ZnO,CO_2+2H_2O$ (also known as zinconise and zinc bloom; German, Zinkblüthe). Hydrozinkite when pure contains 11·1% water, 13·6% carbonic dioxide and 75·3% zinc oxide (57·1% Zn). It is an earthy, chalk-like mineral, which usually occurs massive, but sometimes as incrustations, which may be reniform, pisolitic, or stalactitic. Its hardness is 2 to 2·5, sp. gr. 3·58 to 3·8. Its luster is dull, streak white and shining, and color pure white, or grayish or yellowish white. With the blowpipe hydrozinkite gives the same reactions as smithsonite, with the addition that in the closed tube it yields water, by which it is distinguished.

Hydrozinkite is of rather rare occurrence as an ore, except in the provinces of Santander and Guipuzcoa in Spain, where it is found in large quantities. It occurs less extensively at Bleiberg and Raibel in Carinthia. In the United States it has been found at Friedensville, Penn., and in the Joplin district of Missouri, but the occurrences are only of mineralogical interest. There are several varieties of hydrozinkite, which are classified mineralogically as follows:

(a) *Ordinary,* as described above.

(b) *Auricalcite,* or "green calamine," corresponding to the symbol $2ZnCO_3,3ZnH_2O_2$, in which a part of the zinc is replaced by copper.

(c) *Buratite,* which contains both copper and calcium.

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[1] This spelling is adopted in the Century Dictionary.

WILLEMITE.—Zinc silicate, $2ZnO,SiO_2$ (known also as troostite). Willemite is a brittle mineral with conchoidal to uneven fracture, hardness 5·5 and sp. gr. 3·89 to 4·18. It is transparent to opaque, has a rather weak vitreo-resinous luster and white or greenish yellow color when purest; otherwise apple green, flesh red, grayish white or yellowish brown, the green color being most characteristic. When impure it is often dark brown. Its streak is white and reddish. The crystals of flesh red or gray color, opaque, found in New Jersey often pass under the name of troostite. Willemite crystallizes according to the rhombohedral system, occurring usually in hexagonal prisms. It also occurs massive and in disseminated grains.

Before the blowpipe willemite fuses with difficulty to a white enamel. Heated on charcoal it gives the characteristic reactions for zinc which are more pronounced when the mineral is mixed with soda. It is decomposed by chlorhydric acid with separation of gelatinous silica. The New Jersey variety phosphoresces with a green light after being struck with a hammer in the dark.

Willemite is found at the Vieille Montagne in Moresnet, but the only deposits of industrial importance are at Stirling Hill and Franklin Furnace, N. J., where the mineral occurs intimately mixed with franklinite and zinkite.

HEMIMORPHITE.[1]—Zinc hydrosilicate, $2ZnO,SiO_2+H_2O$ (also known as electric calamine and calamine; German Galmei, Kieselgalmei or Kieselzinkerz; French, zinc silicaté). Hemimorphite contains water 7·5%, silica 25%, and zinc oxide 67·5% (53·7% Zn). It is a brittle mineral of uneven to subconchoidal fracture, hardness 4·5 to 5, sp. gr. 3·4 to 3·5. It is transparent to translucent, has a vitreous luster, and is usually white in color, sometimes with a delicate bluish or greenish shade; also yellowish to brown. It crystallizes in the orthorhombic system, affecting hemimorphic forms, whence its mineralogical name. It also occurs massive and granular and in stalactitic, mammillary, botryoidal and fibrous forms.

Before the blowpipe hemimorphite is almost infusible. In the closed tube it decrepitates, whitens, and gives off water. On charcoal it gives the characteristic zinc reactions. It gelatinizes with acid, even when previously ignited, and is soluble in a strong solution of caustic potash.

Clays carrying varying amounts of zinc silicate are common in Missouri and Kansas and Virginia, where they are known as "joint clays" and "tal-

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[1] This name has been adopted here, contrary to the approved mineralogical nomenclature of the United States, because in a treatise of this character, wherein the term calamine is necessarily used in its metallurgical sense, it is absolutely essential to have a distinct designation for the hydrous silicate of zinc as a mineral.

low clays." The latter are very fine grained and plastic and on drying shrink and crumble into small fragments. Their tenor in zinc oxide varies chiefly between 30 and 40%.

Zinc.—Native zinc is said to have been found in large hexagonal crystals of the rhombohedral system, exhibiting a perfect, basal cleavage, in a geode in basalt, coated with smithsonite, erythine and aragonite, near Melbourne, Australia; also in the gold sands of the Mittamitta River. It is described as having a metallic luster, streak and color bluish white, hardness 2, and sp. gr. 7. There is some doubt as to the authenticity of these reported occurrences of native zinc.

IX.

OCCURRENCE OF ZINC ORE IN NORTH AMERICA.

Deposits of zinc ore are widely distributed in North America, but many of them afford mineral of undesirable character and many others are remote from transportation facilities, on which accounts the workable deposits under existing conditions are comparatively few in number. The most important of them are situated in the Joplin district of Missouri and Kansas, comprising the southwestern corner of the former State and the southeastern corner of the latter, at Stirling Hill and Franklin Furnace, N. J., in the vicinity of Shullsburg, Wis., and the vicinity of Pulaski, Va. Other deposits which have been exploited in the past and may be reopened, or are being worked on a small scale are situated at Friedensville, Penn.; in Wythe County, Va.; at Mossy Creek, Straight Creek and Lead Mine Bend, Tenn., and in Arkansas. In Kentucky there are deposits, which are thought to have some promise, near Marion, in Crittenden County, a short distance south of the Ohio River, in the western part of the State; production therefrom was begun in 1901.

Huge deposits of mixed ores, containing blende, pyrite and galena exist in Colorado, especially at Leadville and Kokomo, and numerous attempts have been made to smelt, or treat otherwise, the blende separated therefrom by mechanical (gravity or magnetic) dressing; lately a product sufficiently clean has been made to find a market with the smelters of Europe and Kansas, while a better grade of ore has been produced to a small amount at Creede, Colo. Deposits of zinc ore existing at Hanover, N. M., have also been worked intermittently.

The Joplin district is by far the most important source of zine ore in North America, and it, together with the New Jersey mines, overshadows all the others. Unfortunately, there are no statistics of the production of zinc ore in the United States, other than those presented in Chapter IV.

The geological occurrence and the date of earliest exploitation (in a serious way) of the principal zinc ore deposits of the United States are summarized in the following statement in tabular form:

ZINC ORE DEPOSITS OF THE UNITED STATES

State	Character of Mineral	Country Rock	Geological Age	Date of First Exploitation
Arkansas	Calamine and blende	Limestone	Lower Carboniferous	...
Missouri-Kansas .	Calamine and blende	Limestone	Lower Carboniferous	1870
New Jersey	Franklinite and willemite	Limestone	Cambrian	1840
New Mexico......	Calamine and blende	Limestone	Lower Carbon iferous	....
Pennsylvania.....	Calamine and blende	Limestone	Lower Silurian	1853
Tennessee........	Calamine and blende	Limestone	....	....
Virginia..........	Calamine	Limestone	Lower Silurian	1879
Wisconsin	Calamine and blende	Limestone	Lower Silurian	1860

ARKANSAS.—Zinc ore has been discovered at numerous points in this State, but few of the deposits have yet been exploited and those only in a desultory manner, owing to lack of transportation facilities heretofore. The deposits which have attracted most attention are situated in Marion, Carroll, Boone, Sharp and Lawrence Counties in the northern part of the State. They are described as irregularly distributed in crevices and along joint planes in magnesian limestone, the crevices frequently enlarging into pockets or cavernous spaces filled with ore, clay and other gangue. At some points the ore has impregnated certain strata and thus occurs at a definite horizon. The Rush Creek district in Marion County is of the latter character. The principal developments in Arkansas have been made in that district, the ore having been shipped in flatboats down the White River to Batesville, a distance of about 100 miles, and thence by the Iron Mountain Railway to St. Louis smelters. The production up to the present time has been intermittent and small, probably not to exceed 1,000 tons in any single year, but in 1901 renewed attention was given to the district because of the certainty of its being speedily opened by several lines of railway which were already building into it. Some of these lines have now been completed.

The zinc region of Northern Arkansas lies, generally speaking, to the north of the Boston Mountains and west of the St. Louis, Iron Mountain & Southern Railway, but the prospecting for and the development of the ore deposits has not yet gone far enough to outline more than vaguely the areas over which workable deposits exist or should be looked for. The tardiness of the development of the region is due in part to its topography, it being a hilly and partly mountainous country, through which there have been no railways until recently. According to Doctor John C. Branner, now of Stanford University, California, and formerly State Geologist of Arkansas, there is no longer any doubt, however, about the existence in that region of large bodies of zinc ore.[1]

Doctor Branner states that the zinc deposits of Northern Arkansas occur in limestone and dolomite of Lower Carboniferous age. The occurrences are classed genetically as (I) bedded deposits mostly contemporaneous with the rocks in which they occur; (II) vein deposits of older age than the inclosing rocks, and occurring (*a*) along faults or fractures, or (*b*) filling brecciated beds formed along underground water courses; (III) alteration products, chiefly carbonate and silicate ores, derived by alteration from the sulphide ores of the first and second divisions. Both the blende and the calamine of Northern Arkansas are remarkable for their purity. Out of a large number of analyses of the blende the largest quantity of iron found was 0·67%. Selected samples of smithsonite from the Morning Star mine showed 51·60% Zn; from the Legal Tender mine 49·91%. The early zinc mines of Arkansas were opened on deposits of smithsonite in the surface clays and soils, along and near the outcrops of deposits of blende. Although there is hardly a prospect in the region which has not yielded some smithsonite, there is but little search nowadays for smithsonite alone. Doctor Branner feels reasonably confident, however, that when the search for zinc ore in Northern Arkansas has been properly systematized large bodies of smithsonite will be discovered, most likely in regions of deep rock decay. Zinc silicate ore is much less abundant in Northern Arkansas than either blende or smithsonite. The most abundant deposits now known are in the Sugar Orchard district.

Besides in Northern Arkansas there appear to be promising resources of zinc ore in the southern part of the State, not far from an existing line of railway. At the Petty mine of the North American Ore and Metal Co., in Sevier County, about 70 miles north of Texarkana and four miles from the Indian Territory, there is a shaft 175 ft. deep, with several levels opened therefrom, which has produced about 700 tons of ore. The vein is a fissure in black slate, which is traversed by dikes of diorite. The ore is a mixture of blende, chalcopyrite and argentiferous galena. Ordinary jigging has afforded a product assaying 56% Zn and 3·5% Fe, but with the aid of magnetic separators a concentrate assaying 60% Zn and 1·81% Fe has been produced experimentally.

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[1] In a paper on the "Zinc and Lead Deposits of North Arkansas," read before the American Institute of Mining Engineers, Nov. 1901.

COLORADO.—The immense shoots of sulphide ore at Leadville have large quantities of a mixture of blende, pyrites and galena which assays about 25% Zn, 10% Pb, 22% Fe, 39% S, 4% SiO_2 and 10 oz. silver per 2,000 lb. In 1889 the superintendents of four mines estimated that they had in the aggregate 1,000,000 tons of such ore actually blocked out. At the present time the quantity opened is certainly very much larger. This ore was early worked to a considerable extent by crushing and jigging, by which process a marketable lead ore was got and zinky tailings, high in iron and lead, were thrown away. There was never much profit from the operation. Numerous attempts were made to work the zinky tailings, by direct smelting for zinc,[1] by hydrometallurgical methods, and by careful gravity separation. The last method, especially by employment of improved shaking tables (like the Wilfley, Cammett, Bartlett and others of that type), has lately made it possible to obtain a clean galena and pyrite concentrate and a by-product assaying about 45% Zn, 12% Fe and 6% Pb, which has found a market with zinc smelters, and although fetching only a low price[2] has made the operation of concentration a profitable one. In 1899, 1900 and 1901 a considerable quantity of such material was shipped to Mineral Point, Wis., Iola, Kan., Antwerp, Belgium, and Swansea, Wales. Recently the separation of blende from Leadville mixed sulphide ore by means of the Wetherill magnetic machines has been undertaken by the New Jersey Zinc Co. at Cañon City and the Colorado Zinc Co. at Denver. The method employed is described in Chapter XI. By this system a concentrate assaying about 50% Zn, 10% Fe and 1% Pb is produced. Some analyses of the Leadville mixed sulphides as mined are given in the subjoined table.

ANALYSES OF LEADVILLE MIXED SULPHIDE ORES.

Name of Mine.	Zn. %	Pb. %	Fe. %	S. %	SiO_2 %	Ag. oz.
Col. Sellers	24·50	10·70	16·60	40·00	3·40	[a]54·30
" "	19·25	16·19	[b]24·60	39·24	0·96	7·50
Moyer................	24·00	15·00	16·00	40·00	....	11·00
Sierra Nevada	24·00	[d]9·20	21·80	....	3·20	[c]4·30
" "	32·40	[e]4·30	20·60	....	....	4·20

[a] The great mass of this ore does not contain more than 10 oz. Ag, but it is not uncommon to find spots where the silver assay is as high as, or even higher than, in this analysis. [b] Includes 1·66% Mn. [c] This ore also contains 0·03 oz. Au. [d] By fire assay. [e] By volumetric analysis.

Since 1899 considerable zinc ore has been produced by mines at Creede, where it is concentrated to a product assaying 55 to 59% Zn, 3·75 to 6% Pb

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[1] Denver Zinc Co., 1887.

[2] About \$6·50 per ton, f. o. b. Leadville.

and 1·1 to 2·1% Fe, which has been shipped to smelters in Kansas. The Creede blende generally carries 2 or 3 oz. Ag per ton and sometimes is auriferous. Blende concentrates have also been produced at Montezuma, Summit County, and at Rico. The Rico Mining and Milling Co. made shipments to Belgium in 1901 of zinc concentrates obtained by the magnetic separation of ore from the Atlantic Cable mine at Rico. The crude ore, which is both gold and silver bearing, is said to assay about 25% Zn. The magnetic separating mill, which has a capacity of 60 tons of crude ore per day, makes a zinc concentrate assaying from 55 to 60% Zn and less than 4% Fe; and a lead concentrate assaying upward of 60% Pb and less than 3% Zn.[1]

Important deposits of mixed sulphides, somewhat similar to those of Leadville, exist at Kokomo, Summit County, where there is a bedded vein, 10 to 12 ft. thick, of great extent. In this there are shoots of high grade silver-lead ore, but the great mass of the ore is very low in lead, or destitute of it, being essentially a mixture of blende and pyrite. Its composition is approximately 20% Zn and 28% Fe. It contains 4 or 5 oz. Ag and 0·2 to 0·4 oz. Au.

KENTUCKY.—Zinc ore occurs in Kentucky near Marion in Crittenden County, where it is found in connection with the deposits of fluorspar and galena which have been worked to some extent. The Columbia mine is said to disclose a fissure vein along the line of a fault of which the displacement has been approximately 500 ft. One of the walls is limestone; the other a sandstone or quartzite. The vein varies in width from 18 in. to 6 ft. In some places the ore spreads out laterally as "flats" or beds. The ore consists of galena and blende, with a gangue of fluorite, some calcite, a little clay, and small quantities of barytes and quartz. In most parts of the vein only the ore and fluorite are met with. The galena occurs in bunches and crystals scattered through the fluorspar and constitutes about 20% of the entire vein material. The percentage of blende appears to be rather less than the percentage of galena so far as the vein has been opened. The mine is situated about four miles from Marion. Its development has not been extensive, the exploitation having been done in a desultory manner, and chiefly for galena and fluorspar rather than for zinc. The Old Jim mine, adjoining the Columbia, produced a considerable quantity of calamine from surface workings in 1901.

The Tabb mine, south of Marion, is said to show a wide vein, standing vertically and of great longitudinal extent, of which one wall is limestone and the other a sandstone or quartzite of undetermined character. The

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[1] Eng. and Min. Journ., Nov. 30, 1901, p. 731.

mineral is a resinous blende, low in iron, which is very finely disseminated in a gangue of fluorspar. The ore is said to assay 25 to 35% Zn. Owing to the finely disseminated occurrence of the mineral and the difficulty of separating it from the fluorspar (sp. gr. 3·1 to 3·2), which the owners of the property have not yet been able to accomplish, no mining has yet been done, except a small amount of development work.[1]

MISSOURI AND KANSAS.—Zinc ore is found in several parts of Missouri, but the only important mines are those of the Joplin district, by which designation is understood a more or less elliptical area comprising the mines of Galena, Kan., Webb City, Carterville and Joplin in Jasper County, Mo., Granby in Newton County and Aurora in Lawrence County, besides various less well known mining centers. Circumscribing this area by an ellipse, its long axis, extending from Aurora, Mo., on the east to Galena, Kan., on the west, a distance of about 50 miles, has a course a little north of west, while its width on the short axis extending north from Granby, Mo., is about 25 miles. The country rock of the region is chiefly limestone of the Lower Carboniferous formation, which immediately underlies the adjacent coal measures of Kansas. This limestone is not everywhere ore bearing throughout the district, but only in local areas where the conditions have favored the deposition of mineral. Surrounding such areas are broad tracts of barren ground. The geographical position of the zinc bearing areas, together with the location of the coal and natural gas fields of Kansas and the principal smelting points, is shown in the accompanying map, of which the scale is about 18 miles to the inch.

Nature of the Ore Deposits.—Broadly considered the ore deposits are lenticular masses of brecciated and mineralized chert, more or less intermingled with limestone or the products of its alteration and decomposition, and surrounded everywhere by limestone. Many of these lenses are of great size, especially in the vicinity of Webb City and Joplin, Mo., and Galena, Kan., where stopes occur 75 to 150 ft. wide, 40 to 80 ft. high, and 200 to 400 ft. long, from which all the material extracted has been milled.[2] The smaller lenses are 15 to 50 ft. wide, 5 to 30 ft. high and 100 to 500 ft. long. In one instance near Joplin a channel of ore was followed for 1,000 ft. These lenses and channels are frequently of highly irregular shape, often sending out sheets and pipes into the surrounding barren country rock. They are connected with a system of fissures in the country rock

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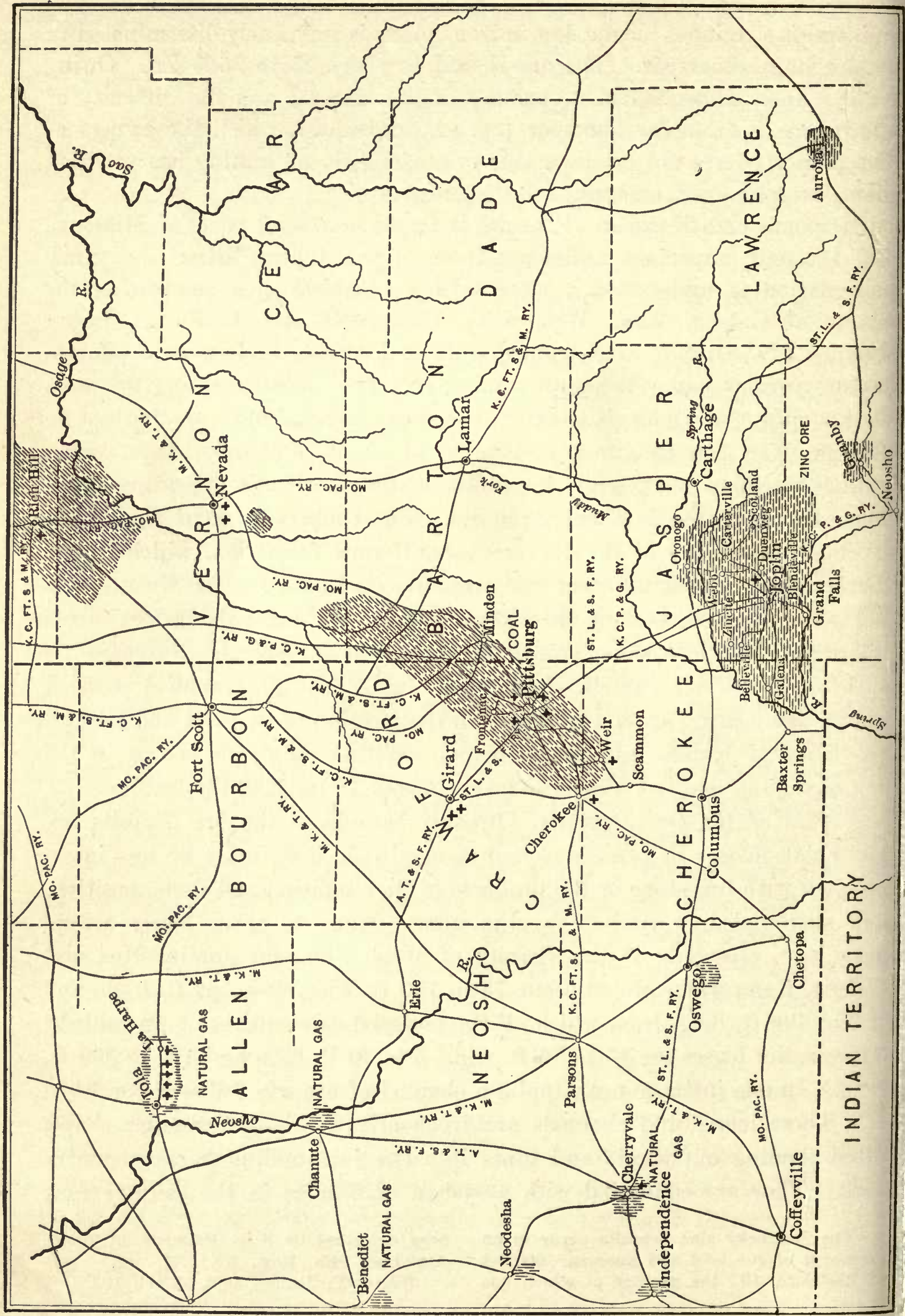
[1] The Kentucky zinc deposits occur in an extension of the lead and fluorspar district of Rosiclare, Ill., the geology of which has been described by S. F. Emmons, in Trans. Am. Inst. Min. Eng., XXI, 31.

[2] Walter P. Jenney, ibid., XXII, 193.


CEDAR
DADE
LAWRENCE
VERNON
BARTON
JASPER
BOURBON
CRAWFORD
CHEROKEE
ALLEN
NEOSHO
INDIAN TERRITORY
Sac R.
Osage R.
Rich Hill
Nevada
Lamar
Carthage
Aurora
Granby
Neosho
Fort Scott
Minden
Pittsburg
COAL
Frontenac
Girard
Weir
Scammon
Cherokee
Columbus
Baxter Springs
Oronogo
Carterville
Scotland
Webb
Duenweg
Joplin
Zincite
Blendeville
Belleville
Galena
Grand Falls
ZINC ORE
Spring R.
Muddy Fork
Iola
La Harpe
NATURAL GAS
Erie
Neosho R.
Parsons
Oswego
Chetopa
Chanute
Cherryvale
Independence
Neodesha
Benedict
Coffeyville
K. C. FT. S. & M. RY.
M. K. & T. RY.
MO. PAC. RY.
K. C. P. & G. RY.
ST. L. & S. F. RY.
A. T. & S. F. RY.

and occasionally the latter are found mineralized in coincident sheets. Thus, Jenney (loc. cit.) describes vertical fissures in the district which traverse the limestone without disturbing the stratification or producing any brecciation except between the cheeks of the fissures, enclosing veins of mineral after the usual manner of "fissure veins." A fissure of that character near Joplin carried ore to a depth of 60 ft. with a longitudinal extent of 200 ft. and thickness of 4 to 12 ft. between walls.

The more common lenticular masses of mineral bearing chert in general lie nearly horizontal. As would be naturally expected, the distribution of mineral through the chert is irregular, The blende occurs impregnated in seams and bunches in the chert, so that frequently the fragments of the latter appear cemented together by the blende, which everywhere permeates the mass in a net-work. As the ore becomes richer the seams of blende increase in thickness and sometimes pockets of solid blende or loose aggregates of crystals (gravel jack) are found. Usually the proportion of blende increases in the lower part of the deposit, where it sometimes entirely replaces the chert, but at the bottom of the deposit there is commonly a layer of very solid, dense chert, which cuts off the zinc ore. It is often the case, however, that another ore body comes in under such a layer of chert, and there are instances where four successive ore bodies have been opened, one under the other, with intervening partings of chert. The ore bodies are sometimes identical with the chert lenses, so that the entire lense is more or less impregnated with blende, but more frequently separate ore bodies occur in various parts of the larger lenses.

Grade of the Ore.—It will be inferred from the foregoing description that the grade of the ore of the Joplin district is very variable, which is in fact the case, the run of the ore mined ranging from an insignificant percentage of zinc to nearly pure blende. Important bodies of the latter character are rare, however, and in general the ore raised must be dressed. With the best ores one ton of concentrate assaying 60% Zn is obtained from two to five tons of crude ore, while the poorer ores mined yield only one ton out of 20 or 25 tons. It is futile to attempt to reckon the tenor of zinc in the crude ore from the above statement, since there are extremely variable, but generally high, losses in dressing according to the Joplin practice. This subject is discussed at more length in Chapter XI, q. v. J. R. Holibaugh, in *The Mineral Industry,* II, 670, stated that some ore bodies would yield 75% to 80% mineral, while in others the average would not be over 10%. In *The Mineral Industry,* III, 537, he stated that at that time (1894) few mines could be worked at a profit which did not yield 10% of mineral, the average value of blende in that year having been $17·10 per 2,000 lb.

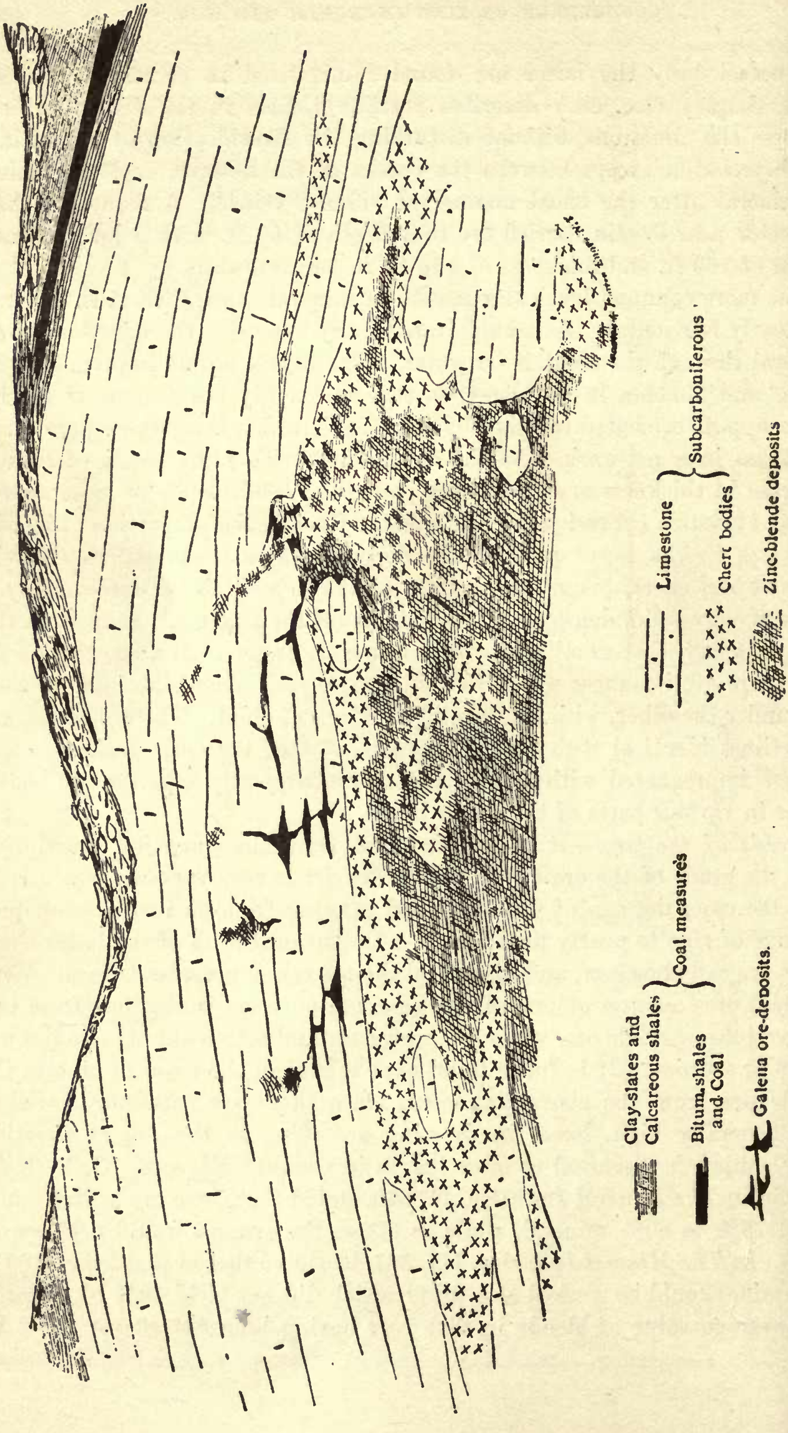


FIG. 10.—TYPICAL ORE-DEPOSIT NEAR WEBB CITY, MO.

at the mines, though a large deposit of ore yielding 7% had been worked profitably in the Victor mine at Carterville, a large tonnage being handled daily. At the present time the average yield of the district is probably not more than 5% and many mines that yield only 4% are being worked.[1]

Deposits of smithsonite and hemimorphite, classed together as "silicate ore" by the miners of the district, occur especially in the vicinities of Aurora and Granby, where ore of that character is got rich enough to ship in lump form. As sent to the smelters it probably averages 40 to 45% Zn. Elsewhere in the Joplin district blende is the predominant mineral. Galena is commonly associated with it, but it has been found that the latter mineral more frequently occurs in the formations near the surface, while in the lower parts of the ore bodies there is less of it, and often none at all.

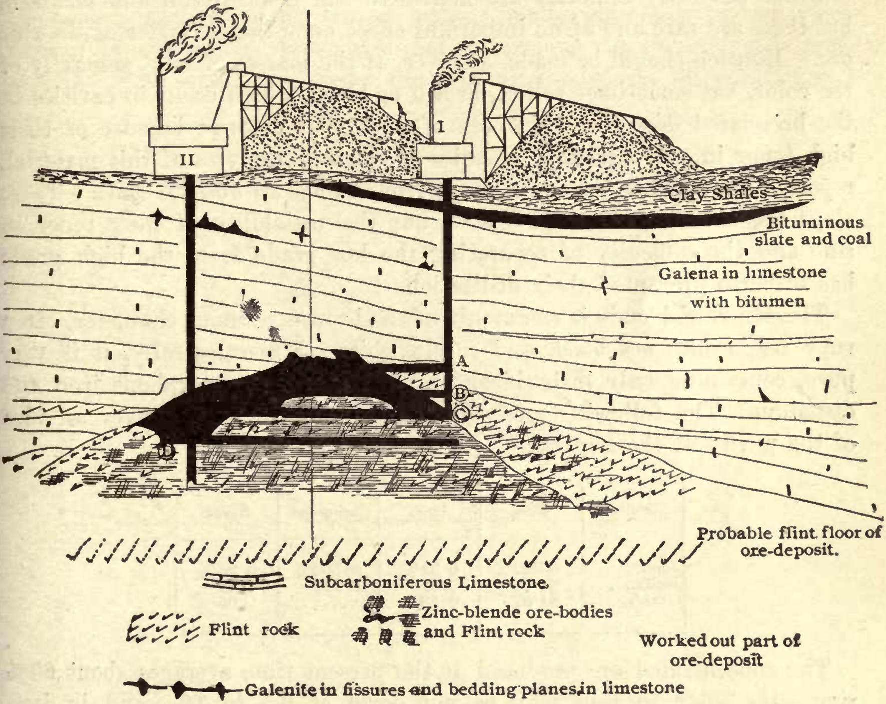


FIG. 11.—VERTICAL SECTION OF A TYPICAL ORE DEPOSIT NEAR WEBB CITY, MO.[2]

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[1] There is no doubt that the average yield per ton of crude ore mined and milled in the Joplin district has greatly diminished during the last 10 years.

[2] These illustrations are reproduced from the paper on "Zinc Blende Mines and Mining near Webb City, Mo.," by Carl Henrich, in Trans. Am. Inst. Min. Eng., XXI, 3.

Where the two minerals occur together there is no sharp line of demarkation, but in dressing there is no difficulty in separating the two minerals when intermingled. As the surface deposits are exhausted the proportion of lead ore to zinc ore won is decreasing.[1] Fortunately, there is not as a rule a large percentage of pyrite associated with the blende of the Joplin district, but there are some mines where it runs so high as to prevent the production of a marketable ore by ordinary gravity concentration.

Other accessory minerals are marcasite and chalcopyrite and bitumen, but these are rare and of no important effect upon the character of the zinc ore. Mention should be made, however, of the masses of clay, generally of red color, but sometimes yellow, brown or black, which occur in cavities in the brecciated deposits, which are of particular interest because of their high tenor in zinc silicate. A series of thirteen analyses of this material, reported by Winslow,[2] shows a range from 2·36% to 56·12% ZnO. These clays are abundant in some places, but the variability of their tenor in zinc and the difficulty of separating the low grade from the high grade has hitherto prevented their utilization.

The Missouri blende is commonly of the brown, resinous character, often ruby red, sometimes black and rarely white. Mineralogically, it is very pure, containing only insignificant proportions of isormorphous iron and cadmium. The following analyses, although of old date, convey an idea of the purity of the mineral:

Locality.	Zinc.	Iron.	Cadmium.	Silica.
Joplin.....	65·92%	0·32%	0·509%	0·25%
Joplin.....	64·87	0·37	0·723	1·41
Granby....	64·67	0·53	....	2·05

The concentrated ore produced at the present time averages about 60% zinc. Its tenor in lead may be put down at 0·5 to 1%, and in iron from 1 to 2%. It assays about 30% S, and the remainder of its composition, besides a little cadmium, is silica. It is free from both arsenic and selenium.

Mining Conditions.—The operating of mines on leased land is almost a universal practice in the district. Under this system the fee-owner usually

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[1] The production of lead ore and zinc ore during the last seven years was as follows, the zinc ore being stated in brackets: 1895, 31,294 (144,487); 1896, 26,927 (153,082); 1897, 29,578 (177,975); 1898, 26,457 (235,123); 1899, 24,100 (256,456); 1900, 28,500 (242,500); 1901, 35,000 (258,000). According to these figures the ratios are: 1895, 1:4·6; 1896, 1:5·7; 1897, 1:6; 1898, 1:9; 1899, 1:10·6; 1900, 1:8·5; 1901, 1:7·4. The Joplin lead ore, as concentrated, assays about 77% Pb on the average, the best grades having a tenor of about 80% Pb.

[2] Missouri Geological Survey, VII, ii, 445.

gives a lease on a large tract of his land at a royalty of from 8 to 15% of the gross output of mineral, the lease being made usually for a period of 10 years. The lessee prospects the tract by drilling or sinking shafts, and if mineral is found, sub-leases portions of the property, usually lots of 200 ft. square, at a royalty of 15 to 25% of the gross output of mineral. In some cases the fee-owner operates his own mines and frequently a lessee prefers to work a rich deposit rather than to sub-lease it. This system has at least one good result, inasmuch as it makes the district largely free from labor difficulties, so many miners being directly interested in the operations. Experience has shown also that in the long run the system is probably the most economical under the existing conditions, since although a company operating its own property on a large scale can mine and mill most cheaply, lessees are at an advantage in exploring for new ore bodies.

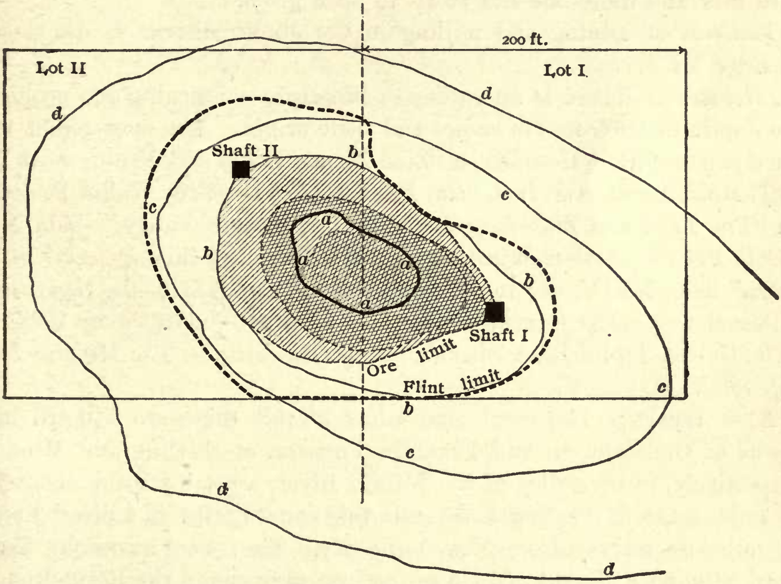

FIG. 12.—PLAN OF ORE DEPOSIT SHOWN IN FIG. 11.

Limits of ore or flint body are *aaa* at level A, Fig. 11; *bbb* at B; *ccc* at C; and *ddd* at D. The dotted line incloses the area within which there is no adequate support of the roof of the ore deposit.

Mining is done in the Joplin district from the grass-roots down to 250 ft., the average depth of all the shafts operated being probably about 120 ft. Many drill-holes have recently been put down to depths of 250 or 300 ft.

and rich ore has been struck at greater depths. The contract price of churn-drilling to ordinary depths is $1·25 per ft., but better figures can be obtained on large contracts. Owing to the shattered condition of the rock formation and the presence of chert in irregular and broken masses, the diamond drill has not been successful in the Joplin district. The cost of shaft-sinking is very variable, depending on the locality or kind of ground. According to Harold A. Titcomb,[1] a 6×7 ft. shaft was sunk 80 ft. on contract in the Belleville district for $2 per ft.; this shaft was in soft open ground. A 5×7 ft. shaft, 70 ft. deep, two miles west of Joplin, cost $20·25 per ft., that figure including a small boiler and pump. A shaft 125 ft. deep was sunk one mile west from Joplin at a cost of $14·30 per ft.; 90 ft. was in hard flint and limestone and 35 ft. in open ground.

The cost of mining and milling in the Joplin district is discussed in Chapter XI.

Literature.—There is an extensive literature concerning the geology of the Joplin district, its ore bodies and their origin. The most recent works are a paper by Carl Henrich on "Zinc Blende Mines and Mining near Webb City, Mo.," *Trans. Am. Inst. Min. Eng.*, XXI, 3 to 25; by Walter P. Jenney, on "The Lead and Zinc Deposits of the Mississippi Valley," ibid., XXII, 171 to 225; by Arthur Winslow, on "The Lead and Zinc Deposits of Missouri," ibid., XXIV, 634 to 689; and vols. VI and VII of the report of the Missouri Geological Survey, by Arthur Winslow, published in 1894; also "The Galena-Joplin Lead District," by E. Haworth, in *The Mineral Industry*, vol. VIII.

NEW JERSEY.—The great zinc mines of this State are situated in the towns of Ogdensburgh and Franklin Furnace, at Stirling and Mine hills respectively, in the valley of the Walkill River, a small stream, about 12 to 15 miles south of the New York state line and 40 miles in a direct line and 60 miles by railway from New York city. They were owned by General Lord Stirling and worked by him for iron previous to the Revolution, and about 1840 were first worked for zinc, though for the next 10 or 15 years the various enterprises were more or less unsuccessful. The zinc mines of this district have had an eventful history, and were involved in what appeared to be an endless litigation until in 1895 the conflicting interests were consolidated in a new corporation, known as the New Jersey Zinc Co. That company now controls all the mines in Sussex County, N. J., and in the Saucon Valley, Penn.

Geology of the Ore Deposits.—Both at Stirling Hill and Mine Hill, the former being two miles, south 20° west, from the latter, the ore bodies occur

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[1] Eng. and Min. Journ., July 28, 1900.

as beds lying between strata of white limestone of Cambrian, Cambro-Silurian, or Archæan age, which have been contorted in a remarkable manner. The Stirling Hill bed, or vein, as it is commonly referred to, strikes southwest for 1,100 ft., then curves around for 300 ft. and strikes northeast, parallel with its own extension, for about 475 ft., until it ends. Both parts of the vein dip about 60° east.

The vein at Mine Hill presents analogous, but even more complicated features. It strikes south 30° west for about 2,500 ft., then bends around in a sharp fold and strikes to the east at an angle of about 30° with its other extension, and after running approximately 600 ft. pitches below the surface at an angle of 27° or 28°, though it has been proved by borings and a shaft to extend about 2,000 ft. further, where its depth below the surface is about 1,000 ft. In the western vein the ore dips southeastward at angles varying between 37° and 60°; in the eastern vein it is nearly vertical.

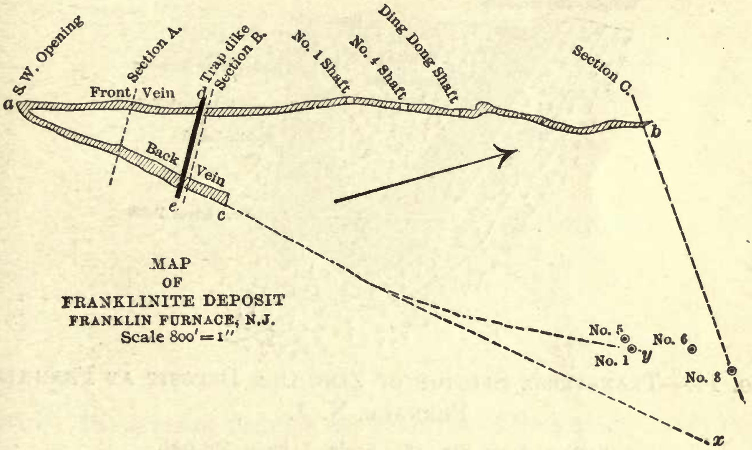

FIG. 13.

The outcrop of the Stirling Hill vein plotted on the map resembles a hook in its shape. That of the Mine Hill vein resembles a wire bent sharply in two legs at an angle of 30°, one leg being short but of equal length to the other if its known extension underground be plotted. In transverse sections both deposits show as two veins dipping more or less in the same direction and one apparently underlying the other, wherefore they are frequently referred to as the "front" and "back" veins.

The Stirling Hill veins vary from 4 to 20 ft. in thickness, throughout which the zinc minerals are disseminated, but not uniformly. In the front vein the portion near the foot wall shows a band richer in zinkite and willemite than that near the hanging, but it is hardly enough to justify speaking of two veins, a zinkite vein and a franklinite vein, as has been done.[1] Both are irregular and pinch because of the coming in of the foot wall. The ore body is an impregnation of the limestone along this horizon with the ore bearing minerals in a greater or less degree and with a streak richer in zinc next the hanging. As much as 20 ft. in thickness has been taken out up to the limits where the walls became too lean to work.

The Mine Hill vein is as much as 50 ft. thick in places and even more at the bend. Frank L. Nason, in a paper in *Trans. Am. Inst. Min. Eng.*, February, 1894, which goes extensively into the geology of the deposit, considers that the ore body is equivalent to a prism 3,500 ft. long, 800 ft. wide and 25 ft. thick. It is probably the largest single deposit of zinc ore ever found in the United States and one of the largest known in the world.

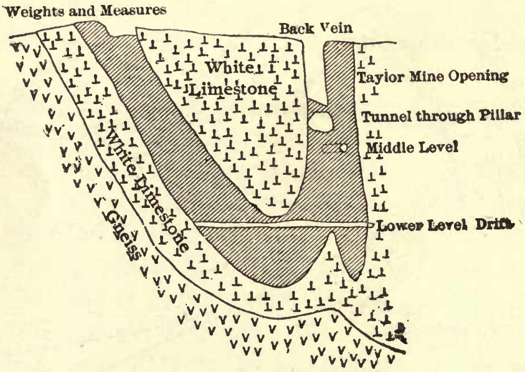

FIG. 14.—TRANSVERSE SECTION OF ZINC ORE DEPOSIT AT FRANKLIN FURNACE, N. J.

Section A on Fig. 13. Scale, 1 inch=200 feet.

Character of the Ore.—The zinc ore of New Jersey is essentially a mixture of zinkite, willemite and franklinite, forming a rock-like mass (according to F. L. Clerc, whom I quote because of the aptness of his description) not unlike in appearance an eruption of granite, in which the franklinite represents the quartz, the willemite the feldspar and the zinkite the mica. The analogy would be more exact if the ore were compared to a garnetiferous syenite, in which the greenish orthoclase might stand for the willemite, black hornblende for the franklinite and the red garnet for the

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[1] J. F. Kemp. The Ore Deposits at Franklin Furnace and Ogdensburgh, N. J., in Trans. New York Academy of Science, XIII. 76 to 98. Reference should be made to this paper for a more complete knowledge of the highly interesting geology of these New Jersey mines.

zinkite, but neither any granite nor the imaginary syenite would have the distinctly crystalline appearance and brilliancy of the New Jersey zinc ore. Besides the franklinite, willemite and calcite there are small quantities of zinkite, tephroite, garnet, and occasionally other minerals. The mineralogical composition of the ore is approximately as follows: franklinite, 51·92%; willemite, 31·58%; calcite, 12·67%; zinkite, 0·52%; tephroite and other silicates, 3·31%; total, 100%. Its chemical composition is ZnO, 29·35% (Zn, 23·58%; O, 5·77%); Fe_2O_3, 32·06%; MnO, 11·06%; $CaCO_3$, 12·67%; SiO_2, etc., 14·57%; total, 99·71%. Owing to the low tenor in zinc and the high percentages of iron and manganese, the ore was not used extensively for the manufacture of spelter until recently, but was employed chiefly for making zinc white. The development of special processes of mechanical separation of the minerals, which are described in Chapter XI, has made the ore a source of high grade, remarkably pure zinc mineral.

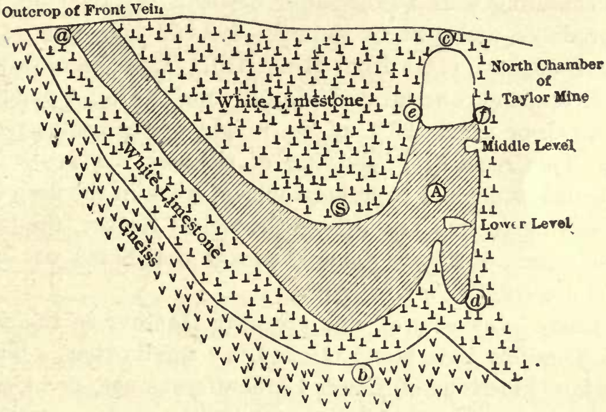

FIG. 15.—TRANSVERSE SECTION OF ZINC ORE DEPOSIT AT FRANKLIN FURNACE, N. J.

Section B on Fig. 13. Scale, 1 inch=200 feet.

Exploitation of the Mines.—At the time of the visit of the American Institute of Mining Engineers to Franklin Furnace, N. J., Feb. 24, 1899, it was stated that the great ore body at that place (Mine Hill) was then known to be over 3,500 ft. long and in places over 200 ft. thick. At the south end of the outcrop the ore was to be worked by an open cut, and there the overlying limestone was being removed. The ore then mined came from a shaft 950 ft. deep, sunk in the hanging wall. The ore body had been opened by a number of drifts and raises, but there were no stopes, nor had any definite system of mining been laid out, as the shape of the ore body in the deep levels was still undetermined. Considerable surface water gets into the mine through fissures in the limestone, one large stream having been cut at 586 ft. in sinking the shaft. All the water collecting at the bottom is sent to surface in one lift by a duplex, triple-expansion Worthington steam pump, which has a capacity of nearly 1,000 gals. per minute. The output of the mine in February, 1899, was about 400 tons of ore daily, but before the completion of the new mill then under construction there was expected to be a productive capacity for 1,000 tons per diem.

Operations at Stirling Hill have now (1901) been discontinued, it having been found that the ore of the mine there, although abundant, is leaner than that of Mine Hill and that greater economy can be secured for the present by confining work to the latter deposit. In 1900 the New Jersey Zinc Co. produced and sent to its mills 198,262 long tons of ore, all of which, save 1859 tons, came from Mine Hill. There were shipped 37,622 tons of crude ore, and concentrated 160,640 tons, the latter yielding 148,917 tons of mineral, or nearly 93% of the weight of the ore as received from the mines. This remarkable showing of richness was made from ore of which the bulk was derived from development work in the deep levels of the mine. According to the New Jersey Geological Survey, the total production of these mines, which are the only ones in that State, was 191,220 long tons in 1901 against 194,881 in 1900.

NEW MEXICO.—Deposits of zinc ore near Hanover in the southwestern part of this Territory have been exploited to a small extent. They occur in gray and white limestone of Lower Carboniferous age, or older. The deposits are of two classes, calamine and blende, and according to W. P. Blake,[1] are rather unique in some respects. Apparently the deposits of calamine and blende have no direct connection. The former have been quarried out from the surface downward in irregular pits and cave-like excavations, in some cases to a depth of 60 ft. or more, gradually thinning out to mere seams. The ore is chiefly smithsonite, often in close association with aggregations of small crystals of quartz. The best ore as shipped assays 35 to 38% Zn. Blende also occurs in masses in the limestone, but although they may be oxidized superficially, these deposits have no connection with those previously described. The blende is of the dark, reddish-brown variety, free from arsenic and antimony, but commonly intermingled with pyrites; galena is entirely absent.

The masses of blende ore generally occur at the contact of the limestone with dikes of igneous rocks, from which the ore is separated by a sheet of hard, tough rock, known locally as "green rock," which consists of fibrous

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[1] Trans. Am. Inst. Min. Eng. XXIV, 187.

amphibole, together with garnet and probably epidote, and in places carries hematite, with blende and pyrite more or less intermingled. The form of the deposits of blende is generally lenticular, but being mostly, if not in all cases, along the planes of contact of intrusive dikes, or following the plane of faults, they have such linear extension and sequence as to present the general appearance of lodes. Actinolite and garnet occur with the mineral elsewhere than near the dikes and Blake considers them to be of contemporaneous origin with the ore. The garnet is of a wax-yellow color, the variety grossularite, and somewhat resembles resinous blende in its appearance. Owing to the nearness of its specific gravity to that of blende it is with difficulty separated from the latter by gravity concentration.

Shipments of zinc ore are made from the Hanover district to North Chicago, Ill., and Mineral Point, Wis., but the high cost of transportation thither has heretofore prevented any extensive exploitation of the mines.

PENNSYLVANIA.—Zinc ore has been mined at several places in this State, especially at Friedensville, Lehigh County; in Sinking Valley, near Birmingham in Blair County; near Landisville in Lancaster County; on the Susquehanna River a few miles below Sunbury in Northumberland County, and on Pickering Creek a few miles south of Phenixville in Chester County. Of these the Friedensville deposits were the most important and are of interest now, since they are likely to be reopened.

Friedensville.—The Friedensville mines are situated in the Saucon Valley, a few miles south of Bethlehem. According to H. S. Drinker,[1] zinc ore was first discovered there in 1845, and mining on an extensive scale was begun in 1853 by the Pennsylvania & Lehigh Zinc Co., which at the same time erected smelting furnaces and a zinc oxide plant at Bethlehem.[2] Three mines, known as the Ueberoth, Hartman and Saucon were opened; they are situated within half a mile of each other. The ore occurs in a stratum of magnesian limestone of Lower Silurian age, which strikes northeast and southwest.

The character of the deposits has been somewhat differently described by Henry D. Rogers,[3] Frederick Prime,[4] F. L. Clerc,[5] and H. S. Drinker.[6] A comparison of their accounts indicates that there are three channels of ore, on which the three mines named were respectively opened, formed by replacement of a part of the limestone, or of parallel strata of it, with connecting cross-seams. In each mine the ore shoot pitches to the south-

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[1] Trans. Am. Inst. Min. Eng., I, 67.

[2] J. D. Whitney, Metallic Wealth of the United States, edition of 1854, p. 351.

[3] Geology of Pennsylvania, two volumes, published in 1868.

[4] Report of the Second Geological Survey of Pennsylvania, D3, vol. I.

[5] Mineral Resources of the United States, 1882, p. 361.

[6] Loc. cit.

west. In the Ueberoth mine the limestone strata are much disturbed and stand nearly vertical; in the Hartman mine, distant about half a mile from the Ueberoth, the strata are less disturbed and the dip is less steep; in the Saucon mine, distant two furlongs from the Hartmann, the regularity of the strata is still more pronounced and their dip more gentle.

In the Ueberoth mine there were six parallel seams of ore interlaminated with limestone strata and connected by six perpendicular seams. At the intersections there were found enlargements of the ore body, some of these bulges being 20 ft. thick and 60 ft. wide. These seams were followed for more than 1,000 ft. on their strike to a depth of 225 ft. vertically, or 250 ft. on the dip, at which depth the operation of the mine was discontinued with no signs of failure. In the Hartmann and Saucon mines the ore shoots were more compact and lenticular in form. That of the Saucon mine was remarkably regular in pitch, course and width. It pitched to the south at an angle of about 30°, was 60 ft. in width and had a thickness of 30 ft. The Hartmann mine was worked to a depth of 150 ft. The Saucon ore shoot was followed 250 ft. by the Bergenpoint Zinc Co., gaining a vertical depth of 110 ft., and was followed 150 ft. further by the Lehigh Zinc Co., into whose property it had passed, a total depth of 200 ft. being reached.

The Ueberoth mine was worked continuously from 1853 to the autumn of 1876, when it was closed on account of the great quantity of water which had to be pumped. Even at the depth of 40 ft. there was a great flow of water; at 150 ft. depth the then famous pump, known as the "President," was installed. This was a Cornish pump, with a 110 in. single cylinder, double acting, condensing, walking-beam engine, 10 ft. stroke, designed to work four 30-in. plunger pumps and four 30-in. lift pumps, with 10 ft. stroke, and to take water from a depth of 300 ft. When work was stopped it was running six to seven strokes per minute, and operating three pairs of 30-in. pumps and one pair of 22-in. and was easily handling the water that came to them.

The Hartmann and Saucon mines were wet, but the flow of water in them was less than in the Ueberoth, and they continued to be operated for some time after the Ueberoth was closed. They were being worked on a small scale in 1882. Up to the end of 1876 the Ueberoth mine is said to have produced about 300,000 tons of ore, of which one third came from a great body opened close to the surface.

Character of the Ore.—The ore of the Saucon Valley was calamine (smithsonite and hemimorphite) near the surface, changing to blende with depth. The oxidation was deepest in the Ueberoth mine and least in the Saucon. The blende is of the cryptocrystalline variety, with a conchoidal fracture and bluish slate color. It is remarkably free from lead, arsenic and antimony, wherefore a high grade of spelter was made from it, but it is somewhat mixed with pyrite, which, together with its cryptocrystalline character, makes it difficult to concentrate. According to Clerc (loc. cit.) it assayed from 35 to 40% Zn as sent to the works, and since he describes it as resembling broken limestone, I infer those figures represent a cobbed and sorted product.

TENNESSEE.—Zinc ore has been mined at several places in upper East Tennessee, where deposits of the mineral have been found in nearly every county of the region, though but few of them appear to be of any economic importance. The most work has been done at Mossy Creek, New Market, Straight Creek and Lead Mine Bend. In 1900 a deposit of carbonate ore was opened near Mascot on the line of the Southern Railway, and an occurrence of blende was exploited near McMillin Station, in Knox County. There are two forms of deposits of zinc ore in Tennessee.

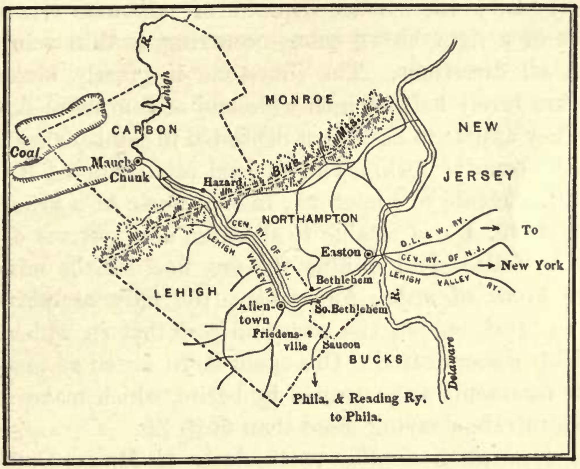

FIG. 16.—MAP OF NORTHAMPTON, LEHIGH AND CARBON COUNTIES, PENNSYLVANIA.

(SCALE 1 IN. = 24 MILES.)

Showing location of zinc mines and smelteries and the eastern extension of the anthracite coal field.

Mossy Creek and New Market.—One type occurs in the so-called Knox dolomite, which is of Lower Silurian age, at Mossy Creek, on the main line of the Southern Railway from Knoxville to Asheville, the zinc bearing strata extending in a southwestward direction from Mossy Creek through

New Market to within six miles from Knoxville. Masses of smithsonite were found originally in this area, lying on a jagged surface of limestone and surrounded by, or covered by, a bed of red clay. The ore was of great purity, like that of the Virginia deposits, which are not a great distance away, and in some respects its occurrence was similar, although where exploited the overlying bed of clay was not so deep nor has the corroded surface of the limestone the pinnacle structure of the Bertha mines. Calamine deposits of this character were worked at Mossy Creek by the Edes, Mixter & Heald Zinc Co. and the Bertha Zinc and Mineral Co., and at New Market by the Ingalls Zinc Co.

Immediately under the surface deposits of carbonate ore the limestone carries blende of a light brown color, occurring as thin veinlets or seams ramifying in all directions. The limestone is greatly broken up. The seams of ore are rarely half an inch wide and seldom more than a few feet in length. They appear to have been deposited in joint planes and irregular fractures in the country rock. The mineral is distributed in this manner through a thick stratum of limestone; in the course of a brief examination in November, 1899, I was unable to discover any systems of enrichment which might be followed in mining the ore, nor did the miners who had worked there know of any. Apparently, the only practical method of mining was to break out all the rock and sort that in which the mineral was most thickly disseminated. Ore could be so sorted as to assay 8% Zn. The blende is commonly accompanied by barite, which makes it difficult to produce a concentrate assaying more than 50% Zn.

The above description applies particularly to Mossy Creek. At New Market the occurrence is similar, but the ore bearing stratum of limestone is said to be only 50 ft. wide, which is less than at Mossy Creek. The following analyses of ore from New Market were made at the University of Tennessee: smithsonite—$ZnCO_3$, 83·29%; SiO_2, 13·10%; $FeCO_3$, 1·32%; Fe_2O_3, 0·66%; $CaCO_3$, 1·07%; $MgCO_3$, 0·56%; blende—ZnS, 84·75%; SiO_2, 7·57%; $Fe_2O_3+Al_2O_3$, 1·66%; $CaCO_3$, 5·98%; $MgCO_3$, 0·04%.

The mines at Mossy Creek formerly owned by the Edes, Mixter & Heald Zinc Co. have been more recently exploited by the John Weir Lead & Zinc Co.; those at New Market are idle; other deposits in the district are worked in a desultory manner by individuals, who sell the ore to the Bertha Zinc & Mineral Co. or to smelters in Indiana.

Straight Creek.—A mine was also worked by the Edes, Mixter & Heald Zinc Co. and later by the John Weir Lead & Zinc Co. at Straight Creek in Claiborne County, about three miles west of Lone Mountain Station, on the Knoxville & Cumberland Railway. At this place there is a limestone formation striking north 70° east, and dipping south 20° east at a slope of 35° at the point where the ore outcropped; in going east the dip increases to 60°. A shoot of ore pitching east and partly replacing a stratum of the limestone outcropped on the side of a hill and has been followed several hundred feet, changing from calamine to blende with depth. The ore bulges and pinches in lenticular form, sometimes attaining a width of 20 ft. and a height of 30 ft., but is generally of less dimensions. Near the ore body caverns of irregular shape are frequently found in the limestone. The carbonate ore has been entirely mined out of this shoot. The sulphide ore body consists of small streaks or lenticles of blende interlaminated with limestone. After breaking it can be readily separated as lump ore assaying 40% Zn. The blende is of the grayish brown, cryptocrystalline variety. It is quite free from pyrites, but some galena is intermixed with it.

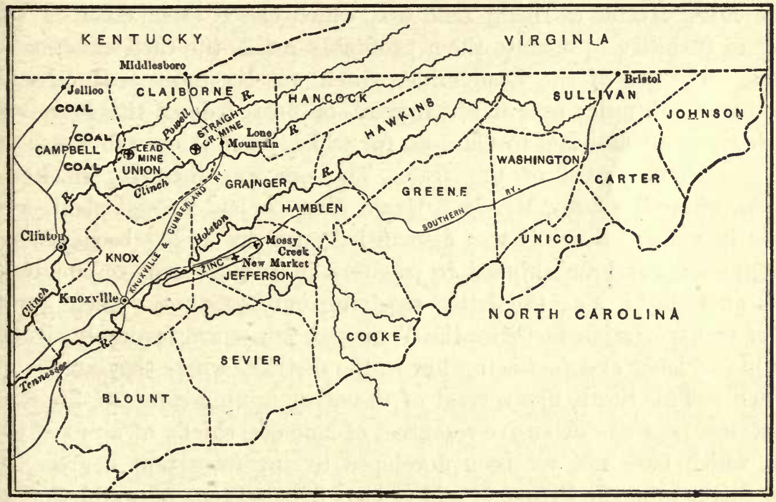

FIG. 17.—MAP OF EASTERN TENNESSEE.

Showing location of zinc mines.

Lead Mine Bend.—About 16 miles west of the Straight Creek mine, within a quarter of a mile of the Powell River, is the Lead Mine Bend property, which occurs in a similar limestone formation, but the beds there lie more flatly. This mine was originally worked for lead, as its name implies, and has been extensively developed. The mineral occurs in a channel made up of bunches and lenticles of galena and blende interlaminated with limestone and following a crevice which breaks longitudinally an anticlinal fold in the limestone. The axis of the anticline dips 8° east. The crevice, which is nearly vertical, shows a nearly continuous streak of ore for a longitudinal distance of several hundred feet and a depth of 40 ft.; perhaps more. The ore extends out into the limestone 50 to 100 ft. on each side of the crevice, and the strata appear to be mineralized for about 20 ft. perpendicularly to their dip. The mineralization is irregular, however, and generally a large proportion of barren rock must be broken to get the ore. The latter can be hand-sorted up to about 40% Zn. The blende is grayish brown and cryptocrystalline, free from pyrites, but intermixed with galena.

Mining Conditions.—Both the Straight Creek and Lead mines belonged to the Edes, Mixter & Heald Zinc Co., which closed them down in 1893, owing to inability to operate them profitably under the then existing conditions. The ore of the Lead mine, which was the most productive, has to be carted 20 miles over a rough road, or floated down the Powell and Clinch rivers in flatboats to Clinton, for which there is enough water only during a short period of the year. The ore was formerly smelted at Clinton, where the Edes, Mixter & Heald Zinc Co. had a small plant, which is now in ruins. Since it was dismantled such ore as has been produced in Tennessee has been shipped to smelters in Indiana and to the Bertha works at Pulaski, Va., the latter receiving only calamine. Save for the lack of transportation facilities the Tennessee mines are favorably situated, the costs of labor and fuel being low in the districts where they are.

UTAH.—This State, like several of the other mining states of the Rocky Mountains, possesses extensive resources of zinc ore, chiefly of a mixed character, which have not yet been developed to any important degree. The Horn Silver mine, at Frisco, is said to have 300,000 tons of ore blocked out, which will average 5 to 8% Pb, 3 to 7% Fe, and about 35% Zn (the range in zinc being from 25 to 40%); it is said to contain about 0·06 oz. Au and 7 oz. Ag. The gangue is silicious. The great pyrites deposits of Bingham Cañon contain considerable blende in some portions. A good deal of blende occurs in the ore raised from the mines at Park City. As compared with Colorado, Utah is at a disadvantage with respect to the markets for zinc ore, which is likely to retard the development of its resources, because of the higher cost of transportation.

VIRGINIA.—The zinc mines of this State are situated in the southeast corner of Wythe County, in the "Valley of Virginia," so-called. Three mines have been worked there, namely, those of the Bertha Zinc & Mineral Co., those of Manning & Squier adjoining the Bertha on the northeast, and those of the Wythe Lead & Zinc Co. at Austinville, eight miles to the southwest. Other mines have been opened at Ivanhoe, a little beyond Austinville. These mines are included within an area 10 miles long in a direct line bearing north 55° east, on the south side of New River, which flows northeastward in a tortuous course. The greatest distance of any of the mines from the river is a little less than two miles. The most important mines are those of the Bertha company. They were discovered in 1866, but were not worked extensively until 1879.

The Bertha mines, situated about 20 miles southwest of Pulaski, occur in magnesian limestone of the Lower Silurian formation, which dips between 6° and 7° toward the river and carries the zinc bearing strata below the latter at a depth of several hundred feet. The outcrop of this limestone is covered with a heavy bed of clay, beneath which, resting in hollows in the limestone, the zinc ore is found. Apparently the outcrop of the limestone has weathered with extreme irregularity, so that if the overlying clay and ores were entirely removed there would be presented a wilderness of limestone pinnacles of varying heights, up to 100 ft. Rarely the limestone shows small caves in the form of clefts and crevices. The deposits of zinc ore invariably rest against the sides of these pinnacles and in the hollows between them. Sometimes they completely cover a pinnacle, particularly the lower ones, but in general they cover them only partially. The occurrence of the ore is irregular and frequently pinnacles and the hollows between them show none at all. The ore is smithsonite and hemimorphite, chiefly the latter, in a clay gangue. The deposits vary in thickness from a few inches up to 40 ft., the latter having been found between chimneys (pinnacles). On the sides of the pinnacles thicknesses of 5 to 10 ft. occur frequently, but the average is less than 5 ft.

Character of the Ore.—The ore consists of hard and soft varieties, the former occurring through the mass in all sizes from small grains up to blocks weighing several tons. Rarely the hard zinc ore clings to the limestone in sheet form. The ore bodies are entirely distinct from the overlying clay and underlying limestone. The matrix, or gangue in which the ore occurs, is a soft unctuous clay, both hard and soft, the former being known as "hard buckfat," and the latter as "soft buckfat." The latter dissolves in water when violently agitated; the former is insoluble, but having a fine grained, brittle structure, like common chalk, and a lower specific gravity than the hard ore, can be separated by jigging.

The ore raised from the mines contains about 26% Zn. It is dressed by sluicing, which washes out a good deal of soluble clay, and by jigging, so as to yield about 33% of concentrate, which assays 38·08% Zn, 29·37%

SiO_2, 9·23% Fe_2O_3 and Al_2O_3, 4·54% $CaCO_3$, 2·07% $MgCO_3$ and 8·23% combined water. This ore yields a spelter of exceptional purity.

Blende is found in irregular deposits through a depth of a hundred feet or more of the limestone underlying the calamine, and in following the limestone on its dip it is expected that larger deposits will be found.

Method of Mining.—The Bertha zinc deposits and the method of working them have been described by William H. Case in *Trans. Am. Inst. Min. Eng.*, XXII, 511 to 536, and by E. C. Moxham in *The Engineering and Mining Journal,* 1893, LVI, No. 22. Since those publications, however, the system of exploitation has been changed radically.

The thickness of the clay cover varies from 10 ft. to 150 ft. In the early days of the mine the method of working was simply to remove and waste this clay covering and then mine the ore by open-cut methods. Of late years, owing to the increasing thickness of this overburden, underground methods of mining have been used exclusively and the open-cut work abandoned; these underground methods consisted of sinking shafts through the clay and zinc ore to the bed-rock and then removing the zinc ore by drifts and galleries, hoisting the ore through shafts, and of course timbering every foot of the ground to support the overburden. In still more recent times two important and determinative circumstances arose which led to another change in the method of mining: (1) the increasing expense of winning the zinc ore by the underground methods described above made them quite unsatisfactory; and (2) the clay overburden was discovered in a greater part of the mineral zone to be capable of yielding a high-class limonite ore, which was valuable for its iron content.

In 1898, therefore, the mining of zinc ore on the property was abandoned temporarily and the work of stripping and washing, by the ordinary methods of the district, all of this overlying iron ore bearing material was begun. In 1900 such progress had been made that about 1,300 tons of clay were being removed and washed per day, yielding about 160 tons of limonite which is shipped to the blast furnace at Pulaski, the idea being to attack again the deeper zinc ore by open cuts whenever and wherever the cover of clay and iron shall have been sufficiently removed. For this reason the zinc output of the Bertha Zinc & Mineral Co. has been temporarily reduced.

Clark Mine.—Some developments have been made at the Clark mine, across the New River from Allisonia, Pulaski County, Va., where calamine and blende have been found over a considerable area. The country rock is limestone interstratified with dolomite, in which large beds of mineralized rock are said to occur. A test of a carload in 1898 showed that 3·12 tons of rock would yield one ton of concentrate assaying 48% Zn and 16% Pb.

Another test of 10 tons of rock gave 2·45 tons of concentrate assaying 40·52% Zn, 16·05% Pb, 2·75% Fe, 20·54 S, and 1·8 oz. silver per 2,000 lb.[1]

WISCONSIN AND IOWA.—The mines of these States, where lead occurs in association with zinc, were first worked for the former metal a little previous to 1800; the mining of zinc ore did not begin until 1860, at which time the establishment of the Matthiessen & Hegeler zinc works at Lasalle, Ill., began to furnish a near-by market for that kind of ore. The area of the Wisconsin-Iowa-Illinois mining region, which for convenience will be referred to as the "Wisconsin region," including under that expression its extensions into Iowa and Illinois, embraces about 2,600 square miles, its length in an east and west direction being about 65 miles, and its breadth north and south about 55 miles, the area being more or less elliptical in plan. About five-sixths lie in Wisconsin and one sixth covers the contiguous corners of Iowa and Illinois. The altitude of the region is between 580 and 1,700 ft. above sea level, its surface aspect being that of a gently undulating plain, with conical flat-top mounds rising 100 to 400 ft. above the general level.

Geology.—The geological formation comprises sandstones, limestones and shales of the Cambrian and Lower Silurian systems, the ore being found in the Galena dolomite, 250 to 275 ft. thick, the blue limestone, 50 to 75 ft. thick, immediately underlying, and the buff dolomite, 15 to 20 ft. thick, which next underlies the blue limestone, the three formations belonging to the Lower Silurian. The Galena limestone is the most important ore carrier.

Geographically, lead ores are distributed over the whole region, but especially in the southern portion, in the vicinity of Galena, Ill., and Dubuque, Ia., while zinc ore is more abundant in the northeast, at and near Mineral Point, Mifflin, Linden and Dodgeville, where the lower beds of the ore bearing horizon are more exposed. It is a generally observed rule in this region that the zinc ore occurs at greater depth than the lead ore, even when both are found in the same crevice, and it is considered possible by the geologists who have studied the region that deeper mining would reveal zinc ore in the southern portion. Besides the places named above, Platteville, Shullsburg, Benton, Hazel Green and Potosi are important mining centers.

The ore first mined in the Wisconsin region was found chiefly in residuary clays near the surface, but since then it has been obtained from vertical crevices or flat sheets in the country rock. The former characterize especially the Galena limestone and are preëminently lead bearing. They rarely

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[1] The Mineral Industry, VII, 729.

exceed 100 ft. in depth, but are often several hundred feet in length. They sometimes exist as fine seams filled solidly with galena, and elsewhere expand to large caves, 50 or 100 ft. long and as much as 30 ft. wide. Flat sheets are most abundant in the Trenton limestone, and like the vertical crevices may be filled with solid masses of galena or blende, but more often the mineral is intermixed with a gangue of calcite and sometimes barite. A modification of the flat deposits are the so-called "flats and pitches," in which ore extends downward, generally on two sides, from a central flat sheet in a series of steps, which are alternately transverse to and parallel with the strata. The largest mines are in deposits of this class.

Kinds of Ore.—The zinc ores of Wisconsin comprise blende, smithsonite and hemimorphite, with which are associated galena and its derivatives, pyrite, marcasite, chalcopyrite, calcite, dolomite and barite. The blende is characteristically of a black color and is consequently apt to be rather high in isomorphous FeS. Calcite is more common than dolomite and barite in the gangue, and as oxidation product of the blende, smithsonite is a good deal more common than hemimorphite. According to Winslow, in the report of the Missouri Geological Survey on *Lead and Zinc*, I, 148, the production of zinc ore in Wisconsin from 1860 to 1876, both years inclusive, was 69,467 tons of smithsonite and 57,728 tons of blende. Since 1890, the production of all kinds of ore appears to have been from 15,000 to 20,000 tons per annum.

Mining Conditions.—The methods of mining in vogue in Wisconsin are crude, nine tenths of the mines being worked by horse and man power. Until a few years ago there was no thought of mining below what was then the water level, but during the last 30 to 40 years the water has fallen 10 to 15 ft. and has permitted shafts to be sunk that additional distance. It is between the old and the new water levels that most of the blende has been mined. Some of the ore mined in Wisconsin is jigged, but a good deal is concentrated by hand sorting only and is marketed in lump form. Out of 12,500 tons of ore sold in nine years from one mine at Benton 77% was sorted, lump ore. A good deal of low grade ore is said to have been left in some of the mines because of lack of machinery for handling it. The Wisconsin region is considered by many persons to have a capacity for a larger output of zinc ore than at present is made if more attention were devoted to it, but the presence of a considerable percentage of marcasite mixed with the blende makes the production of a high grade concentrate difficult in many cases, and the larger deposits and more docile ores of the Joplin district make the latter more attractive to miners.

Occurrence of Zinc Ore in Canada and Mexico.

Canada.—Deposits of zinc ore on the north shore of Lake Superior attracted attention in 1898, when the Grand Calumet Mining Co. began the exploitation of a vein of blende near Rossport in the Algoma district. The investment of capital in that enterprise stimulated prospectors to a more diligent search for new deposits, with encouraging results according to the report of the Bureau of Mines of Ontario for 1900. The Zenith mine, owned by the Grand Calumet Mining Co., is situated about 12 miles north of Rossport, a small town on the line of the Canadian Pacific Railway, the road to the mine leaving the railway five miles east of Rossport. The country is extremely rough and access to the mine, which is about 1,200 ft. in elevation above the level of the railway, is difficult, and the road, crossing twelve small lakes, shipments can be made under present conditions only in winter, when the lakes are frozen over.

The Zenith mine is opened on a lode intersecting a trap formation. This lode, which stands nearly vertical, is continuous on the surface for a considerable distance, but although it presents a good showing of ore at some places, the ore is said not to extend to much depth, so far as has been yet discovered. The ore is blende, dark in color, occurring mixed with country rock, which is separated by cobbing, whereby a product assaying about 50% Zn is produced. The cost of transportation from the mine to the railway is about $2 per 2,000 lb; according to the report of the Ontario Bureau of Mines for 1900, it is not probable that the cost of carting will ever be reduced unless very great quantities of ore be discovered in the district, since the cost of building any kind of a railroad through a country of such difficult topography would be prohibitory. Some prospecting has been done on properties near the Zenith mine, where apparently similar occurrences of ore have been discovered. The total production of zinc ore in Ontario in 1899 was reported by the Ontario Bureau of Mines as amounting to 1,200 tons of 2,000 lb., valued at $24,000, all of which was derived from the Zenith mine. At the end of 1900 the company operating that mine had, according to *The Engineering and Mining Journal,* of Jan. 12, 1901, about 2,000 tons of 50% ore ready for shipment, which probably represented the production of 1900.

Mexico.—Zinc blende occurs at numerous places in this Republic, but many of the deposits are so remote that they are unworkable under present conditions. A few are more favorably situated. Near Monterey there is said to be a mine of high grade blende ore, nearly free from lead, which assays 3 to 8 oz. silver per ton. At Charcos, San Luis Potosi, there are galena-blende ores assaying 12 to 40% Zn and 12 to 60 oz. silver per ton. This is an old mining district, and a largely worked one, where the blende was avoided or sorted out and thrown on the waste dumps in the previous exploitations. Similar accumulations of rejected zinc ore are to be found elsewhere in Mexico. At Matehuala and Maroma, in San Luis Potosi, there is blende associated with silver, lead and copper minerals. Other zinc bearing districts exist in the states of Mexico, Puebla, Hidalgo and Tamaulipas, but zinc being disliked in the treatment of ores for copper, lead and the precious metals, little is known as to their capacity or chances for development. In 1896 the cost of transportation from Monterey to Liverpool or Swansea was $15, Mexican currency, per metric ton. Charcos and Maroma have good railway communication, and the cost from those places would probably be only a little more than from Monterey. The Helena Mining Co. is said to have at Cusihuiriachic, Chihuahua, large bodies of ore, which average 25% Pb, 30% Zn, and a considerable tenor of silver. A sample of this ore assayed 26·4% Pb, 35·5% Zn, 5·6% Fe, 1·8% SiO_2 and 7·5 oz. Ag per ton.

X.

OCCURRENCES OF ZINC ORE IN EUROPE, AFRICA AND AUSTRALIA.

Deposits of zinc ore are more numerous in Europe than in America, if the at present largely non-available resources of the Rocky Mountains be left out of account, and their aggregate production is larger. Outside of Europe there is an important production in Algeria and Tunis, while lately a considerable supply of ore has been obtained from Broken Hill, New South Wales, where the blende which occurs in great quantity intimately mixed with galena has been successfully separated under favorable market conditions. The zinc resources of Europe are probably now well known, with the exception of parts of Russia which are too remote for profitable exploitation at this time. Those of Asia, Australasia, Africa and South America have hardly been explored. The most important zinc producing countries of Europe are Germany, Italy, France, Spain and Sweden, the statistics of which countries, together with others, have been presented in Chapter IV. It should be remarked that the great preponderance of Germany in the statistics is due largely to the fact that the ore of Upper Silesia is smelted in a comparatively low grade form, while that of other countries is mostly concentrated.

ZINC ORE DEPOSITS OF EUROPE.

The geological occurrence of some of the important deposits of zinc ore in Europe is summarized in the following table, which is admittedly very incomplete, but is of some interest in showing the general tendency of zinc ore deposits to form in a country rock of limestone or dolomite. The same phenomenon is manifested in the case of the principal non-argentiferous deposits of the United States, which were summarized in tabular form on page 179. It will be observed in comparing the two statements that the zinc ore deposits of Europe occur through a wider geological horizon than do those of America, being found in rocks from the Laurentian age up to those of the Cretaceous, whereas the American deposits are chiefly in Lower Carboniferous and Silurian formations.

Country.	District.	Character of Mineral.	Country Rock.	Age.
Belgium	Moresnet.	Calamine.	Dolomite.	Carboniferous.
France	Les Malines.	Calamine and blende.	Dolomite.	Jurassic.
Germany	Upper Harz.	Blende.	Slate.	Devonian and Lower Carboniferous.
"	Gladbach.	Calamine.	Dolomite and shale.	Devonian.
"	Iserlohn.	Calamine and blende.	Limestone and shale.	Devonian.
"	Upper Silesia.	Calamine and blende.	Dolomite.	Triassic.
Great Britain ..	Cumberland.	Blende.	Limestone.	Lower Carboniferous.
" ..	Shropshire.	Blende.	Slate and shale.	Cambrian-Silurian.
Greece	Laurium.	Calamine and blende.	Limestone and mica schist.	Silurian.
Italy	Iglesias.	Calamine.	Limestone and schist.	Silurian.
Russia	Poland.	Calamine and blende.	Dolomite.	Triassic.
Spain	Cartagena.	Blende.	Schist.	
"	Reocin.	Calamine and blende	Dolomite.	Cretaceous and Jurassic.
Sweden	Ammeberg.	Blende.	Gneiss.	Laurentian.

AUSTRIA.—The principal zinc mines of Austria are situated in Southern Carinthia, Styria and the Tyrol. The Carinthian deposits are distributed in a belt a few miles wide and about 100 miles long in an east and west direction. The chief mines are at Bleiberg, Kreuth, Raibl, Windisch-Bleiberg, Kappel, Miess and Schwarzenberg. The country rock is limestone and dolomite of Triassic age in which are found bodies of ore of irregular form, their axes appearing to follow the intersections of stratification planes and fault fissures. Such pipe-shaped deposits have been as much as 1,200 ft. long and from 6 to 90 ft. in diameter. Sometimes they spread out on the stratification planes of the country rock, forming flat lenses as much as 20 ft. in thickness. The ore is chiefly galena, but blende occurs with it, and in places bodies of smithsonite are found.

In the Tyrol zinc is mined at Schneeberg, where the mines which had been worked for many centuries and exhausted of their lead ore were reopened for zinc in 1866. The mines are opened on lodes 7 to 56 ft. thick, which have been followed on their strike a distance of 1·5 miles and to a depth of 3,000 ft. The country-rock is mica schist and the ore is blende and galena, with some pyrite intermixed.

Belgium and Moresnet.—Belgium was formerly a large producer of zinc ore, but since 1856 its output has steadily decreased and is now comparatively unimportant. Its mines are of great historic interest, however, having been the first exploited in the world for zinc and established a great smelting industry, which now maintains its importance by the importation of foreign ores. The principal zinc deposits of Belgium occur at Bleyberg, and near Verviers and Liège in the Province of Liège. In the neutral territory of Moresnet[1] are the famous mines of the Vieille Montagne.

Moresnet.—The zinc mines of Altenberg, or Vieille Montagne, are believed to have been worked as early as the twelfth century, though the first reference to them in existing documents is 1435. Extensive mining was begun in 1846, by open cuts, and in 1856 underground. Up to 1878 the total output was 1,600,000 tons of concentrated ore, derived from 200,000,000 of crude ore. The deposit existed in a narrow synclinal basin in Carboniferous dolomite, walled in by Devonian shales standing nearly vertically. The deposit was about 1,400 ft. long, 600 to 700 ft. wide and 200 ft. deep. The ore was chiefly smithsonite and hemimorphite, replacing completely the dolomite in some places. Intercalated strata and masses of dolomite occurred, however, and one such horse divided the ore body in two parts. The mineral was generally distributed irregularly, but sometimes was found in compact masses and sometimes in layers separated by clay. Besides the smithsonite and hemimorphite, willemite occurred in masses as large as 100 cu. yds. The smithsonite was more abundant in the upper portions of the deposit and the hemimorphite in the lower, the willemite being promiscuously distributed. But very little blende or galena was found. The Vieille Montagne mine has been owned since the early part of this century by the Société Anonyme de la Vieille Montagne, which takes its name from it. The mine is still operated, but its production is no longer large.

Bleyberg.—The mine at Bleyberg, near Altenberg, which has also been a large producer, but to a far less extent than the Vieille Montagne, is opened on a fissure vein about 3 ft. thick, which traverses Lower Carboniferous limestone and Coal Measure shales, sandstones and grits, having been proved for a distance on its strike of 1·5 miles in the former and three miles in the latter. The vein stands vertically, or dips at an angle of 75° to 80°. The vein matter consists chiefly of broken country rock carrying about 18% of metalliferous minerals, principally blende and galena which exist in

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[1] This belongs neither to Belgium or Germany, between which countries it is situated. Its mineral production is not represented in the official statistics published by any government.

nearly equal proportions. At certain places large cavities have been corroded in the adjacent limestone, as much as 1,600 ft. in length and 200 ft. in width, which were lined with blende and galena, but through movements in the rock those incrustations were dislodged and buried in a mass of breccia on the floor of the caverns. The Bleyberg mine is very wet.

Welkenrodt.—At Welkenrodt, near Altenberg, a bed of zinc ore occurred between limestone and shale of Carboniferous age. Along its strike it measured about 750 ft. The lower portion of the ore body, next to the limestone (foot wall), was an earthy variety of hemimorphite passing into an ochery iron ore in the upper levels. The upper portion, next the shale (hanging wall) was a black clay containing nodules and seams of blende, galena and pyrite. The deposit dipped steeply and conformed to the bends of the enclosing rocks.

Nouvelle Montagne.—At Verviers the Nouvelle Montagne mine was opened on a large pear shape body of zinc ore surrounding a core of dolomite.

Corphalie.—At Corphalie, near Liège, beds of hemimorphite, blende and galena, from 3 to 25 ft. thick, standing nearly vertically, occurred between Lower Carboniferous limestone and Coal Measure rocks.

Philippeville.—Contact beds of galena and blende in the form of inclusions in limestone were worked at Philippeville, one impregnated stratum of dolomite being traceable for two miles.

FRANCE.—The production of zinc ore in this country which began about 1870 has increased heavily since 1885. The more part of the ore is calamine. It is mined chiefly in Southern France, in the region embraced between the Alps and the Pyrenees. The mines of the Department of Gard are the chief producers, affording nearly 50% of the total production of France. The Department of Var ranks in importance next after Gard. Inasmuch as the names of the modern departments into which France is divided convey an idea of their geographical position to only a few who are not Frenchmen, I have grouped the zinc mines of the Republic according to the ancient provinces, or governments.

Angoumois.—Deposits of mixed sulphide ore occur at Alloue and Ambernac, in the valley of the Charente, near Angoulême. A sample of the material smelted experimentally by the Ellershausen process assayed 21·1% Zn, 11·3% Pb, 6·1% FeO, 10·8% S, 27·2% SiO_2, 1·7% CaO, 4·1% BaO, 0·1% Al_2O_3, 0·07% As, 0·15% Sb, and 2·43 oz. Ag per ton.

Brittany.—The mines of Pontpéan, near Rennes, in the Department of Ille-et-Vilaine, are the most important in the Province of Brittany. They are opened on a vein, attaining sometimes a width of 25 ft. and averaging about 7 ft., which cuts vertically through Silurian schists. It has been explored to a depth of nearly 500 m. The ore is galena and blende, both argentiferous, and pyrite, occurring as seams in a gangue of quartz and clay. In 1895 the production was 119,310 tons of crude ore, which yielded 15,057 tons of galena, assaying 55% Pb, and 30,502 tons of galena-pyrite-blende.

At La Tauche, a vein in granite contains galena and blende.

Dauphiny.—Near Merglon, in the Piemart Mountains in the Department of Drome, pockets of smithsonite, which peter out with depth, are found in Middle Jurassic limestone. There is also a vertical vein, sometimes 30 ft. between walls, which is filled largely with calcite, replaced by smithsonite in places, blende and galena being of occasional occurrence.

Gascony.—Zinc ore is produced in numerous departments of the Pyrenees. In the Canton of Castillon, Ariége, the Sentein mines are opened on a lode between Lower Carboniferous limestone (foot wall) and schist (hanging); the ore is argentiferous galena, cerussite, anglesite, blende and calamine, with gangue of quartz and calcite. The mines are at elevation 6,888 ft. The lode is traceable a great distance along the ridge of the mountains. Near St. Girons there are veins carrying blende, siderite, argentiferous galena, calcite and quartz. In the Department of Haute Garonne, near Arguts, numerous thin veins of blende, with some galena, traverse Silurian slates and schists. There are deposits of zinc ore in the departments of Haute and Basse Pyrenees, which produce several thousand tons per annum.

Languedoc.—At Les Malines in the Department of Gard there are two classes of deposits. The more important occur in dolomite of Middle Jurassic age, in and along crevices, in caves, and in large masses inclosing partially unaltered pieces of the country rock. The ore is smithsonite, hemimorphite, hydrozinkite, anglesite, pyromorphite, blende and galena. The other form of deposit is a large vein traversing limestone, which has been opened on its outcrop for 1,500 ft. The vein consists of an aggregate of parallel and interlacing fissures, mineralized with blende, galena, pyrite and barite. The production of Les Malines in 1894 was 17,100 tons of calamine, 20,300 of blende and 1,000 of galena. At Clairac, near Alais, a series of fault fissure veins, from 6 in. to 3 ft. wide, leanly mineralized with blende and pockets of galena and carrying barite, traverse Lower Jurassic limestone. The Roussan group includes irregular deposits of mineralized Jurassic limestone which become poor with depth. At Les Avinieres a deposit of calamine, now exhausted, occurred impregnating a bed of dolomite between overlying marl and an underlying bed of silicious dolomite. Similar deposits, some of them containing blende, occur at different horizons in the same district. In the Department of Ardeche, which adjoins Drome, there are veins of blende and galena.

Provence.—In the Department of Var there are deposits of calamine and blende which make a considerable production. The Bormettes mine is the principal producer; its output in 1894 was 24,000 tons, all of which was blende.

Germany.—This Empire possesses the only single zinc mining district which in point of production approaches the Joplin district of the United States, namely, Upper Silesia. Besides that, however, there are important zinc mines in Rhineland and Westphalia, while considerable quantities of zinc ore are obtained from the mines of Nassau, the Harz, and elsewhere, which are also worked for both lead and zinc.

Baden.—The Grand Duchy of Baden no longer produces zinc ore, but owing to their great historic interest space may be spared for a brief reference to the once famous mines of Weisloch, in the northern part of the Duchy, which were worked for galena as early as the eleventh century. The country rock there is limestone and dolomite of the Muschelkalk formation of the Lower Triassic, in the upper beds of which smithsonite, blende, galena, marcasite and limonite, associated generally with a red clay, occurred in irregular enlargements of vertical fissures at the intersections of the latter with the stratification planes of the limestones, the ore bodies extending out along those planes in sheets, pipes and irregular masses, sometimes for distances of 2,000 ft., attaining thicknesses of 20 ft. The smithsonite was remarkable for its high tenor in cadmium, as much as 3·36% of that metal having been reported.

Hanover: Upper Harz.—A considerable quantity of blende is separated as a marketable product by the dressing works in the Upper Harz, where the mines, of which the most important are in the vicinity of Clausthal and Lautenthal, are worked chiefly for lead and silver. The ore occurs there in veins in zones of crushed slate of Devonian and Lower Carboniferous age, said zones being 65 to 262 ft. (20 to 80 m.) in width and extending longitudinally about nine miles (15 km.). The ore bodies are distributed irregularly through them. The rock of the ore zone is called vein-clay-slate (Gangthonschiefer) to distinguish it from the ordinary clay slate (Culmschiefer) of the region, but their composition is identical, the former having been derived from the latter, chiefly by mechanical alteration due to the force which produced the great faults that traverse the country rock. The Lautenthal mines are particularly productive of blende, but with depth in all the mines the proportion of that mineral increases and galena decreases.

Lower Harz: The famous ore deposit of the Rammelsberg, near Oker and Goslar, in the Lower Harz, first worked between 930 and 940 and more or less continuously ever since, occurs as a bed interstratified with Lower Devonian slates and sandstones, which have been overturned so that the original foot wall is now the hanging. The ore body, which conforms to the structure of the country rock, dips about 45°. In the direction of its length it has been exploited over 1·5 miles and in depth nearly 1,000 ft., having a general thickness of 50 ft. At its maximum development it was 1,900 ft. wide and 150 ft. thick; at the depth of 800 ft. it was 750 ft. wide and 20 ft. thick.

The ore is an intimate mixture of blende, galena, pyrite, chalcopyrite and barite, together with some calcite and quartz. It is sorted at the mine into two classes: (1) copper ore, averaging 8 to 10% Cu and (2) zinc-lead ore averaging about 25% Zn and 12% Pb. The mineralogical composition of the latter is as follows: blende, 36%; pyrite, 24%; barite, 16%; galenite, 14%; chalcopyrite, 1·5%; gangue, 8·5%; total, 100%. The copper ore is sent to the smelting works at Oker and the zinc-lead ore to the Sophienhütte at Langelsheim and the Juliushütte at Astfeld, where it is treated first for the recovery of a part of its zinc as sulphate, after which the residue is smelted in blast furnaces for lead. When the component minerals of the Rammelsberg ore occur in distinct bands, which is sometimes the case, the sorting into classes is facilitated.

The ore deposits of Broken Hill, New South Wales, Leadville, Colo., and the Rammelsberg of Hanover, are occurrences of mixed sulphide ores which are remarkable for their size and high metallic contents. The Broken Hill ore is essentially galena and blende; that of Leadville, galena, blende and pyrite; that of the Rammelsberg, galena, blende, pyrite and chalcopyrite. In each case the minerals are argentiferous. The Broken Hill ore contains about 66% of sulphides; the Rammelsberg, about 75%; and that of Leadville upward of 90%.

Nassau.—There is an important lead producing district, from which zinc is also obtained, in the valley of the Lahn, Duchy of Nassau, where a series of remarkably strong veins are found in a graywacke belonging to the Lower Devonian formation. There are two great veins on which several mines are opened. One of these extends from a point near St. Goar, on the west bank of the Rhine, to Holzappel, on the Lahn, at which point it is richest—a distance of nearly eight miles. The other, the Ems vein, extends from Braubach, on the Rhine, across the valley of the Lahn to Deerbach, near Montabaur. The most important mines are situated at Holzappel, where they are operated by the Rheinisch-Nassauische Bergwerks- und Hüttenactiengesellschaft, and near Ems, where they are operated by the Emser Blei- und Silberwerke and the Silber- und Bleibergwerk "Friedrichssegen."

The mines at Holzappel, which have been exploited since 1785, had in

1892 been opened continuously on the strike of the vein for a distance of 11,480 ft. (3,500 m.). In general the ore of the Lahn is a complicated mixture of silver-bearing galena, blende, siderite, barite, calcite and quartz. It is concentrated in the dressing works at Laurenberg, Silberau and Friedrichssegen, which rank among the largest in Germany. The crude ore received at the Silberau works averages 4% Pb, 2·5% Zn and 54 g. silver per 1000 kg. The galena is concentrated to 36% Pb with 300 g. silver per metric ton, and the blende to 44·5% Zn; the blende does not carry silver. The loss in dressing is 8% for silver, 6% for lead and 34% for zinc.[1] The cost of dressing is about 85c. per metric ton of crude ore.

The mines at Friedrichssegen produce argentiferous galena, argentiferous chalcopyrite, pyrite, blende and siderite, with a quartzose gangue, which are classified by hand sorting and gravity concentration. The class of concentrates which is a mixture of blende and siderite, and assays about 15% Zn and 27% Fe, is calcined[2] and treated magnetically, yielding a product assaying 37 to 42% Zn, and at the most 6% Fe, which is sold to zinc smelters, and an iron ore assaying 40% Fe and less than 4% Zn which is sold to iron smelters (vide Chapter XI).

Rhenish Prussia.—Deposits of zinc ore occur at several places in Rhenish Prussia. At Gladbach, east of Cologne, they are found in troughs and basins in magnesian limestone of Devonian age, having the appearance of having been washed in mechanically. They are covered by later deposited beds of clay and shale, which contain brown coal. The ore is smithsonite, hemimorphite and galena mixed with shale. The mines of the Société Anonyme de la Vieille Montagne at Bensberg, near Gladbach, are very important. Their product is chiefly blende. At Altglück a bed of blende is intercalated in Lower Devonian shales and sandstones. The ore carries some galena and has a quartzose gangue. With depth the deposit splits up and diffuses. Veins of blende striking in various directions occur in the Eifel district of the Moselle, in Lower Devonian rocks. At the intersections they show enrichments. Deposits of blende are also found at Siebengebirge, on the eastern side of the Rhine below Coblentz.

Saxony.—The historic mines of Freiberg produce a small quantity of blende in connection with galena and other minerals, which come from the remarkable series of intersecting veins, seldom more than 2 ft. in width, but more than 900 in number, which traverse gneiss. Mining was begun there as early as the twelfth century. The blende of Freiberg is of the black,

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[1] M. Bellom, "Préparation Mécanique des Minerais dans la Saxe, le Hartz, et la Prusse Rhénane," Annales des Mines, 1891. No. 4. pp. 82 and 91.

[2] The heat attained is not sufficient to desulphurize the blende.

shining variety and is remarkably high in combined monosulphide of iron; it is also argentiferous. The ore raised from the mines is concentrated in the Himmelfahrt works, which produce besides galena and other products a zinc ore (blende) assaying 40% Zn and 300 g. silver per metric ton. This blende is smelted in a department of the Muldnerhütte, and the residues are sent to the lead furnaces for recovery of their silver contents.

Upper Silesia.—The zinc deposits of Upper Silesia occur in the extreme southeastern corner of the Province, near where the three empires of Germany, Russia and Austria meet. They extend into Russia, but the greater part of the mineral district lies in Germany. This entire region was formerly a part of the Kingdom of Poland, and at the present time the population in both the German and Austrian divisions is chiefly Polish. Lead was mined there as early as the twelfth century and zinc ore in the sixteenth.

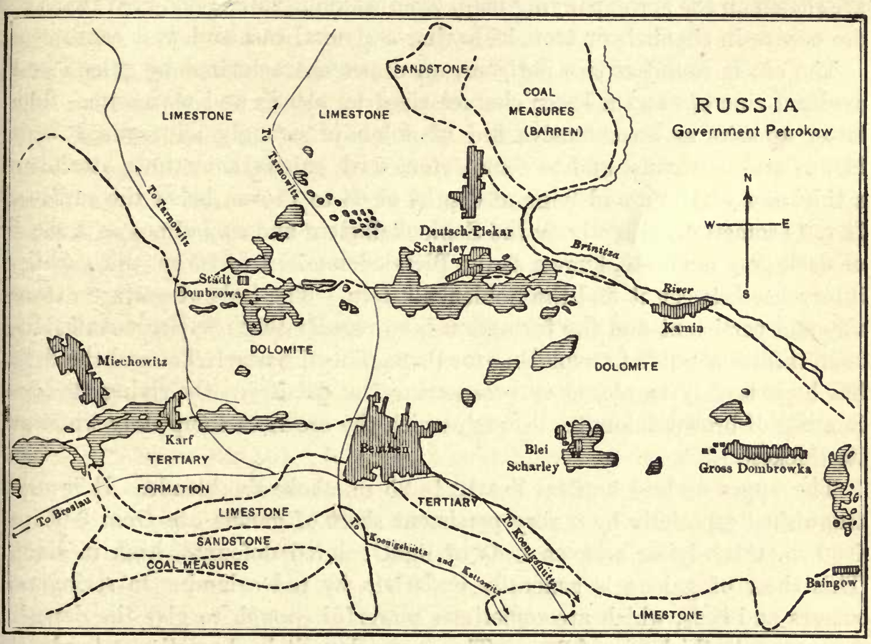

FIG. 18.—MAP OF THE ZINC MINING DISTRICT OF UPPER SILESIA.
Scale, 1:120,000.

Geology.—The lead and zinc deposits occur in the Muschelkalk series of the Triassic formation, which rest on the Bundsandstein of the same formation, the latter overlying Carboniferous rocks. The coal measures, which carry powerful seams of coal, come near the surface a short distance south of the zinc mining district, forming a conjunction between ore and fuel similar to that which exists in Kansas and Missouri, and presenting the basis for the great smelting industry which has been developed. The lowest bed of the Muschelkalk series is a blue limestone known as the Sohlenkalkstein, which forms the lower limit of the ore deposits. Overlying the Sohlenkalkstein are dolomites, which carry the ore.

The chief mineral district lies between the towns of Scharley, Brzosowitz, Kamin, Baingow, Antonienhof, Beuthen, Miechowitz and Städt Dombrowa. The situation of those places and the trends of the mineral zones are shown on the accompanying map, from which it will be observed that the ore occurs in channels or troughs having a general east and west course.

The ore is found at two horizons, an upper characterized by galena and oxidized zinc ore and a lower characterized by blende and marcasite. The latter appears as an extensive bed of dolomite strongly mineralized with blende and marcasite, and to some extent with galena, sometimes attaining a thickness of 15 m. and lying at depths of 64 to 115 m. below the surface. It rests sometimes directly on the Sohlenkalkstein and sometimes on a sheet of dark gray dolomite, known as the Blendedolomit, 1 to 1·5 m. thick which intervenes between it and the Sohlenkalkstein. The beds of ore are extensive and persistent and the formation is so regular that the Sohlenkalkstein is an infallible guide in exploring for them. They lie nearly flat and the drill has been largely employed in prospecting the country. Overlying the ore is a bed of brown dolomite, through which the ore rises completely when at its thickest.

The upper or lead horizon lies 12 to 25 m. above the blende. It is distinguished especially by a very persistent sheet of galena ore from 0·05 to 0·30 m. thick lying between beds of light colored dolomite, high in lime. This sheet of galena is generally underlain by red calamine in stringers, masses and beds, which are sometimes powerful enough to give the deposit an aggregate thickness of 2 m. The upper deposit is above the water level of the district and has been subject to oxidation and secondary mineralization; the lower is below the water level and generally presents unaltered sulphide ore, the only exception being at the edges of the troughs or basins, where the two horizons frequently unite; at such places ore bodies of great thickness are apt to be found.

Deposits of calamine in the form of nests and stock-works, pipes and sheets are also found in the dolomite, sometimes lying in or between beds of residual clay. There is no regularity in the deposits of this class, and being scattered through the entire dolomite formation of the Muschelkalk, prospecting for them is more difficult than for the regularly lying beds

(Flötze) of blende. These deposits of calamine, which were the first to be worked in the district, are now approaching exhaustion. The deposits of blende in the Beuthen-Scharley basin, to which attention was first directed in the decade 1860-1870, are so extensive, however, that it must be a long time before they are worked out.

Kinds of Ore.—The calamine of Upper Silesia is chiefly smithsonite; to a less extent hemimorphite, which mineral is rather characteristic of the lower levels. Near the Sohlenkalkstein the ore is generally white calamine (chiefly silicate); higher in the dolomite it is red calamine (chiefly smithsonite). Sometimes both kinds occur in the same deposit, as in the Elisabeth mine, where the ore body showed white calamine on the foot wall of Sohlenkalkstein and red calamine against the hanging wall of dolomite. The red calamine is rather ferruginous and frequently cadmium bearing. The blende of the great bed is of cryptocrystalline character and dark brown to black in color. It is apt to contain both cadmium and arsenic. Marcasite is generally associated with it, especially in the upper parts of the deposit, and sometimes next to the Sohlenkalkstein. Pyrite is rather rare, but galena is of frequent occurrence, appearing in the blende deposit in the form of stringers, seams and irregular bunches.

The zinc ore of Upper Silesia, especially the calamine, is of low grade, or rather it should be said the smelting and other conditions permit it to be sent to the reduction works in the form of low grade ore. A part of the product of the mines is dressed, certain of the works being highly efficient types of modern design; a large part is shipped as hand-sorted lump ore. Calamine as low in grade as 8% Zn is marketable, but ore of lower tenor must be dressed. The rich white calamine in lumps assays 43 to 45% Zn; red calamine 28 to 35% Zn. Concentrated calamine assays about 29 to 30% Zn. The crude sulphide ore which goes to the dressing works assays about 10% Zn. At the Neuhof mill ore assaying 10 to 11% Zn is concentrated in the ratio 5:1, yielding a product assaying 33% Zn (showing a saving of 66%). At the Neue Helene blende-mill (a highly elaborated plant) the blende concentrate assays 37 to 40% Zn. An average for the district is perhaps in the neighborhood of 33 to 35%. The purest blende ore in lump form assays 55 to 60% Zn; the average is 40 to 45% Zn. Blende as low as 18% Zn is marketable. Nearly all the Silesian zinc ore contains some lead, the best blende concentrates having about 1%, other grades 2% or more. (As in the Joplin district a considerable part of the lead in the crude ore is separated as a galena concentrate, assaying 70% Pb). Concentrated blende from the Neue Helene mill, in 1893, had the following composition: 41·23% Zn; 2·45% Pb; 5·35% Fe; 3·24 Al_2O_3; 1·43% SiO_2;

13·63% CO_2; 0·33% SO_3; 20·46% S; 3·52% MgO; 8·65% CaO; total, 100·29%. Concentrated calamine from the same mill assayed: 27·32% Zn; 5·35% S; 0·23% SO_3; 6·62% CaO; 23·02% CO_2; 37·46% undetermined (oxygen, silica, alumina, iron, magnesia, etc.). Other analyses of Silesian zinc ores are given in the following tables, which are compiled from data in *Verdichtung der Metalldämpfe in Zinkhütten,* by Doctor Victor Steger, and *Das Cadmium, sein Vorkommen, seine Darstellung und Verwendung,* by Edmund Jensch.

Besides the zinc ores proper the iron ore of Upper Silesia frequently contains zinc, some of which is recovered in the scaffolds and dust from the blast furnaces and is sold to the zinc smelters; some analyses of those products are given herewith, together with those of the iron ores from which they are obtained.

The footnotes to which the reference letters in these tables pertain will be found on p. 219.

ANALYSES OF CALAMINE.

No.	Mine	Variety	ZnO %	Fe_2O_3 %	CaO %	MgO %	SiO_2 %	CO_2 %	H_2O %	Ref.
I	Neue Helene.......	Red	40·46	12·08	5·23	3·91	4·04	25·70	8·29	*a*
II	“ “	“	39·05	12·41	9·17	4·33	5·74	24·66	4·35	*a*
III	“ “	“	15·94	15·79	20·73	9·79	1·82	33·08	2·72	*a*
IV	Karl Gustav........	White	11·13	6·77	18·24	4·71	9·87	30·09	......	*b*
V	“ “	“	14·96	5·80	17·30	4·47	12·53	25·63	......	*b*

No.	Mine	Variety	Al_2O_3 %	PbO %	MnO %	SO_3 %	Na_2O K_2O %	Total %	Ref.
I	Neue Helene...............	Red	......	......	......	......	......	99·71	*a*
II	“ “	“	......	......	......	......	......	99·71	*a*
III	“ “	“	......	......	......	......	......	99·87	*a*
IV	Karl Gustav...............	White	14·90	1·06	nil	0·58	2·38	99·77	*b*
V	“ “	“	16·17	0·72	0·04	0·44	1·63	99·69	*b*

Mine	Variety.	Product	Total Zn %	Zn combined with SiO_2 %	Pb %	Fe %	Ref.
Neuhof	Red	Klaubgalmei	14·92	4·64	0·84	18·50	*c*
“	“	Setzgalmei	18·20	5·12	1·20	12·11	*c*
“	“	Grabengalmei	9·94	3·88	1·54	9·88	*c*
“	“	Lagergalmei	9·60	3·50	0·96	19·12	*c*
“	“	“	11·33	2·92	1·76	29·50	*c*
“	“	“	11·22	4·12	1·10	24·16	*c*
Rokoko........	“	“	11·96	3·15	2·30	13·20	*c*
Hugo..........	White	“	15·30	2·40	0·36	5·84	*c*
“	“	“	13·16	4·24	0·92	8·65	*c*
“	“	“	17·52	7·10	0·50	3·15	*c*
Redlichkeit.....	“	“	12·34	4·00	0·72	9·86	*c*

ANALYSES OF BLENDE.

Mine	Product	Zn %	Pb %	Fe %	S %	Al_2O_3 %	$CaCO_3$ %	$MgCO_3$ %	Total %	Reference
Neuhof......	....	32·50	2·18	9·54	26·44	2 84	16·25	10·20	99·95	*c*
Unknown...	Stückbende....	33·25	1·86	15·15	33·29	0·64	9·45	6·25	99·89	*c*
	Erdblende.....	32·20	1·94	17·64	34·50	0·25	8·45	5·75	100·73	*c*
	Schliechblende.	34·35	2·18	9·62	27·78	3·15	12·65	9·40	99·13	*c*
Aufschluss....	Grobkorn......	30·54	1·50	25·42	40·12	......	......	....	......	*c*
"	Feinkorn......	25·20	1·86	24·16	39·16	......	......	....	......	*c*
"	Schliechblende.	29·80	1·92	21·50	37·54	......	......	....	......	*c*
"	Erdblende.....	37·08	1·88	4·64	24·62	......	......	....	......	*c*
"	Schlammblende.	28·50	1·84	13·95	28·42	......	......	....	......	*c*

ANALYSES OF IRON ORE.

Mine	Fe_2O_3 %	MnO %	PbS %	ZnO %	CaO %	MgO %	P_2O_5 %	*d* Volatile %	*e* Insol. %	Total %
Redlichkeit; Reden shaft .	61·60	2·05	1·96	3·27	0·43	0·15	0·28	13·15	16·10	99·49
" Silva shaft ..	62·90	2·50	2·07	3·17	0·40	0·06	0·22	11·95	16·60	99·37
Unschuld; Shaft No. 18.. .	58·80	2·60	1·92	3·93	0·45	0·09	0·33	9·80	22·00	99·22
" " No. 29. .	72·65	1·20	0·81	2·70	0·25	0·20	0·41	11·20	10·15	99·57
Georgenberg............	63·20	6·60	......	1·785	0·244	0·973	0·231	10·50	15·15	98·683

BY-PRODUCTS FROM SMELTING IRON ORE OF THE ABOVE CHARACTER.

No.	Class	ZnO %	PbO %	FeO %	Fe_2O_3	MnO %	S%	SO_3%	Ref.
I	Fluedust.....................	28·22	8·72	22·96	tr.	2 58	0·52	0·49	*b*
II	"	21·37	6·55	26·60	nil	3 58	0·26	0·70	*b*
III	Zinkbrocken (Ofenbruch).........	59·42	3·93	14·82	1·06	4 17	0·12	0·38	*b*
IV	Dust from gas purifiers............	26·68	4·50	25·96	nil	2·20	0·42	0·69	*b*
V	" " scrubbers (waschkästen) ..	13·46	1·32	30·41	nil	7 46	0·19	0·44	*b*
VI	Wall accretion	72·67	3·41	......	1·26	....	......	....	*b*
VII	" "	27·94	4·62	......	*j* 5·06	0.27	......	....	*b*

No.	Class	Cl %	CaO %	C %	MgO %	Al_2O_3 %	SiO_2 %	P_2O_5 %	Total %	Ref.
I	Fluedust.....................	0.02	tr.	11·68	......	0·30	23·64	......	99·13	*b*
II	"	0.07	nil	13·79	......	0·62	25·98	......	99·52	*b*
III	Zinkbrocken (Ofenbruch).........	....	0·10	2·02	0·08	1·26	12·34	0·16	99·86	*b*
IV	Dust from gas purifiers...........	....	0·21	14·28	0·04	0·84	23·62	0·25	99·69	*b*
V	" " scrubbers (waschkästen)	....	0·14	20·64	0·11	0·66	25·02	0·21	100·06	*b*
VI	Wall accretion	....	....	*g* 0·22	......	*h* 3·37	*i* 18·64	......	99·57	*b*
VII	" "	....	....	*g* 1·38	......	*h* 5·40	*i* 55·10	......	100·17	*b*

a, A. Lindner; *b,* E. Jensch; *c,* authority not mentioned; *d,* carbonic acid and chemically combined water; *e,* clay and sand; *f,* partly in metallic form; *g,* as graphite; *h,* soluble; *i,* firebrick; *j,* includes both FeO and Fe_2O_3. These foot-notes refer to all of the above tables of analyses.

Mining Conditions.—In considering the mining conditions in Upper Silesia it is necessary to remark that the more part of the mineral rights now belong to the comparatively small number of corporations and great capitalists who own and operate the smelting works. The portion of the ore production made by other individuals is relatively small. The mines and works can thus be operated on a systematic policy to secure the best result and there are few conflicting interests. The mining is conducted on an extensive scale, with large and permanent works. In 1893 I visited the Neue Helene mine (opened in 1876, since which time it has been one of the largest producers of the district) and entered it through a circular, brick-lined shaft, about 4 m. in diameter and 103 m. deep, from the bottom of which led off a main working gallery, 3 m. wide, 2 m. high and 1,000 m., more or less, in length, with brick walls the entire distance and roof of heavy plank resting on iron beams—a rather magnificent outlay, but one that indicated confidence in the permanence of the ore body.

Westphalia.—The chief deposits of zinc ore in Westphalia exist at Iserlohn and Brilon. At Iserlohn calamine and blende are found in irregular pockets, sometimes connected, at the contact between Eifel limestone and the underlying Lenne shale, both of Devonian age; also in crevices which traverse the limestone in a network. The ore occurs with residual clays and sands and has some galena intermixed. The deposits at Brilon are of similar character, but much of the ore is found in irregular crevices in the limestone and is associated with pyrite.

GREAT BRITAIN: *England.*—Zinc ore is found in connection with lead ore in the North of England, counties of Northumberland, Cumberland, Westmoreland, Durham and Yorkshire, in which region the formation is made up of thick beds of limestone of Lower Carboniferous age, alternating with sandstones and shales. The ores are associated chiefly with the limestones, which are traversed by a great number of veins striking in various directions. The so-called "rake veins" running diagonally across the strata are the most productive. They have a zig-zag cross-section, somewhat like that of a flight of steps, dipping nearly vertically (pitches) and stretching off horizontally along the strata (flats), then dipping vertically again, and so on. The flats are often particularly rich. The veins are usually 1 to 4 ft. thick, but sometimes attain 17 ft. Pipe veins are also known. The zinc ore of this region is blende. The mines at Nenthead in Cumberland are now being exploited on a large scale by the Société Anonyme de la Vieille Montagne, which acquired them a few years ago.

Zinc ore is also produced in Cornwall, the principal mines being north of Truro, in the western part of the County, and near Liskeard in the eastern part, where veins mineralized with blende and galena traverse Devonian clay slates. In the Isle of Man lodes intersecting Cambro-Silurian slates and grits and feldspathic rocks afford blende and galena. The Foxdale lode has been worked for a distance of four miles on its strike and has been found to attain a thickness as great as 40 ft. The Foxdale mine, which is situated at Foxdale, belongs to the Isle of Man Mining Co.; the Great Laxey mine at Laxey Glen is operated by the Great Laxey Mining Co.

Wales.—Zinc ore is mined at several places in Wales, but the most important producers are situated in Flintshire and Denbighshire, where the country rock is Lower Carboniferous limestone overlain by the Millstone Grit and the Coal Measures. The geological structure and ore deposits are both similar to those of the North of England, whence the limestone beds dip westward under the Coal Measures, Permian and New Red Sandstone, rising again in Flintshire and Denbighshire. The formation is traversed by parallel lodes, which are mineralized in the grit and limestone and are barren in the Coal Measures, where they are represented only by a fault fissure. The ore deposits occur normally as crevice-filling veins, but in the limesone they occur also as pipes and in caverns, and as flats, which form at the contact of the limestone and sandstone. The ore is blende, associated with galena, occurring largely with a clay gangue. The mineral zone runs from Llangollen on the south to Flint, Holywell and Prestatyn on the north. The Minera and New Minera at Wrexham and the Holkyn at Holywell are the principal producing mines.

In Montgomeryshire lodes cutting a strongly folded formation of Cambrian-Silurian beds, comprising slates, shales and sandstones, are worked. The ore is blende and galena occurring in pipes and lenses in the vein filling. In the most important mining district, the Van district, the lode is 12 to 80 ft. wide, filled with country rock and clay and traversed by seams of calcite, barite and quartz, the metalliferous minerals occurring in bunches connected by stringers. In Cardiganshire there are similar deposits; also in the English County of Salop (Shropshire), which adjoins Montgomeryshire on the east, the most important mines being at Minsterley.

GREECE.—The large production of zinc ore in this Kingdom is derived chiefly from the ancient mines at Laurium, which are thought to have been worked for lead and silver as early as 1200 B. C., and are known certainly to have been worked on a large scale as early as 600 B. C. The magnitude of the ancient workings is testified by the statement of M. André Cordella, director-general of the Société des Usines du Laurium, that in 1892 his company had 105,000,000 metric tons of refuse ore, assaying 4 to 8% Pb and 1,000 to 1,300 g. of silver per ton of lead, left by the former miners, while there was about 2,500,000 tons of old slag containing 10·5% Pb, which did not represent the entire production, however, inasmuch as large quantities of the slag made before the Christian era were dumped into the sea.[1] In 1860, after 17 centuries of neglect,[2] the mines were reopened by a French company, which had obtained from the Grecian government a concession for their exploitation. The Société des Usines du Laurium, successor to the original company, and the Compagnie Française des Mines du Laurium are the chief operators at Laurium at the present time.[3]

The country rock at Laurium consists of alternate formations of limestone and mica schist (probably of Silurian age) varying in thickness from 200 to 300 ft. and aggregating about 1,200 ft. The ore occurs in lenses and pockets at the contacts of the lower formation of schist with the underlying and overlying limestone; also in funnel shape pockets penetrating the limestones. Crevices which cut the schists contain ore occasionally and irregularly. The bedded deposits are from 1·5 to 40 ft. in thickness. In the Camaresa district of Laurium they are developed over an area of more than two square miles.

The ores are galena and blende, with pyrite and chalcopyrite intermixed, and cerussite and smithsonite, the gangue being largely carbonate of iron and ocher. Those of the upper contact are chiefly smithsonite and are less rich than the lower. On the lower contact the galena occurs frequently as a central band in the deposit, separated from smithsonite on both sides by a variable thickness of ocher. Blende is found chiefly in the lower horizon. In the case of the Laurium ore deposits Posepny considers it doubtful if the smithsonite was derived as a secondary product from the blende.

The ancient Greeks had no use for zinc ore and those deposits remained untouched until the French companies undertook operations. It has been the Compagnie Française des Mines du Laurium which has concerned itself especially in zinc mining. The ores got at first were high grade, assaying 60% Zn after calcination (smithsonite is referred to) and of excellent quality. During the decade 1890-1900, however, most of the high grade ore appeared to have been won, while the grade of the smithsonite produced ran down to about 40% Zn after calcination, and it was much more ferruginous in character. In 1898 new ore bodies were opened and production increased again. From 1875 to the end of 1893 Laurium exported 555,000 metric tons of calcined calamine. The production since 1893 is given in Chapter IV.

Elsewhere in Greece there is a small production of zinc ore from the Island

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[1] A. Cordella, The Mining and Metallurgical Industries of Laurium, prepared for the Columbian Exposition at Chicago in 1893.

[2] It has been estimated from the accumulations of slag that the ancient miners obtained from the deposits at Laurium over 2,000,000 tons of lead and 270,000,000 oz. of silver.

[3] Extensive operations by the French companies did not begin until 1876.

of Antiparos. At Mt. Hymettus on the mainland zinc ore is found in limestone.

Italy.—The zinc mines of this Kingdom, which are at present the most productive in Europe after those of Germany, are situated chiefly in Sardinia. Less important mines are worked in Lombardy, Piedmont and Tuscany.

Sardinia.—The richest mines in Sardinia are situated in the Iglesias district in the southwestern corner of the island, where both lead ore and zinc ore are found. Mining was carried on there by the Romans and likely by the Carthaginians and Phoenicians before them. The mines surround Iglesias to the south and west and extend about 15 miles to the north. The country is composed of a central mass of granite surrounded by Silurian schists and graywackes, Cambrian grits, quartzites and schists and a "metalliferous" limestone or dolomite, which is probably of Silurian age. Deposits of ore occur in all of those rocks, but chiefly along the contacts of the limestones and schists; they are found also in crevices in the limestone and along the contacts between two different limestones. The lead ore and zinc ore are intimately associated, but the proportion of blende increases with depth. Large deposits of smithsonite and hemimorphite are found near the surface.

The Malfidano mines, situated about eight miles northwest of Iglesias, are among the largest producers of zinc ore. Calamine is found there interstratified with limestone and sometimes in alternating beds. The ore carries a small quantity of lead. As sent to the dressing works it assays about 15% Zn. It is concentrated in the ratio 4·5 :1, yielding a product which assays about 35% Zn. (a saving of about 52% of the zinc in the crude ore). Lump ore is also produced. All the ore is calcined at the mines before shipment. The average of 25,000 tons exported to Antwerp, about 1892-1893, was 54·40% Zn; 6·00% Pb; 13·75% O; 6·80% $Fe_2O_3+Al_2O_3$; 6.60% $CaO+MgO$; 9·40% SiO_2; 2·80% H_2O+CO_2; total 99·75%. The mines are operated by the Société des Mines de Malfidano, which since 1894 has had smelting works of its own in France.

At Caitas, near Malfidano, calamine occurs in conical or columnar bodies in limestone on either side of a great zone of brecciation, 100 ft. wide, which traverses the country rock; with depth, blende and some galena are found.

The Monteponi mines, about one mile southwest of Iglesias, are the oldest in the district, having been worked successively by the Carthaginians, Romans and Spaniards. They produce both lead and zinc ores, which occur in separate deposits and at different horizons. The zinc ore occurs in crevices and brecciated masses in limestone. It assays about 33% Zn and contains much iron oxide, which is separated by magnetic concentration (vide Chapter XI). Some cerussite is mixed with the ore.

In 1893 a dressing works capable of treating 250 tons of ore per day was erected at Monteponi to work an accumulation of ore, estimated to amount to 800,000 tons, and to contain 13 to 18% Zn and 1% Pb, which had previously been considered of too low grade to work. This ore was a mixture of smithsonite, hemimorphite, blende and galena, with a gangue of siderite, feldspar and chalk. It yielded about 13% of concentrate.[1]

The Montevecchio mine, which is exploited chiefly for lead, is opened on a huge lode, which traverses Silurian schist almost parallel to the contact of the schist against the granite. The lode has been traced nearly six miles along its outcrop and is 75 to 100 ft. thick. The ore which is galena, mixed with blende, pyrite, chalcopyrite, siderite, barite and quartz, occurs as veins along the hanging and foot-walls and also as lenses in the interior of the filling of the lode. The proportion of blende increases with depth.

Other Mines.—The San Giovanni mines, two miles southwest of Iglesias, have irregular lodes, standing nearly vertical, in limestone, which contain argentiferous galena in a gangue of quartz, barite, limestone and clay and some blende in columnar masses or zones. The Malagalzetta mines, a few miles north of Iglesias have shallow pockets of zinc ore in limestone, near the surface. At Nebida, five miles west of Iglesias, there are great chimneys of calamine, sometimes as much as 60 ft. in diameter and extending to depths of 600 ft., in limestone. Veins of galena and calamine are also worked, the wall rock being much mineralized by the zinc ore.

Composition of Sardinian Ore.—The following analyses of calcined calamine show approximately the composition of the ore at present exported from Sardinia.[2]

No.	%Zn	%Pb	%Fe	% Ca	% Mg	%SiO_2	No.	% Zn	%Pb	%Fe	%Ca	%Mg	%SiO_2
I	50·83	4·50	7·63	5·25	3·70	7·24	VII *c*	44·70	1·40	15·26	4·25	3·60	9·20
II	44·66	1·00	16·16	2·40	2·50	11·50	VIII	48·00	2·28	12·95	2·38	3·64	8·42
III	47·70	2·73	13·65	2·75	2·25	10·92	IX *d*	46·00	6·00	*a*8·50	4·50	1·50	8·00
IV *b*	49·34	7·90	8·22	2·88	1·35	8·30	X	51·90	4·50	*a*4·27	5·75	0·70	12·60
V	54·30	0·25	10·15	1·75	1·57	10·60	XI	45·65	4·20	*a*5·25	13·55	0·72	7·85
VI	46·80	12·50	14·64	2·92	2·55	5·96							

a Includes manganese. *b* Also contained 1·00% S. *c* Also contained 0·56% S. *d* Also contained 0·19% Cd.

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[1] Eng. and Min. Journ., Sept. 22 and 29, 1894, pp. 269 and 293.

[2] Ad. Firket. Annales des Mines de Belgique, Vol. VI, No. 1.

Zinc Deposits of the Italian Mainland.—The Tenda mine in the Turin district of Piedmont has galena, with which blende and pyrite are associated. At Argentiera, near Aurongo, in Lombardy, zinc ore is found in irregular deposits in Lower Triassic shales and dolomites. At Bottino, in Tuscany, blende occurs with argentiferous galena, stibnite, pyrite and siderite in quartzose lodes in Paleozoic schists. The English Crown Spelter Co. operates the Costagels, Gremme and other mines in the Valle Seriana, district of Milan, Province of Bergamo, whence it obtains about 5,000 tons per annum of calamine assaying 44% Zn.

The zinc deposits of Italy, aside from those of Sardinia, are of considerable importance. Besides the ore which is exported to Wales, the smelters of other countries receive a good deal therefrom. Thus, the Belgian smelters in 1898 imported 59,118 metric tons of ore from Sardinia, and 12,072 tons from elsewhere in Italy.

RUSSIA.—Little has been published as to the zinc resources of Russia, which it may be inferred have not yet been thoroughly explored. Only the deposits of the Caucasus and Poland have been described, and of those only the Polish have been exploited, the Caucasus being still too remote from smelting works.

Caucasus.—Various discoveries of zinc ore have been reported from this region, especially in the vicinity of Alagir, about 50 versts from the Wladikawkas railway, where a Belgian company has operated. The occurrence of blende near the Tschorok river, between Artwin and Bortschka, in the Kutaïs Government, has been mentioned, and in connection with galena at Petrowsk. Near Tschiatury, in the Kutaïs Government, blende assaying 57·82% Zn, has been found.[1]

Poland.—The zinc mines of this Kingdom have been opened on deposits which are an extension of those of Upper Silesia (q. v.), and are probably similar to them in most respects. Their exploitation has been confined so far to near the surface and has been done mostly by open pits. In appearance the ores are identical with Silesian. The principal mining is near Olkusz and Boleslaw, where a great deal of lead ore is said to have been got as early as the fifteenth century. The Boleslaw mines are owned by the Sosnowice Company; those near Olkusz belong to the Crown, and are worked by lessees. As mined at present, the Polish zinc ore is of lower grade than the Silesian (which is largely, if not wholly, due to the facts that but little blende is mined and the calamine is less extensively dressed by washing). There are two zinc smelters in Poland, namely the Sosnowice Company, which owns the Paulina works and Derwis, Pomeranzow &

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[1] N. Sokolow. Zap. imp. russk. tech. obschtsch., 1897, XXXI, viii, p. 7.

Co., who lease the Pod Bendzinen works, which belong to the Crown. The mines and smelteries are situated near the town of Bendzin.

SPAIN.—The large production of zinc ore in Spain is derived chiefly from the Province of Santander, in the north of the Kingdom, bordering on the Bay of Biscay; a smaller quantity is obtained from mines in the vicinity of Cartagena, in Murcia, a province of Southeastern Spain, abutting on the Mediterranean Sea. Zinc ore is also found in the Province of Teruel, in the East of Spain.

Murcia.—Zinc ore is found in this Province, near Cartagena, more or less in connection with the important lead deposits which occur in that region. The country rock is Permian limestone, overlying a formation of schist. Lenses of blende occur in the latter. Smithsonite, associated with siderite, is found along crevices and in masses in the limestone.

The shipments of blende from Cartagena in 1901 were 3,750 tons to Great Britain, 45,900 tons to Belgium and 2,080 tons to Germany, a total of 51,730 tons, which was 12,440 tons more than in 1900. In 1899 the shipments were about 80,000 tons. The shipments of calamine in 1901 were 197 tons to Great Britain, 2,512 to Belgium, and 1,690 to Germany, a total of 4,399, or about 1,000 tons more than in 1900.

Santander.—The zinc deposits of this Province occur in dolomites of Cretaceous and Jurassic age and in Lower Carboniferous limestones. The most productive are those of Reocin, Udias, and La Florida. Their ore consists chiefly of smithsonite and hydrozinkite, together with hemimorphite and blende, the last appearing in the lower levels. The deposits assume chiefly the form of bed-like masses in Cretaceous and Jurassic dolomites, which they have evidently replaced; they also occur in crevices and as impregnations. The production of the Reocin mines in 1896 was 24,000 tons of calamine and 810 tons of blende; Udias and La Florida produced 7,200 tons of calamine. The mines are controlled by the Compagnie Royale Asturienne des Mines, which smelts the ores at Arnao, near Aviles, in the Province of Asturias, and at Auby in France.

The Picos de Europa mines are in disturbed Carboniferous limestone. The Andosa deposits show a series of parallel veins, a few inches to 32 ft. in width, in a zone about a mile long and half a mile wide. The ore is principally smithsonite. In the Aliva deposits, six miles further west, the ore is chiefly blende.

Teruel.—Valuable deposits of calamine assaying 49 to 54% Zn exist at Linares in this Province, where they were first opened about 1890. Their production has not yet been large, owing to the lack of cheap transportation facilities, there being no railway near, but it is considered that the mines will be an important source of zinc in the future.

SWEDEN.—Zinc ore is found in Sweden in the provinces of Orebro, Kopparberg and Nerike, but the mines of Ammeberg in Nerike, which are owned and operated by the Société Anonyme de la Vieille Montagne, are the only ones of importance. Those mines are situated at the northern end of Lake Wetter, about 120 miles W.S.W. of Stockholm, and about eight miles from Ammeberg, with which place they are connected by a railway. The dressing works and roasting furnaces are at Ammeberg and Johannesborg, about three miles from Ammeberg. The country rock is schistose gneiss of Laurentian age, folded and contorted. This rock, which is fine grained, is in places impregnated with blende, pyrite and galena, the mineralized portions having the form of lenses, which occupy a nearly vertical position and conform to all the undulations of the country rock. Their average thickness is about 25 ft., though they sometimes attain 50 ft.; their length reaches hundreds of feet and in depth they have been exploited 600 ft.

The blende appears to have taken the place of the mica in the schist, so the mineral has ordinarily a finely intermixed gangue of quartz and feldspar, but there is considerable blende, which is sufficiently pure to be separated by hand sorting, while a good deal of worthless country rock, or gangue, can be culled in the same manner. About 7,000 tons of lump blende are thus picked out annually. The ore which goes to the dressing works contains about 20% Zn and under 1% Pb. It is concentrated with great care, the ore being slightly roasted previous to crushing and jigging in order to facilitate removal of the pyrite (vide Chapter XI), so that the loss in dressing is comparatively low, amounting to only 20 to 21%. The ratio of concentration is very low, however, inasmuch as after roasting, to which all the ore is subjected at the mines before shipment, the mineral assays only 38% Zn, its tenor in Fe_2O_3 being 6%.[1] The culled blende raises the average tenor of zinc in the ore shipped to 42%. Great pains are taken to effect a good separation of the galena in the ore and the Ammeberg concentrate ranks consequently as mineral of excellent character as to purity.

TURKEY.—According to H. R. Jastrow,[2] zinc ore is found to some extent in Balia, Province of Brussan, but the chief source of supply is Karsasu on the Black Sea. Zinc ore is also found in the neighborhood of Smyrna. The total production of the Empire approximates 5,000 tons per annum.

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[1] P. G. Lidner. Trans. Am. Inst. Min Eng., XXIV, 494.

[2] Eng. and Min. Journ., May 18, 1901.

the most part being shipped to Belgium. In 1898, Belgian smelters received 3,053 metric tons of ore from Turkey.

ZINC ORE DEPOSITS OF AFRICA.

ALGERIA.—Zinc ore is mined rather extensively at various places in this Colony. The deposits of Sakamody, Guerrouma and R'arbou are in the northern part, near the Little Atlas Mountains. They occur in veins in Cretaceous marls, schists and limestones, all more or less argillaceous. The ore is calamine above the water level and blende below it, the gangue being country rock together with barite and siderite. Galena is apt to be associated with the blende. At Sakamody there is a schist breccia with blende cementing the fragments. In the Department of Constantine, in the northeastern corner of Algeria, are the Hammam N'bails and Ain Arko mines, which are exploited by the Société Anonyme de la Vieille Montagne. The Oued Moziz mine in the Department of Oran yields both zinc and lead ores. New explorations for zinc ore were undertaken in 1896 in the Djebel Belkif, Commune de Morsott, where there are promising outcrops of calamine and galena, not far from the Youks and Morsott stations of the Bône-Guelma line of railway from Soukarras to Tébessa.

TUNIS.—Five calamine deposits are worked under concessions in this Colony, all situated in the northern part.[1]

1. Kanguet and Tout lies about 30 km. from Beja, on the road from Beja to the port of Tabarka, which at present is only finished from Beja to the mine. The ore occurs in an irregular stock-work. The mine is worked open-cast. The ore is sorted into lump and fines, the former being calcined in kilns with charcoal. The calcined calamine is freighted by mule or camel to Beja, and thence to Tunis. From Tunis the ore goes to Antwerp. The production is between 3,000 and 4,000 tons per annum.

2. The Sidi-Ahmet concession, belonging to Compagnie Royale Asturienne des Mines, lies north of the Sidi-Ahmet Mountains, about 40 km. from Beja. It contains at least three deposits, of which the first yielded about 35,000 tons of good calamine previous to 1896. The ore is hand-sorted and calcined in kilns. The annual production is about 3,500 tons of calcined ore.

3. Fedj-el-Adoum lies 20 km. southwest of Tebursuk, in the highest part of the Jouaouda Mountains, which rise to an elevation of 907 m. There are three groups of deposits, of which only one is exploited. This is very extensive; in 1896 about 50,000 tons of ore was estimated to be in sight. The ore is calcined, after hand-sorting, and is shipped via Tebursuk.

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[1] E. de Fages, Glückauf, March 27, 1897, p. 245.

4. Zaghouan lies about 60 km. south of Tunis, near the village of Zaghouan. There are two important deposits; in 1897 about 40,000 tons of carbonate ore, averaging 40% Zn, had been proved. Zinc silicate appears more frequently here than in the other localities. The mines are connected by a mule-path and a 15-km. tramway with the railway between Tunis and Zaghouan. There are three shaft furnaces and one reverberatory at the mines for calcining the ore. The annual production is 5,000 tons of calcined ore.

5. El Akhouat lies about 32 km. southwest of Tebursuk. The ore is carried in two-wheel carts to Madjez-el-Bab, and thence by rail to Tunis. This concession was only granted in 1896, and its production has not yet become important..

ZINC ORE DEPOSITS OF AUSTRALIA, TASMANIA AND NEW CALEDONIA.

AUSTRALIA.—The production of zinc ore in Australia is confined to the Broken Hill district of New South Wales, where it is made by the mechanical separation of the blende of the mixed sulphide ore which exists there in vast quantity. The ore deposits of Broken Hill occur in an immense lode in primitive, crystalline schists, which cuts the country rock at a slight angle in its dip, but agrees with it in strike. The schist is for the most part fine grained, but is sometimes gneissose in character and at other times is silicified to a quartzite; garnetiferous sandstone also occurs. The lode has a prominent cap of iron ore at the surface, which further down passes into ferruginous quartz, silver bearing kaolin and oxidized silver-lead ore, and below the water level changes to galena and blende. Included portions of the country rock are common in the lode and at one point it is split by the schist, which has led to the suggestion that the lode is a "saddle reef." The thickness of the ore body averages about 60 ft., but in some places it has been found 100 ft.

The sulphide ore, which is the only source of zinc in these mines, is a mixture of galena and blende, which are so intimately commingled that distinct minerals are difficultly detected by the unaided eye. It assays 10 to 15 oz. silver per metric ton, 15 to 20% Pb and 20 to 25% Zn. The gangue minerals are garnet, rhodonite, quartz and feldspar, the garnet being especially a constituent of the friable ore, which constitutes about 20 to 30% of the whole production. The percentage of iron in the ore is not very high; part of it exists as sulphide, the remainder as a constituent of the gangue minerals, the same being true of the manganese. Ashcroft gives the following analysis as representative of the available ore: 20% Pb, 20%

Zn, 8% Fe+Mn (present as sulphides), 18% S, and 34% gangue.[1] According to Greenway, the average grade of the Broken Hill ore is not so high, his estimate being 14 to 20% Pb, 10 to 20% Zn, and 6 to 16 oz. Ag per ton.[2] The intimate association of the constituent minerals of this ore in which considerable proportions of both the galena and the blende occur as mere films and specks in joints and in fracture and cleavage planes, and the presence of the garnet (sp. gr. 3·5 to 4·3) and the rhodonite (sp. gr. 3·5 to 3·7) make it very difficult to obtain a clean high grade zinc concentrate by gravity concentration; and the economical treatment of the Broken Hill ore, of which upward of 10,000,000 tons are available,[3] has long been a complex metallurgical problem, of which a satisfactory solution has not yet been found.[4]

The present results obtained in the treatment of the mixed sulphide ores of Broken Hill, New South Wales, by wet dressing and smelting the concentrates are shown in a recent report of the Sulphide Corporation, Limited, which operates the Central mine at Broken Hill, where it has a dressing works, the concentrates being smelted at works at Cockle Creek. In 1899 there were treated 201,411 tons of ore, which produced 40,989 tons of lead concentrate, 2,338 tons of zinc concentrate, 41,173 tons of middlings, 16,125 tons of slimes and 100,786 tons of tailings. The recovery of lead in the concentrates was 70·3% and of silver about 45%, the total contents of the ore treated having been 37,277 tons of lead (18·508%), 44,362 tons of zinc (22·026%), and 2,510,033 oz. of silver (12·46 oz. per ton). The cost of mining and milling was as follows: mining, including handling waste and filling, $2·37 per ton; development, $0·24; traction, crushing and maintenance, $0·18; dressing, $0·73; total, $3·52.

At the Cockle Creek smelting works there were smelted 31,305 tons of lead concentrate, 2,732 tons of Ashcroft residues, 67 tons of dry ore from the Central mine, 1,413 tons of purchased ores, and 1,041 tons of matte and flue dust, a total of 36,558 tons, which yielded 18,596 tons of silver-lead, containing 946,855 oz. of silver and 1,787 oz. of gold; and matte containing 876 tons of lead and 91,140 oz. of silver. The total quantity of silver produced was 1,037,995 oz. The approximate averages were 50·5% Pb, 28·4 oz. silver and 0·05 oz. gold per ton of ore smelted. The total profit, before deducting interest on the bonds, depreciation, etc., was £131,218.

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[1] In a paper entitled Sulphide Ore Treatment by the Phœnix Process, read before the Institution of Mining and Metallurgy, June 19, 1901.

[2] The Mineral Industry, IX, 745.

[3] E. A. Ashcroft, Transactions of the Institution of Mining and Metallurgy, 1898.

[4] The direct smelting for zinc by the Picard & Sulman process of the middlings obtained from this ore by gravity concentration has lately been inaugurated at Cockle Creek, but the process is still too new to determine its commercial results. Vide Eng. and Min. Journ., July 26, 1902.

New Caledonia.—At least one large deposit of zinc-lead sulphide ore is known in the northern end of this island, where it occurs in a schistose formation. The deposit is said to be of rather large dimensions. The upper portion has become thoroughly oxidized, losing at the same time much of its zinc contents and becoming an ore of a class suitable for lead smelting. A lead smelting plant was erected on the seashore a few miles away to treat this ore, but has been abandoned; not, however, for lack of ore, there being a good deal of carbonate ore still in sight, it is said, independent of what may yet be uncovered. The tenor of the ore in silver is low, being 10 to 25 oz. per ton. Beneath the oxidized ore occur the undecomposed lead-zinc sulphides, which in appearance resemble those of the Broken Hill district in New South Wales.

Tasmania.—A large deposit of mixed sulphide ore has been opened in the Roseberry mine, belonging to the Tasmanian Copper Co., on the western slope of Mt. Black in the County of Montague in the West Coast district of Tasmania, about 12 miles northeast of Zeehan and 18 miles north of the Mt. Lyell copper mines. Roseberry is distant from the port of Burnie, on Emu Bay, 72 miles by the line of the Emu Bay Railway Co. The country rock in the vicinity of Rosebery is micaceous schist, with interstratified sheets of limestone, slate, sandstone and quartzite and frequent intrusions of diorite. There are numerous large and well defined mineralized lodes, which usually cut across the strike of the stratified rocks.

The Rosebery lode is apparently a fissure vein, which has an average dip of about 60°, and has been traced on the surface for upward of 3,000 ft. Its width where opened underground is from 15 to 41 ft., averaging about 25 ft. The ore is a solid body of sulphide, chiefly a laminated mass of pyrite and blende, in which there occur irregularly bands of an intimate mixture of blende and galena (about three parts blende to one of galena), and also bands of comparatively clean pyrite. Throughout the whole deposit there is a small quantity of chalcopyrite, with occasional small veins and pockets of ore of good grade in copper. At the end of 1899 the Tasmanian Copper Co. estimated that there was upward of 400,000 tons (of 2,240 lb.) blocked out and probably a great deal more in the mine. The average grade of the ore was computed to be 0·155 oz. gold and 8·625 oz. silver per ton, 0·77% Cu, 25% Zn and 4·73% Pb, and about 20% Fe, the remainder being sulphur corresponding to the above metals, and the gangue of silica and alumina, chiefly the former. The cost of mining was estimated at 12s. per 2,240 lb., the rate of wages in the district being 9s. per eight hours. Coal costs 20s. per ton, plus an import duty of 4s. and coke 44@46s., both those supplies being obtained from New South Wales.

XI.

MECHANICAL CONCENTRATION OF ZINC ORES.

The Belgian and Rhenish methods of zinc winning, which are the ones generally in use outside of Upper Silesia, require, or at all events give the most satisfactory results, with a comparatively high grade ore—i.e., one that contains at least 40% Zn, after calcination or roasting, and preferably more. A roasted blende which assays 50% Zn must have contained 45% Zn before roasting if 10% in weight were lost in that operation, and a calcined calamine (zinc carbonate) of the same tenor would have had 37·5% Zn before calcination if the loss in weight had been 25%. A sulphide ore assaying 45% Zn contains 67% blende, even if the latter bears no isomorphous iron or cadmium sulphide. There are few blendes which are quite free from iron monosulphide chemically or mineralogically combined with the zinc, and the foregoing calculation shows the rather high degree of purity that is possessed by a raw sulphide ore assaying only 45% Zn. European zinc smelters do not require this grade to be exceeded, but in the Joplin district of the United States it has become the custom to produce blende concentrates assaying 60% Zn. These contain a little galena and pyrite and some monosulphides of iron and cadmium, so that it may be assumed that an assay of 60% Zn indicates a tenor of at least 92% of "mineral," the remainder being chiefly silica. It is obvious that so high a degree of concentration cannot be effected mechanically without extraordinary losses in the tailings except in the case of favorable kinds of ore.

Objects and Limitations of Concentration.—The mechanical concentration or dressing of zinc ores is done practically by manual selection, by gravity separation and by magnetic separation, two or all three of these systems often being combined. In concentrating zinc ores it has to be kept in view not only to enrich the ore by removal of the gangue, which is composed usually of light minerals, but also to separate the heavy minerals which may be injurious in the smelting process. For example, all of the lead, iron and manganese minerals are particularly objectionable and ought to be eliminated as completely as can be done economically. Even when this be done blende ore is apt to contain a large percentage of iron on account of monosulphide, FeS, combined isomorphously with the zinc sulphide; thus the shining, black blende of Freiberg, Saxony, sometimes contains as much as 30% Fe; a specimen of similar appearance from Mexico analyzed by me gave 10% Fe. Cadmium sulphide also occurs isomorphous with zinc sulphide, especially in the case of reddish blendes, and similarly the carbonates of iron and cadmium are associated ismorphously with zinc carbonate, the Silesian ores almost invariably containing from 0·1 to 0·2% Cd.

The dressing of ordinary zinc ores, which are apt to be mixtures of blende and galena, with such gangue minerals as quartz and calcite, or calamine with similar gangue, does not offer especial difficulty, owing to the great difference in the specific gravities of the component minerals. The presence of pyrite, marcasite, barite, or siderite complicates matters because those minerals are of about the same specific gravity as blende, smithsonite, hydrozinkite and hemimorphite, and in this case magnetic separation is likely to be necessary as an accessory process. The complex silver-bearing, zinc-lead sulphide ores, like those of the Rammelsberg in the Lower Harz (Germany), Kokomo and Leadville, Colo., Broken Hill, N. S. W., and elsewhere, cannot be separated, either by gravity or by magnetism, so as to produce a first class zinc ore, although recent progress in the art of ore dressing, especially the introduction of the improved shaking tables, has enabled a marketable zinc product to be obtained from the Leadville ore. Magnetic separation has been applied with more or less success at Broken Hill for the separation of the mixed sulphide ore which is mined there; and with quite satisfactory results in connection with the mixed ore of Leadville, Colo. With calamine ores gravity separation is frequently rendered difficult by their earthy or drusy character, which prevents so successful a removal of impurities of nearly the same specific gravity as in the case of the denser and crystalline blendes.

In general the mechanical concentration of zinc ore does not differ in principle from the practice with lead and copper ores and with respect to the subject reference should be made to the special treatises upon it. The scope of this work permits only general suggestions and detailed accounts of a few special processes, except that the subject of hand sorting, because of its importance and the scant attention that is usually given it, is treated rather fully.

MANUAL SELECTION OR HAND SORTING.

When the mineral is coarsely mixed with the gangue it is feasible to separate it to a considerable extent by means of hand sorting. This process is generally practised to a certain degree as a part of the mining of the ore, in which high grade ore may sometimes be broken out of the lode separately; sometimes pieces of worthless waste can be picked out of mixed ore and left underground to fill old stopes, etc. As to whether the hand sorting process shall be carried further on the surface depends on the local conditions, especially the losses experienced in mechanical dressing, the cost of dressing and the cost of sorting, of which the loss in dressing is likely to be the most important consideration. If pure mineral be broken, a portion of it will be converted into fines (the percentage depending upon the brittleness of the mineral, the size to which it is broken, and the method of breaking) and in washing with water a good deal of such fines will escape settling, no matter how perfect and painstaking be the process of settling. It is therefore an axiom in mechanical dressing to avoid breaking the mineral any finer than necessary, and avoid breaking it at all if that be economical. These conditions are met most completely by the process of hand sorting, wherein two classes of ore will usually be made: I, clean mineral, perhaps divided into (*a*) blende and (*b*) galena; and II, mixed mineral and gangue, which is sent to the dressing works. If the cost of dressing be high it is sometimes economical to sort out a third class, III, clean gangue, or waste, thereby saving the expense of crushing and washing worthless material and increasing the capacity of the mill. Under other conditions it may be cheaper to let the waste go through the mill. This can be determined only by a study of the conditions in each particular case. It may be remarked, however, that the merits of hand sorting are seldom given in America the consideration which they deserve, and where practised the system is rarely designed to secure the best results.

METHODS OF BREAKING THE ORE.—Hand sorting of ore involves usually two processes: I, breaking; II, selection. The breaking is done mechanically by jaw crushers or manually with the aid of hammers. The former method is the cheaper, but it produces the more fines; while breaking with hammers is not excessively costly if properly done, and if the ore must not be broken to a very small size. The size to which the ore must be broken depends obviously on what is required to free the pieces of pure mineral for the maximum; and what can be picked over economically for the minimum; the choice between the two extremes will depend naturally upon the grade which it is desirable to obtain for the sorted mineral.

Proportion of Fines Made.—With respect to the relative efficiency of machine-breaking and hand-breaking in so far as the proportion of fines made is concerned, in default of figures relating specifically to zinc ore, I am compelled to quote the results of a test on a copper ore by Doctor E. D.

Peters, Jr., which, so far as I am aware, is the only reliable datum of this nature on record.[1] The test was made with three different varieties of sulphide ore of average hardness, the lots aggregating 2,220 tons, after the fines which they contained, as received from the mine, had been removed by passing over a screen of three meshes to the linear inch, the apertures of the screen being 6×6 mm. One half (1,110 tons) was broken by means of a 7×10-in. jaw crusher, with corrugated plates (which produce decidedly less fines than smooth plates) run at 240 r. p. m., with a discharge opening of 2·5 in.; the other half was broken by experienced men, with proper hammers, into pieces of a similar maximum size—i.e., 2·5 in. The products were passed separately over a 3-mesh screen (6×6 mm.) and the fines were weighed. The results were as follows:

Character of Product.	Jaw Crusher.		Hand Spalling	
	Tons.	%	Tons.	%
Fine product—below 6 mm in diameter.	192·25	17·32	103·34	9·31
Coarse product—bet. 6 mm and 64 mm..	917·75	82·68	1006·66	90·69
Total............................	1110·00	100·00	1110·00	100·00

Cost of Breaking.—With respect to the relative cost of machine-breaking and hand-breaking the advantage is decidedly in favor of the former. With a jaw crusher capable of breaking 200 tons of ore per 10 hours and elevator and screen of corresponding capacity the cost will be from 7 to 10c.[2] per ton (varying according to the wages for labor) including power, labor, repairs and renewals, and amortization of plant, if the plant be run at its full capacity. The cost of spalling 100 tons per 10 hours is computed by Peters substantially as follows: One foreman, $2·50 per day; 14 men breaking ore, including screening and loading, @$1·50 per day, $21·00; four men sledging and loading, @$1·50, $6·00; five hammer handles, @30c., $1·50; 7 lb. steel, @15c., $1·05; labor of smith in making hammers, one third day, @$3·00, $1·00; screens, forks and shovels, $1·67; general repairs, $0·55; total, $35·27 (=$0·3542 per ton), of which $29·50 is for labor and $5·77 for material.

SPALLING.—Such an economical hand-spalling is possible only when the work is properly conducted. Attempts to sort ore with one set of men, who do both the breaking and sorting, are likely to be failures, because of the costliness of the process. The breaking should be done by one gang and the sorting by another. The breakers, or spallers, should stand up to the work,

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[1] Modern Copper Smelting, p. 89.

[2] The cost will be about 10 cents when labor is paid at the rate of $3 per day.

not try to do it sitting. Peters gives the following directions for efficiency in spalling, which are expressed so well that I cannot do better than quote his own words,[1] with which my own experience has been entirely in accordance.

"The style of hammer is seldom suited to the purpose, though both the amount of labor accomplished and the personal comfort of the workmen depend more upon the weight and shape of this implement and its handle than on any other single factor, save the quality of the ore itself. There should be several cast-steel sledges, differing in weight from 6 to 14 lb., intended for general use in breaking up the larger fragments of rock to a size suitable for the light spalling-hammers. Each laborer should be provided with a hammer 6 in. in length, forged from a 1·5 in. octagonal bar of the best steel, and weighing about 2·75 lb. This should be somewhat flattened and expanded at the middle third, to give ample room for a handle of sufficient size to prevent frequent breakage. The handles usually sold for this purpose are a constant source of annoyance and expense, being totally unsuited to this peculiar duty. It is better to have the handles made at the works, if it is possible to procure the proper variety of oak, ash, hickory, or best of all, a small tree known in New England as ironwood (hornbeam), which, when peeled and used in its green state, excels most other woods in toughness and elasticity. The handles should be perfectly straight, without crook or twist, so that, when firmly fastened in the eye of the hammer by an iron wedge, the hammer will hang exactly true. Their value and durability depend much upon the skill with which the handles are shaved down to an area less than half their maximum size, beginning at a point about 6 in. above the hammer-head and extending for about 10 in. toward the free extremity. If properly made and of good material, they may be made so small as to appear likely to break at the first blow, but in reality they are so elastic that they act as a spring, and obviate all disagreeable effects of shock, wear longer and do more work than the ordinary handle. Such a handle has lasted five months of constant use in the hands of a careful workman, whereas one of the ordinary make has an average life of scarcely four days, or perhaps 30 tons of ore.

"Where the ore is of fairly uniform character, it is advantageous to adopt the contract system for this kind of work. A skilful laborer, under ordinary conditions, will break seven tons of rock per 10-hour shift to a size of 2·5 in., taking coarse and fine as it comes, and in some cases is also able to assist in screening and loading the ore into cars. The latter operation should be executed with a strong potato-fork having such spaces between

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[1] Op. cit., p. 93.

the tines as to retain the coarsest size, while the finer classes are left upon the ground. The amount of space required for convenient spalling is about 40 sq. ft. per man, which will allow for ore-dumps, tracks, sample boxes, etc. A good light is essential."

CULLING.—The ore having been broken, either by machine or by hand, it must be passed over a screen or grizzly to remove the fines, which will be sent directly to the dressing floors, before going to the sorters. The latter will be probably either men who are incapacitated for other work or boys. In Europe, women and girls are frequently employed for such work, but American ideas would scarcely permit that. Boys become very expert in the work and are satisfied by materially lower wages than men, compared with whom they are equally or more alert and efficient. In sorting ore by hand the success of the operation depends largely upon the convenience of the manner in which the ore is presented to the pickers. This may be done by stationing the latter along a bench on which they may draw from pockets the ore to be sorted; better by discharging the ore on a large, circular or annular table of wood or iron, revolved slowly, around the periphery of which the pickers stand; or better still, by discharging it on a traveling belt from which the pickers can select it.

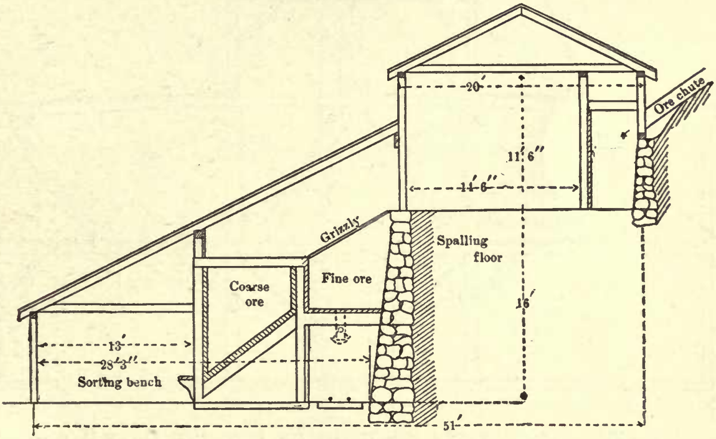


FIG. 19.—TRANSVERSE SECTION OF SPALLING AND SORTING HOUSE.

Stationary Tables.—An arrangement of the first kind at a mine in Mexico for sorting an ore composed of blende, pyrite, galena and quartz, which was to be separated into those four classes, is shown in the accompanying engraving, Fig. 19. The spallers on the upper floor received the ore from the mine cars through a chute. They broke it to 2·5 in. size by means of long hammers and shoveled their product on grizzlies, whence the coarse slid into the pockets designed for it and the fines passed through into a bin, whence they were taken to the jigs. The sorters, seated along a bench in front of the ore pocket, drew ore from holes in the latter and dropped the various kinds through holes in the bench, beneath which sacks were suspended to receive them.

A similar arrangement, but on a larger scale, is to be seen at the dressing works at Clausthal in the Upper Harz, Germany, where the sorting house is somewhat according to the design shown in Fig. 20. At those works the ore is crushed to 60 mm. size and passed over a screen with 32 mm. holes in a separate house. The stuff from 32 to 60 mm. size is trammed to the sorting house, where it is dumped into the ore pockets, the tramway passing over the latter. The ore pockets have a sorting bench on each side. The pickers throw the various kinds of ore into boxes placed conveniently on the floor and the waste into chutes in the floor. The mixed ore which is sorted out is cobbed and resorted at benches along the sides of the house.

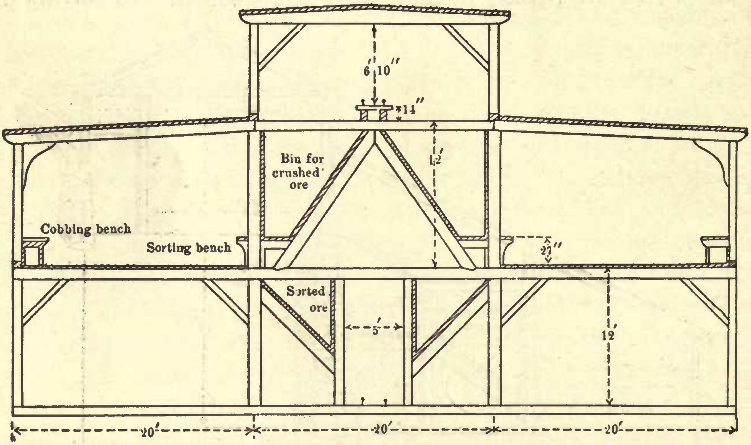
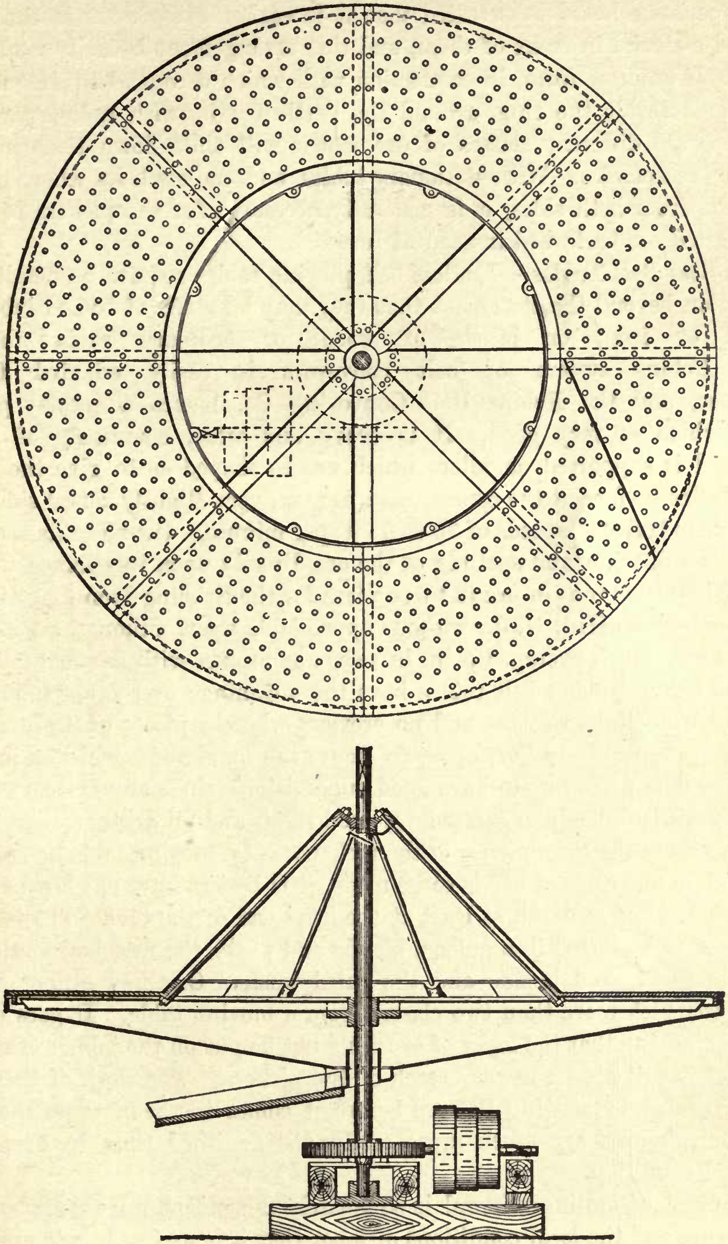


FIG. 20.—TRANSVERSE SECTION OF ORE SORTING HOUSE.

Revolving Tables.—An annular, revolving picking table, as built by Fraser & Chalmers, of Chicago, is shown in Fig. 21. The broken ore is led on the table by a chute, and spreading out is carried by the revolution of the table until, meeting an inclined, stationary scraper, it is swept off into a chute, which delivers it into cars, or a continuous conveyor, to transport it to the next operation. Around the table boys stand, who pick out mineral from the slowly moving layer of ore. The table itself is made of punched iron or steel plate.



FIGS. 21 AND 22.—ORE PICKING TABLE.
Fig. 21: Plan. Fig. 22: Vertical Section.

Experience has demonstrated that it is easiest for pickers to throw the sorted material in front of them, and this arrangement is easily made with tables of annular form, in connection with which a conical surface may be arranged inside the ring, around the vertical axis, with radial partitions to separate different classes of ore which will slide down the cone into proper receptacles. With stationary tables, and endless belts where picking is done from both sides, it is generally necessary for the pickers to throw the sorted ore backward instead of forward.

Endless Belt Tables.—Endless belt picking tables require no detailed description beyond the statement that they may be made of linked tablets of wood or iron, or billets of wood, or ordinary rubber belting. The Robins system of belt conveyors is easily adapted to this purpose, and the Robins Belt Conveying Co. makes a special picking belt, which is very good. It is heavy and wide, commonly 32 to 36 in., and is supported on idlers which are so shaped as to give the belt a broad, flat surface at the center, with narrow, very slightly raised sides. It is made to travel at speeds of 30 to 60 ft. per minute. Owing to its elasticity the belt will withstand spalling of the ore directly upon its surface. A belt carried 300,000 tons of ore in three years for the Stirling Iron and Zinc Co. at North Mine Hill, N. J., without appreciable wear, although a good deal of the ore, which averaged 5 in. in size, was broken with hammers directly on the belt. Rubber belt tables have the advantage over other kinds that there are no links to wear and no crevices wherein pieces of ore can jam.

Efficiency in Hand Sorting.—To secure the maximum efficiency in hand sorting, it is necessary to have good supervision (since any system of contract work is difficult to carry out), good light, and all arrangements which may increase the convenience of the pickers. If a moving table be used and more than one class of ore is to be made, it is best to have one boy assigned to each kind of mineral, at least at the head end of the table. Thus if the ore were to be sorted into galena, blende and waste, the first boy would pick galena, the second blende and the third waste. One boy cannot advantageously pick more than two classes from a moving table. It goes almost without saying that the layer of ore must not be put on the table so that any part of it will not be in easy reach of the pickers. The work of the latter is facilitated by wetting the ore before it comes to them, when the component minerals are much more easily distinguished than in dry, dust-covered condition.

The cost of culling mineral is so entirely dependent upon the character of the ore and the local conditions of labor that it is useless to cite examples from practice; the probable economy must be determined by experiment in

FIGS. 23 AND 24.—ROBINS CONVEYING AND PICKING BELT.
Fig. 23: Installation at plant of Tennessee Copper Co. Isabella, Tenn. Fig. 24: At works of British Columbia Copper Co. Anaconda, B. C.

each particular case. In Europe, especially in some of the older works, culling is carried to a degree of subdivision which would not be done in modern practice; certainly not under American conditions. In the United States a breaking of the ore by spalling, preliminary to culling, would be recommended probably only in exceptional instances; especially in those where the blende is of such occurrence that the saving in mineral by avoiding loss in slimes is more than enough to offset the cost of spalling plus that of culling, inasmuch as the cost of milling the rejected ore per ton would be the same, substantially if not identically, as if all the ore from the mine went directly to the mill.

Under ordinary circumstances culling would be advised probably as a step in the milling process, where all the ore from the mine, having been broken by a crusher to the size determined for the next machine, would pass over a grizzly or through a trommel from which the coarse material would go to the picking table and the rejected stuff from the latter to the next crushing machine. In this case there would be no extra cost for crushing, and only the cost of culling, minus the cost of milling the mineral picked out, would have to be considered against the increased saving of mineral. For example, if the cost of milling be 25c. per ton of crude ore and five tons be concentrated into one, the cost per ton of concentrates is $1·25; leaving out of account the question of losses in treatment (very important) and cost of repairs on picking tables, interest, amortization, etc. (comparatively unimportant), it would be an equal thing to produce a ton of culled mineral of the same grade at $1·50—i.e., the cost of the concentrates in the first case plus the saving of milling one ton of material; the higher saving of mineral would probably permit the hand sorting to be done economically at considerably higher cost than $1·50 per ton of product.

GRAVITY CONCENTRATION.

In concentrating zinc ore, like any other ore, by gravity it must first be crushed sufficiently fine to detach the various component minerals from one another, taking care to crush as little as possible finer than is necessary to effect such a liberation. The crushed ore is then separated into proper sizes, by screens for the coarser and hydraulic classifiers for the finer sizes, and each size is washed separately; the coarser on jigs, whereby the mixture of mineral particles are shaken up in water, with the result that the heavier and more valuable sink to the bottom; and the finer on shaking tables or buddles, whereby the lighter worthless mineral is washed off, while the heavier is discharged by proper devices as a concentrated product. There is always an intermediate product, consisting of interlocked particles of mineral and gangue, which the original crushing failed to separate. This should be crushed more finely by proper machinery to effect the separation, and then should be sized and washed again. The ideal concentrating plant conforms to these conditions and the better the details are worked out the lower will be the cost of dressing and the higher the saving of mineral, although in the best designed works there is inevitably a certain loss of mineral owing to some of it becoming crushed so finely that it is impossible to settle it economically, while there is always some escaping attached to larger or smaller particles of gangue which it does not pay to recrush any further.

CARDINAL PRINCIPLES.—The proper size to which to crush the original ore, the proper sizes into which to divide it by screens, the point at which it is best to discontinue sizing by screens and begin sorting by hydraulic classifiers and many other details, are only to be determined by tests and calculations for each particular ore. However, experience has demonstrated some cardinal principles in ore dressing practice of which the more important may be summarized as follows:

Crushing.—The comminution of the ore should be effected gradually by a series of machines and never attempted with one machine; the finer it is to be crushed, the more should be the members of the series, except that in the case of the ball crushers it is possible to effect the reduction satisfactorily with only two machines, namely a rock-breaker and a ball-mill, regardless of the degree. With rock-breakers and rolls Mr. Philip Argall, who is a high authority on the crushing of ores, considers that it is inadvisable to attempt a greater reduction than one fourth the diameter of the pieces by any one machine. Thus if the pieces were of 16 in. size he would break to 4 in. cubes in the first operation and to 1 in. cubes in the second. If the ore were to be crushed to 0·5 in. size it would be best to make the reduction in the ratios of 16:5 in., 5:1·5 in., 1·5:0·5 in., and so on. Rock-breakers of the Gates and Blake types are the best for preliminary crushing machines, but according to Mr. Argall rolls are preferable when the material is smaller than 2 in. cubes. For fine crushing, taking the product of the breakers, rolls are generally the most efficient type of machine; they can be used in wet crushing, which is invariably done in dressing works, with very good results down to 20-mesh, and with fair results down to 40-mesh, if there be not much clayey matter in the ore. For fine grinding, however, the ball-mill may be superior to rolls under certain circumstances. It produces a product of entirely satisfactory character as to granularity, and has the advantage of combining the grinding, elevating and screening apparatus in one compact machine, which is able to give a finished product from coarse material in one operation.

Screening.—The screening capacity should be ample because the cleanness of separation depends largely upon the perfection of the sizing. There should be two lines of trommels, duplicates of each other, through either of which the stream of ore may be passed at will. By this arrangement the necessity of stopping the whole mill to repair a single screen is avoided. In designing very large mills all the machines may be advantageously arranged as a system of units. The modern tendency in ore dressing is to reduce the number of screen sizes. Screens with apertures smaller than 1 or 1·5 mm. are seldom used, the further grading of the ore particles being effected by hydraulic sorting.

Jigging.—The Harz jig is undoubtedly the most efficient separating machine for the coarser sizes of mineral sand, or down to particles of 1 mm. or 0·04 in. diameter. Jigs of special design are also very efficient separating machines for the finer sands or coarser slimes. For a two mineral separation, e.g., blende and quartz, a three-sieve jig is advisable; for a three mineral separation, e.g., galena, blende and quartz, a four-sieve jig is preferable. Jigs with five, six and even seven compartments are employed advantageously in the peculiar practice of the Joplin district. The improved shaking tables, like the Wilfley, Cammett, and Bartlett, which are developments of the Rittinger side bump table, have recently come into extensive use and give excellent results in separating the finer sizes of ore. They have a wide range of application, and are perhaps the most efficient fine concentrators that have yet been devised.

Slime Washing.—A close saving of values in ore dressing can never be effected without a careful washing of the slimes which are unavoidably produced in the crushing of the ore. These should be sorted by means of an hydraulic classifier and then washed on shaking tables or buddles. The coarser climes are treated advantageously on shaking tables; for the finer, the revolving, convex, circular buddle (round table) is still a standard type of apparatus.[1]

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[1] The principles laid down in the paragraphs on screening, jigging and slime washing have been generally accepted for a long time, but it must be confessed that the wide range of applicability which the improved shaking tables, like the Wilfley, have been demonstrated to possess, has upset the old ideas in some respects. These tables have been proved capable of taking pulp sized only within wide limits, and effecting therefrom not only a cleaner separation but also a higher percentage of recovery than close sizing followed by jigging and washing on the conventional tables and buddles. This has been done among other instances in the treatment of the mixed sulphide ore of Leadville, Colo. The tendency of ore dressing practice in the United States has been for a long time toward reducing the number of screen sizes; it appears now to be

General Arrangement.—As to the general arrangement of an ore dressing works, the consensus of the best engineering opinion at the present time is for a flat site, rather than a hillside, although in choosing the former it is well to take advantage of a natural slope for a tailings dump, if that can be done without incurring disadvantages in operation in other respects. The level mill site is advocated because the original construction is cheaper, no cuts and fills and retaining walls being necessary, a better and more roomy arrangement of the machinery can be had, besides better light, more facility of supervision and greater elasticity in extending the works, while the extra cost of elevating material over what is necessary in a hillside location is insignificant.

SPECIFIC GRAVITY OF ZINC AND ASSOCIATED MINERALS.—In the following table are summarized the specific gravities of the most important zinc minerals and the impurities which generally occur with them, the specific gravity of water being deducted in each case to show their relative weights when immersed in water:

SPECIFIC GRAVITY OF ZINC AND ASSOCIATED MINERALS.

Mineral.	Spec. Gravity.	Relative weight in water.	Mineral.	Spec. Gravity.	Relative weight in water.
Blende	3·90 to 4·10	2·90 to 3·10	Calcite......	2·60 to 2·80	1.60 to 1·80
Smithsonite ...	4·30 to 4·45	3·30 to 3·45	Dolomite....	2·85 to 2·95	1.85 to 1·95
Franklinite....	5·07 to 5·22	4·07 to 4·22	Barite.......	4·30 to 4·70	3.30 to 3·70
Zinkite........	5·43 to 5·70	4·43 to 4·70	Siderite......	3·70 to 3·90	2·70 to 2·90
Hydrozinkite ..	3·58 to 3·80	2·58 to 2·80	Pyrite.......	4·80 to 5·20	3·80 to 4·20
Willemite......	3·89 to 4·18	2·89 to 3·18	Marcasite....	4·65 to 4·90	3·65 to 3·90
Hemimorphite.	3·40 to 3·50	2·40 to 2·50	Limonite....	3·40 to 3·95	2·40 to 2·95
Quartz........	2·50 to 2·80	1·50 to 1·80	Galena......	7·20 to 7·60	6·20 to 6·60
Chalcopyrite...	4·10 to 4·30	3·10 to 3·30	Garnet	3·50 to 4·30	2·50 to 3·30
Fluorite.......	3·10 to 3·20	2·10 to 2·20	Rhodonite ...	3·50 to 3·70	2·50 to 2·70

ORE DRESSING IN UPPER SILESIA.—The concentration of the low grade zinc ores of Upper Silesia is rendered difficult in the case of calamine by the earthy and drusy (porous) character of the zinc minerals and the slight difference between their specific gravities and those of the other minerals associated with them; in the case of the sulphide ore by the common occurrence of marcasite interwoven with the blende. Consequently the larger proportion of the calamine ore produced is concentrated by hand sorting

toward restricting the use of jigs to larger sizes and substituting shaking tables for the work formerly done by the fine jigs and the buddles. This tendency is manifested in its extreme in the new mill of the Federal Lead Co. at Flat River, Mo., which is to have neither jigs nor buddles, but only shaking tables. The conventional ore dressing practice of the Joplin district, which is described further on in this chapter, also affords a striking example of how an ore (in that case a comparatively easy ore, to be sure) can be jigged successfully without a preliminary close classification by screening. It is of course desirable to dispense with screening as much as possible.

and a good deal of the blende is handled in the same manner. There are certain mines, however, which mill all the ore raised. That is done at the Neue Helene mine, at Scharley, which is the largest producer in the region, and is equipped with what is probably the best dressing works.

The Neue Helene dressing works consists of two divisions, one for blende and one for calamine, the ore being divided in the mine into those classes. Both mills are situated close to the main hoisting shaft of the mine, so that the mine cars are raised directly to the highest level of the mills, to which they are trammed over a short bridge. The transverse section of the blende

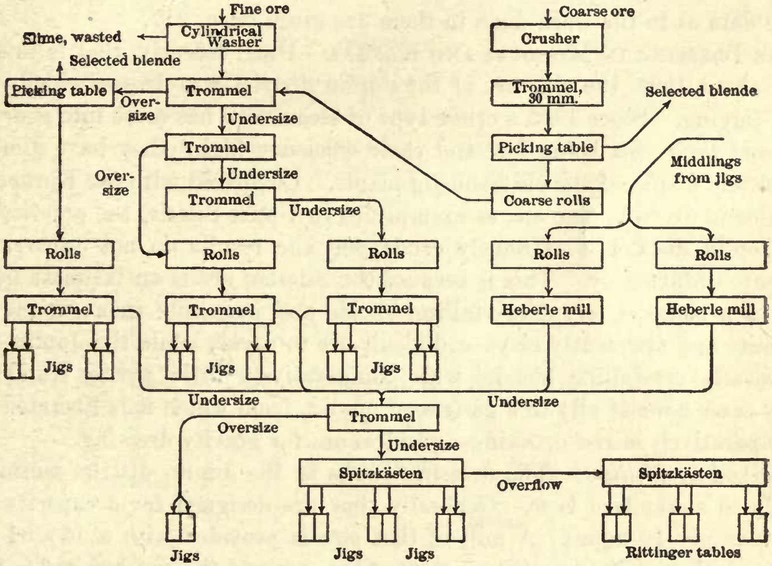

FIG. 25.—DIAGRAM OF ORE DRESSING PROCESS AT THE NEUE HELENE BLENDE MILL, UPPER SILESIA.

mill shows three steps, with the first crusher on the highest and the slime washers on the lowest; on the middle step the mill has three floors, the sand jigs being placed on the lowest, the fine crushing rolls and Heberle mills on the second floor, and the picking tables on the third floor, with the coarse crushing rolls and the first train of trommels on framework higher up, but below the level of the first crusher on the highest step. The mill is therefore arranged on five levels. The general scheme of the ore dressing system is shown in the accompanying diagram, Fig. 25.

The crushing rolls are all geared and of slow speed, the first set of the series being 900 mm. in diameter. The jigs are built entirely of iron and steel and have semi-cylindrical bottoms. The Rittinger tables are double, and also are built of iron and steel. The calamine mill is of similar design, but it has no Heberle mills, and small "Stossherds" are substituted for the Rittinger tables of the blende mill. The work in both mills is done largely by women, of whom there is an army, who in 1893 were paid from 0·9 marks (21·4c.) for attending the jigs, to 1·2 marks (28·6c.) for tramming, 12-hour shifts. A small quantity of galena concentrate is produced, besides the blende. The works are driven by a 180 h. p. engine. They were built in 1880 and were the first large dressing works erected in Upper Silesia. Some data as to the work done in them are given on p. 217.

ORE DRESSING IN MISSOURI AND KANSAS.—Until recently, that is to say until about 1895, the zinc ore of the Joplin district was dressed chiefly by hand-jigging. Since 1895 a crude type of steam mill has come into general use, and their cost being low and their efficiency high,[1] they have almost completely displaced the old hand-jig plants. Compared with the European practice in dressing zinc ore, as exemplified in Upper Silesia, the practice in the Joplin district is extremely crude, but the results do not *apparently* compare unfavorably. This is because the Silesian ore is an intimate mixture of a compact, cryptocrystalline blende and marcasite in a gangue of dolomite and frequently clay—a difficult ore to dress; while the Joplin ore is coarsely crystalline blende, with comparatively little pyrites (and in many cases none at all) in a gangue of quartz, from which it is liberated by a comparatively coarse crushing—an ideal ore for gravity dressing.

Method of Milling.—The dressing works in the Joplin district conform closely to a standard type. Generally they are designed for a capacity of 100 tons per 10 hours. A mill of that size is provided with a 16×24 in. Blake crusher, delivering into a storage bin, whence the crushed ore is fed mechanically to a set of 12×36 in. rolls, driven by belts at 35 r. p. m. The product of the rolls is elevated and discharged into a 36×84 in. trommel, covered with sheet steel punched with 0·5 in. round holes, which makes 20 r. p. m. The oversize from this trommel passes to another set of 12×36 in. rolls, and having been crushed finer by them, it is delivered back to elevator No. 1 and trommel No. 1. The product which passes through the 0·5 in. holes of trommel No. 1 goes to trommel No. 2, which is also 36×84 in., but is covered with plate punched with 0·125 in. holes. The oversize from trommel No. 2 goes to the roughing jig; the undersize to the sand jig. There is also a cleaning jig which reworks certain products of the roughing jig.

The jigs are of the Cooley pattern, in which the box is built up with 2×4

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[1] This word is used with qualifications which will be explained further on.

in. scantling laid together on the wider sides and spiked and bolted through, the sides of the box being thus made 4 in. thick. This has been proved to make a very solid and watertight jig box, which is excellent in all respects. The roughing jig has five compartments, each 30×42 in., with trays of 4-mesh steel wire cloth; the counter shaft is driven at 140 to 175 r. p. m. The cleaning jig has six compartments, 30×36 in. (sometimes 26×42 in.), with trays of 4- and 6-mesh brass wire cloth; the counter shaft is driven at 170 to 225 r. p. m. The sand jig has four or six compartments with 24×36 in. or 25×30 in. trays of 12- and 14-mesh brass wire cloth; the counter shaft is driven 200 to 260 r. p. m.

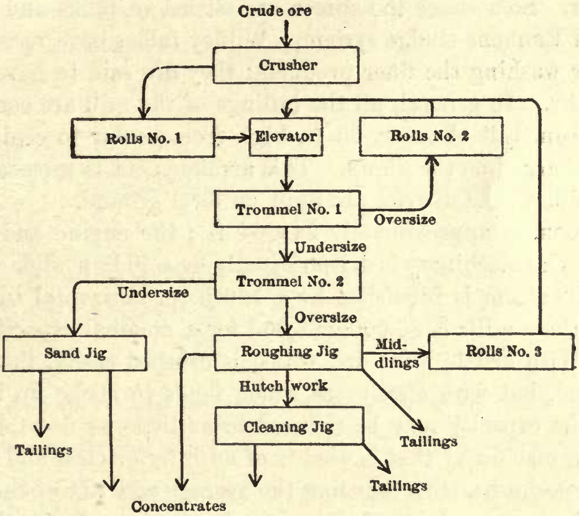

FIG. 26.—DIAGRAM OF THE ORE DRESSING PROCESS EMPLOYED IN THE JOPLIN DISTRICT, MO.

The roughing jig, which receives material finer than 0·5 in. and coarser than 0·125 in., furnishes a clean concentrate from above the sieves of the first two compartments. The third, fourth and fifth compartments yield a mixed product, which is conveyed to a third set of 12×36 in. rolls, set to crush fine. The product of these rolls is united with those of rolls Nos. 1 and 2, and is passed again through the screens. The concentrate made through the sieve of each compartment, which ranges in size from 0·145 to 0·125 in., is delivered by means of a belt elevator to the cleaning jig. The latter makes a clean concentrate both above and below the sieves in three or four compartments. The mixed product from above the sieves of the other compartments is recrushed by rolls No. 3 and is returned to the system through the trommels; the hutch-work is delivered to the sand jig. The sand jig also receives the undersize from screen No. 2. It makes clean heads and clean tails. The above scheme is illustrated in the accompanying diagram, Fig. 26.

The variations from the system described above are slight and exceptional. In some mills the rolls are smaller than 12×36 in. (14×24 in. being a common size), and in some the third set is omitted. Some mills have only one sizing trommel and some have three, one with 0·375 in. holes being interpolated between 0·5 and 0·125 in. In exceptional cases the finest product is sorted by means of a pointed box, of which the overflow is washed on a Frue vanner. Sometimes the slimes are settled in tanks and reworked by the so-called Emmons sludge system. Wilfley tables have recently been introduced for washing the finer products; they are said to have given satisfactory results. In general, all the tailings of the mill are conveyed to the boot of a 16-in. belt elevator, 50 ft. high from center to center, by which they are discharged on the dump. This arrangement is necessary inasmuch as all the mills of the district are built on level ground.

The jig room is approximately 24×64 ft.; the engine and boiler room 30×28 ft. The machinery is driven usually by a 60 h. p. slide valve engine, for which the steam is furnished by a 100 h. p. horizontal tubular boiler. The cost of these mills is $6,000@$8,000 for a nominal capacity of 10 tons per hour. With brittle, flint ore, which is crushed easily, that capacity is often attained, but with clayey ore which tends to choke up the crushing machinery the capacity may be reduced to as little as five tons per hour. The crushing machinery that is used is of an inferior class and is subject to frequent break-downs, thus reducing the average capacity of the mill. The regular mill crew comprises four men, viz., two crusher-feeders, one jig man and an engine tender.

Cost of Treatment.—The objects chiefly aimed at in the existing ore dressing practice in the Joplin district are (1) to put through the maximum quantity of ore per day that a unit crew of men can handle, reducing the cost of dressing per ton of crude ore to the minimum, and (2) the production of a concentrate which will meet the market requirements and fetch the best price. In the attainment of those objects, ore is crushed and jigged in the largest and best works for as low as 20c. per ton (mill men being paid $2·00@$2·25 per day and coal costing $1·40@$1·60 per ton for mine run), while a concentrated product assaying from 57 to 65% Zn, the latter being practically pure blende, is obtained.

The cost of ore dressing in the Joplin district varies between wide limits, not only among various mills, but also in the same mill. The variation between different mills of the same design and efficiency is due to differences in the character of the ore milled. The variations in the same mill are ascribable largely to the inferior class of crushing machinery which is employed. Break-downs are frequent, leading to large bills for repairs and renewals, but more important than that to great losses of time. This is shown in the records of single mills, wherein the cost may be as low as 20c. per ton of crude ore in one month and 40c. per ton in the next month. For this reason it is quite unsafe to base comparisons on the results of a single month.

The cost of dressing for six consecutive months in three different mills, for which a careful record was kept, was as follows:

MILL No. 1				MILL No. 2				MILL No. 3			
Tons ore milled	Tons concentrate produced	%	Cost per ton of ore milled	Tons ore milled	Tons concentrate produced	%	Cost per ton of ore milled	Tons ore milled	Tons concentrate produced	%	Cost per ton of ore milled
3,461	231	6·7	$0·550	3,685	130	3·5	$0·28	3,431	213	6·2	$0·44
3,985	313	7·9	0·245	3,413	202	5·9	0·26	4,251	165	3·9	0·44
4,914	259	5·3	0·190	3,812	186	4·9	0·29	8,410	430	5·1	0·37
5,710	259	4·5	0·200	4,196	211	5·0	0·24	9,904	268	2·7	0·29
5,130	229	4·5	0·180	2,695	147	5·5	0·25	3,607	167	4·6	0·44
....	....	....	......	2,295	135	6·0	0·32	5,283	284	5·4	0·38
[a]4,640	258	5·6	0·270	3,349	169	5·0	0·27	5,814	254	4·4	0·39

[a] This line gives the averages for the six months, except in the case of Mill No. 1, which was idle during the last month of the period.

Loss of Mineral.—The production of a concentrate assaying 63% Zn and even one of 60% Zn, which is thought to be approximately the average of the entire district, is certainly successful work, but it is accomplished at the cost of the mineral in the ore. The mills of the district are characterized by the absence of means for treating slimes[1] and large losses are experienced naturally on that account, though since the ore does not require very fine crushing the losses of slime are less than would be the case otherwise, while it is an established fact in ore dressing that the higher the degree of concentration the higher are the losses of mineral.

As to what the losses in dressing experienced in the Joplin district actually are, it is difficult to conclude, there being few mills at which the concentrates are assayed for zinc, or even for their moisture content when weighed, and

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[1] This has lately been rectified in some of the better mills, wherein vanners and shaking tables have been introduced.

fewer still, perhaps it is safe to say none at all, where the crude ore put through them is weighed and assayed. In the absence of these data it is impossible to determine accurately the percentage of mineral that is lost. Carl Henrich, in 1892, discussed the subject in the following words:[1]

"When we look at the tailing piles, and see the vast amount of good blende left in them; when we follow the course of the water flowing from these concentration works into larger streams, and see everywhere the fine blende shining golden yellow in a bright sunlight, we begin to doubt the success of a method of concentration, which on an average wastes certainly not less than one-third, and more frequently nearly one-half, of the ore contained originally in the crude material treated. No doubt this assertion of losses will be denounced as an exaggeration by most practical miners in Webb City. But I believe a fairly conducted test will demonstrate that no concentrating works near Webb City, as at present conducted, recovers more that two-thirds (probably much less) of the blende contained in the original mine-stuff hoisted to the surface." Henrich's conclusions in 1892 have been borne out by the considerable quantity of mineral which has been recovered from accumulations of old tailings by the operators of "sludge mills."

The percentage of mineral recovered from the ore milled in the Joplin district varies considerably according to the character of the ore, which differs more or less in various parts of the district. Some tests made recently have indicated that 80% of the mineral in certain docile ores is recovered, while in the case of certain refractory ores the recovery may not be more than 50%. It is conservative to conclude that a recovery of 70% represents good average work in ore dressing in the Joplin district at the present time. There is little probability that the figure is any higher, and it may be no more than 66⅔%. It is proper to remark in this connection that a mineral recovery of 80% is high in dressing any kind of ore in a perfectly designed mill, and if it be possible to save as much as 70% by the Joplin practice it is indicative chiefly of the docility of the ore.[2]

[1] "Zinc Blende Mines and Mining near Webb City, Mo.," Trans. Am. Inst. Min. Eng., XXI, 23.

[2] A saving of 96% is extraordinarily high, but is sometimes attained under favorable conditions. According to a paper in the Eng. & Min. Journ. of July 20, 1901, careful tests at the new mill of the Moctezuma Copper Co. at Nacosari, Sonora, Mexico, showed a recovery of 86% of the mineral in the first 16,000 tons of ore milled, while subsequent improvements raised the recovery to 90%. The mill is apparently designed upon the most approved principles and is well equipped with machinery for carefully sizing the ore, recrushing middlings and washing the slimes. The coarse sand is washed on jigs, the medium on Bartlett tables (a modification of the Wilfley), and the fine on vanners. The ore is chalcopyrite (sp. gr. 4·2) in a gangue of rhyolite (sp. gr. 2·5). The high saving of mineral is "due largely to the fact that the sulphide is solid and coarse and does not permeate the solid fragments of the gangue." Another example of an apparent-

It is not to be thought that the Joplin miners and mill men are not aware that they are losing a great deal of ore value that might be saved. On the contrary, they are quite alive to that fact, but the continuance of the milling practice, which Henrich and many others after him have criticized so severely, is due to the conclusion that it is the most profitable. This conclusion may be incorrect, but it should not be lightly rejected, as many innovators with preconceived opinions as to the crudity of the Joplin method have learned in a costly manner. The present practice is the concrete result of the experience of many men extending over many years, which has naturally evolved a system that conforms to the peculiar conditions of the district. Further improvements will be made, without doubt, but they are much more likely to originate in the district than outside of it.

A modification of the method which will afford an increased saving of the mineral of the ore milled is one of the improvements that are likely to be made in the future. If a mill be operated on ore which yields a concentrate of 5%, without saving anything from slimes, and the coarse tailings assay 2 to 2·5% mineral;[1] if the slimes can be impounded and reworked profitably by tributers, and if after an accumulation of tailings has been made they can be profitably remilled by men who make a specialty of that business;[2] if those things can be done (they are done, in fact), it would appear best to do them in connection with the original handling of the material and deposit the waste products in a final resting place. Rational as this proposal appears to be, the reasons why it has not been done, and why perhaps it will never be done, are not difficult to perceive. The Joplin mill men as a class are not familiar with slime dressing. The ore bodies of the district are of variable magnitude and frequently are exhausted quickly. The method of mining does not open reserves of ore, and the life of a mine is uncertain. The margin over the cost of mining and milling is likely anyway to be narrow. The capital of the men exploiting the mines is generally small. All of these conditions tend to limit the expenditure for original plant to the least that will enable the mine to be worked. In order to meet that requirement a type of mill has been evolved which costs only $30@$40 per ton of daily capacity (as compared with $150@$200 for

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ly easy two mineral separation is afforded by the lead ore of the Flat River and Bonne Terre districts in St. Francois Co., Mo., where galena (sp. gr. 7·4) occurs in a gangue of dolomite (sp. gr. 2·9). The mills are of large capacity and well equipped, but only about 72% of the mineral in the ore is recovered. The fact that the recovery is no higher is due to the fine dissemination of the galena through the gangue, while the brittleness of galena causes it to go largely into the slime.

[1] The tailings made in the district are sometimes as low as 1% Zn, but sometimes are as high as 3%; much depends upon the character of the ore.

[2] Under favorable conditions tailings yielding as little as 1·5% mineral have been reworked profitably.

an ordinary mill in other districts).[1] The design has become well established, one mill resembling 90% of all the others in the district like peas in a pod, and the developer of a new mine can contract for the erection of a mill as easily as he can for a barn and with as little concern as to its promised efficiency in operation. The addition of recrushing and slime washing machinery would introduce unfamiliar complications and would increase the first cost. The complications might be learned, of course, but the increased cost would always raise doubts as to the advisability of the additional investment until the mine had been tested. Hence it is that the reworking of tailings has been left until a sufficient quantity has accumulated to make it worth while.

Total Cost of Production.—The cost of dressing in the general run of the plants in the Joplin district probably averages about 30c. per ton of crude ore.[2] The cost of mining is, of course, variable. The general average for six months of five mines which produced about 120,000 tons of crude ore, yielding about 6,000 tons of concentrates, was 70c. per ton of crude ore, the extremes being 56c. and 94c. General expense and amortization of plant were not included, however. Frank Nicholson states[3] that mining has been done as low as 30c. per ton in certain of the properties under his management, while in others the cost has been $1·00. The mean between those extremes is 65c. All of the figures quoted above refer to the operations of large companies, which are subject to a general expense of perhaps 10c. per ton of crude ore over and above the cost of mining and milling.

The minimum and average cost of mining and milling 100 tons of ore in the Joplin district are approximately as follows:

Item	Minimum.	Average.
Mining	$30·00	$70·00
Milling	25·00	30·00
General expense	10·00	10·00
Total	$65·00	$110·00

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[1] These figures refer to the daily capacity on the basis of double shifts; the Joplin 100-ton mill is run only one shift.

[2] I am aware that a lower figure is sometimes stated. For example, Eric Hedburg said (in a paper read before the American Institute of Mining Engineers, Richmond meeting, February, 1901) that the cost of milling 100 tons per day would be $22·75=22·75 c. per ton. That figure, and even lower ones, may be attained as the average of a single month, but a yearly average would not be so low. Hedburg estimated the cost of hand-jigging at 56c. per ton, reckoning two men at grizzlies ($3); four men jigging ($8); two men jigging ($5); and one man handling waste rock ($1.50); total, $17.50 for the concentration of 30 tons of crude ore per day of nine hours. The substitution of steam-mills for hand jigs was one of the important economies introduced in the Joplin district between 1891 and 1900.

[3] The Mineral Industry, VIII, 677.

The cost of production per ton of mineral from ore yielding 3 to 8% of concentrate is consequently more or less as follows:

Yield	Ratio	Minimum	Average	Yield	Ratio	Minimum	Average
3%	33⅓:1	$21·66	$36·66	6%	16⅔:1	$10·83	$18·33
4%	25:1	16·25	27·50	7%	14$\frac{3}{10}$:1	9·29	15·73
5%	20·1	13·00	22·00	8%	12½:1	8·12	13·75

The ultimate cost of production is greater than is indicated by the above figures, because of the charges which must be made for interest on the capital invested and amortization of the cost of plant and land, exploration and development work, etc.[1] Under the leasing system the charges on the land and some of the dead work are covered by the royalty of 15 to 25%, which is deducted from the proceeds of the ore as the share of the land owner and prime lessee—i.e., if mineral sells for $25 per ton the miner realizes $21·25@$18·75. Besides his product of zinc ore, however, he recovers as a by-product, so to speak, a certain quantity of lead ore, worth $40@$48 per ton, but subject to a royalty ranging up to 33⅓%. The proportion of lead ore to zinc ore varies in different mines, but the average for the whole district is approximately 1:10; the statistics as to this have been given on page 188.

It is impossible to arrive at any reliable conclusion as to the average grade of the ore now mined in the Joplin district, but it is probably in the neighborhood of 4·3% Zn=6·4% blende, corresponding to a yield of 5% of mineral assaying 60% Zn on the assumption that 70% of the content of the ore be recovered. This is indicated by the fact that the production of the district, which was considerably stimulated by the high prices for ore in 1899, was not materially restricted by the decline in price in 1900, and actually increased in 1901, when 60% ore sold at $24@$25. In the early part of the decade 1891-1900 the average grade of the ore mined must have been higher, inasmuch as there was a steadily increasing production on a market which was generally below $24.

Relation Between Ore Dressing and Smelting.—In dealing with a zinc ore which requires a preliminary mechanical concentration before smelting, it is important to adjust the method of dressing with respect to the conditions of smelting in such a way that the margin between the value of the ultimate products and the total cost of production will be the maximum possible. This is especially important where the same company operates

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[1] The cost of opening a mine in the Joplin district and equipping it with the necessary machinery, concentrating mill, etc., is generally about $10,000@$15,000.

both mines and smelting works, and is therefore relieved of the necessity of conforming to the more or less arbitrary requirements of independent smelters. In any case, the smelting practice is the less capable of modification, wherefore the practice in dressing should be made to conform thereto. As to how that may be done will depend of course upon the ore. With many ores of a difficult character there is not much leeway for modification in the dressing practice; with some ores of an easy character, like those of the Joplin district, for example, there is a wide latitude. The question to be settled, then, is as to whether the lower cost of smelting a high grade ore is offset by the increased loss of metal in making a high grade ore, or vice-versa.

In the Joplin district at the present time the ore is dressed to a product of 60% Zn—i.e., the concentrate must consist of at least 90% of mineral, blende. The loss of mineral in tailings and slimes is naturally higher in making a concentrate of that grade than in making one containing only 45% Zn or 67·5% of blende. For the purpose of illustration, let it be assumed that in making a 60% concentrate out of Joplin ore assaying 5% Zn, there will be a loss of 30% of zinc in dressing, while in making a 45% concentrate out of the same ore there will be a loss of only 20%. On this assumption, 100 tons of ore raised from the mine would contain five tons of zinc, equivalent approximately to 7·5 tons of blende. If 30% be lost in dressing, there remains 3·5 tons of zinc, equivalent to 5·25 tons of blende, to go into the concentrate; if the concentrate assays 60% Zn, or 90% blende, there must be 5·86 tons of concentrate. If, on the other hand, only 20% be lost in dressing, there will be four tons of zinc, equivalent to six tons of blende, recovered, and if the concentrate assay 45% Zn, or 67·5% blende, its weight must be 8·89 tons. The only extra cost in the mill is that of moving 3·03 tons of mineral to the storage bins and loading the increased quantity on board cars. This would not amount to much, but in the transportation and smelting of the ore the costs would count up rapidly on the greater quantity of valueless material contained in the ore.

Assuming a freight rate of $1, a smelting cost of $9·50 and a metal recovery of 86% in each case, the results would be as computed in the subjoined table, if spelter were worth 4c. per lb.; from which it appears that an additional saving of 10% mineral at the expense of making a concentrate only three fourths as rich would lead to a gain of only about 2c. per ton of crude ore, which gain would increase as the price of spelter rose or the cost of smelting diminished. Practically, however, the cost of smelting the low grade ore would be higher per ton than for the high grade ore, and the percentage of metal recovered would be less, so there probably would be no gain at all. The table referred to is as follows:

Weight of Ore and Assay in Zinc.	5·86 tons. 60% Zn.	8·89 tons. 45% Zn.
Carting and loading on cars @ 25c.	$ 1·47	$ 2·22
Railway freight @ $1.00	5·86	8·89
Smelting @ 9.50	55·67	84·46
Totals	$63·00	95·57
Tons of zinc in ore	3·50	4·00
Per cent recovered in smelting	86·00	86·00
Tons zinc recovered	3·01	3·44
Value @ $80 per ton	240·80	275·20
Cost of smelting and freight	63·00	95·57
Margin	177·80	179·63

It is evident therefore that an increase of only 10 units in the recovery of mineral in the dressing works would not pay for the extra cost of smelting the greater bulk of ore, even in Kansas, where the cost of carriage of ore to the smelter is comparatively low. If the ore had to be carried a long distance the comparison would be still more unfavorable to the lower grade. In other words, it may be more profitable to throw away a considerable percentage of mineral than to stand the cost of smelting an increased quantity of lower grade ore.

The figures cited above are used to illustrate the method of calculation and are not intended to indicate any actual results. They call attention, however, to the advisability of determining the efficiency of the dressing practice with the particular kind of ore which is being concentrated. If the percentage of mineral recovery can be increased without reducing the grade of the concentrate there will be of course a clear gain of the extra value of the mineral recovered, less the additional cost of dressing. For example, if 100 tons of ore assaying 5% Zn yield 5·86 tons of concentrates assaying 60% Zn (70% recovery) and selling for $25 per ton, the value of a ton of the crude ore would be $1·465; if by an additional milling process 0·8 ton more of 60% ore could be produced, raising the percentage of recovery to 80, the value of a ton of crude ore would be increased to $1·667.

Separation of Blende and Pyrites.—Owing to the proximities of the specific gravities of blende (3·9 to 4·1) and pyrite (4·8 to 5·2) a simple separation by jigging can seldom be made very successfully. Blende and marcasite are even more difficult to separate than blende and pyrite, the specific gravity of marcasite being only 4·8. Sometimes, however, an advantage can be gained by roasting a mixed ore sufficiently to make the pyrites less dense or otherwise alter the physical conditions of the two minerals.

A process of this nature has been employed by the Société Anonyme de la Vieille Montagne at Ammeberg, Sweden, for nearly 40 years.[1] The raw ore after hand sorting is slightly roasted in shaft furnaces, in which the ore is charged with coal in alternate layers. The blende remains unaltered, while the pyrite crumbles, becomes spongy and porous and is easily washed off by water in the subsequent treatment. A similar process was introduced at the works of the Matthiessen & Hegeler Zinc Co. at Lasalle, Ill., when a mechanical concentrating plant was operated there. About 1891, W. P. Blake inaugurated the system in the Helena mill of the Wisconsin Lead and Zinc Co., three miles west of Shullsburgh, Wis., and it is now practiced on a small scale in the Joplin district in the treatment of ores high in pyrite.

Practice in Wisconsin.—Blake described the practice in Wisconsin in a paper in *Trans. Am. Inst. Eng.,* vol. XXII. The sulphide ore of Wisconsin contains a good deal of marcasite, together with some galena, mixed with the blende. It was roasted in a special furnace, which had a circular, revolving hearth, arranged in the form of a series of concentric annular steps, on which the ore was stirred and moved forward mechanically by rakes fixed in the dome-shape roof of the furnace. By the use of preheated air for oxidation and the close control of the roasting, which was possible thereby, together with the type of furnace, Blake succeeded in effecting a complete desulphurization of the marcasite without changing the zinc sulphide or even sintering the galena, the blende and galena being drawn from the furnace with scarcely a tarnish on the cleavage surfaces of the crystals.

The roasting which accomplished that remarkable result was done at a dull red heat; so dull that no incandescence was visible by daylight.[2] With mineral consisting of about equal quantities of marcasite and blende, of the size of coarse sand or grains of wheat, about 20 tons (40,000 lb.) could be put through a furnace 16 ft. in diameter in 24 hours; with mineral in which the marcasite was of 0·5 in. size about 10 tons (20,000 lb.) could be done; the most desirable mineral was of 0·25 in. size. The roasting caused the fragments of marcasite to swell, expand, crack open and exfoliate in a peculiar manner, while there was only a slight breaking up of the fragments of blende by decrepitation. Jigging yielded a clean, marketable blende as a concentrate and oxides of iron as tailings. Galena was obtained in the first compartments of the jigs, and it was found that the heating in the furnace caused attached particles of blende and galena to split apart, so that a better separation of those minerals could be effected.

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[1] P. G. Lidner, "Ore Dressing in Sweden," Trans. Am. Inst. Min. Eng., XXIV, 490.

[2] W. P. Blake, Trans. Am. Inst. Min. Eng., XXI, 948.

In carrying out the process the ore from the mines was first crushed and jigged for a product containing about 25% blende, 25% marcasite, and 5 to 10% galena, the remainder being dolomite and flint. This concentrate was dried, roasted as described above, and rejigged, affording a product of which the best samples assayed 62% Zn, less than 3% Fe and less than 1% Pb. According to Blake the success of the process depended upon a complete and even roasting. Every particle of marcasite must be decomposed. Even a kernel of unchanged marcasite would cause it to remain with the blende. The outer coating of oxidized iron might be broken away and washed off, but the kernel of unaltered marcasite would not go over.

Practice at Iserlohn.—An analogous process was once employed at Iserlohn, in Westphalia, but there the roasting was carried far enough to oxidize nearly all of the zinc before the product was subjected to jigging. In executing the process it was found that about 3% S had to be left in the ore, in order to preserve a certain coherence in the zinc oxide. The scheme was suggested by the finely crystalline and closely banded structure of the component minerals, which would have necessitated a very fine crushing in order to liberate them. By the action of heat, however, the ore was broken along the planes of banding, and the roasted charge was obtained largely in the form of small tabular pieces. In washing the loss of zinc was heavy, as would be expected, and the process was hardly a success, although it was said that careful jigging yielded a fair separation of the ore into marketable products and the process was considered profitable at the time it was practised.

SEPARATION OF BLENDE FROM OTHER MINERALS BY SIFTING.

A separation of blende from other minerals can sometimes be effected by sifting, advantage being taken of their difference in brittleness or friability, or some other physical property, such as decrepitation when heated, to create a difference in the average size of the grains of the various minerals.

Practice at Lintorf.—A dry method of mechanically separating blende and pyrites was formerly employed at the Lintorf mines in Rhenish Prussia. The mixed concentrate of those minerals obtained from the jigs was dried and passed through a Vapart mill, in which the particles are pulverized by centrifugal force against the circular walls of the machine, dropping thence through a chute into an ordinary sizing trommel. The speed of the mill—usually about 350 r. p. m.—was regulated by experiment so as to develop a velocity in the movement of the particles sufficient to break up by impact the blende, but not the pyrites. The two minerals could then be separated by sifting, a middling product being returned to the Vapart mill. The capacity of the mill was 2·5 to 3 tons per hour. In this manner it was possible to produce a blende concentrate assaying 50% Zn. It was found that for satisfactory work the material fed into the mill should be of at least 2·5 to 3 mm. size and the best results were obtainable with 5 to 6 mm. stuff. The process was due to F. Büttgenbach.[1]

Practice at Oberlahnstein.—The gravity separation of blende and siderite presents even greater difficulty than the separation of blende and pyrites, inasmuch as the two minerals are of almost identical specific gravity, that of blende being 3·9 to 4·1 and that of siderite, 3·7 to 3·9. At Oberlahnstein, on the Rhine, Germany, a mixture of those minerals was formerly separated by heating to redness and throwing into water, whereby the siderite was thoroughly disintegrated into small particles, which could be easily removed from the blende by sifting.[2]

Heusschen Process.—The difference in the decrepitation of various minerals constitutes the principle of the Heusschen process, which was described in the *Comptes Rendus Mensuels* of the Société de l'Industrie Minérale for 1894, p. 98. It is assumed that cleavable minerals are more likely to decrepitate than massive and that each mineral will have its own temperature of maximum decrepitation, whence mixed minerals of the same size, if treated for the decrepitation of one of them, may be separated by sifting. To accomplish that, a metal table is suspended at an inclination of about 12° over a grate fire. The table is jarred lengthwise by a cam, which causes the mineral to travel downward over the end, whence it goes to a screen for separation. In the experiments a furnace was used, which, in 10 hours, put through (at a temperature of 250° C.) 1,526 kg. of blende-galena-barite-schist ore of 1 to 3 mm. size, assaying 8% Pb and 23·3% Zn, yielding a product of 1,005 kg., which assayed 16·4% Pb and 29·9% Zn; of the barite present in the original ore 31% went into the tailings, which assayed only 2% of the metals.

MAGNETIC SEPARATION.

The separation of blende and calamine from other minerals, irrespective of specific gravity, is accomplished with great success by the aid of magnetism; since the discovery of Wetherill, this has been done even with minerals which display no marked magnetic properties and cannot be converted into magnetic forms, for example, certain kinds of blende can be separated thus from both pyrite and galena. The mixed sulphide ore of Broken Hill,

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[1] School of Mines Quarterly, III, p. 55.

[2] Kunhardt, Ore Dressing in Europe, p. 103.

N. S. W., and of Leadville and Kokomo, Colo., has been treated successfully in that manner. Both the Wetherill and the simple magnetic processes of separation have undoubtedly a great future, although inasmuch as in plant and performance they are more costly than ordinary gravity separation they will be probably employed as an accessory to the latter process rather than as a substiute for it. For convenience, this subject may be considered under the captions "Separation of Strongly Magnetic Minerals" and "Separation of Feebly Magnetic Minerals."

Separation of Strongly Magnetic Minerals.

Iron sulphide, carbonate and sesquioxide may under certain conditions be converted into the magnetic oxide, Fe_3O_4, in which form it is easily attracted by the magnet and thereby may be separated from compounds of zinc, all of which are non-magnetic, or at least only feebly magnetic. Processes depending upon that principle were first applied many years ago, perhaps as early as 1855. Similarly, iron bisulphide may be changed into magnetic sulphide. The conversion of non-magnetic iron minerals into the magnetic form requires considerable care and many failures have been due undoubtedly to ignorance of the precise conditions.

Conversion of Iron Bisulphide into Magnetic Sulphide.—The reduction of iron bisulphide to the magnetic subsulphide has been suggested by Eustis and Howe among others,[1] but experiment has shown this to be a very delicate operation and it has never yet been put in practice on a working scale. It is effected by moderately heating pyrite, when the change expressed by the following equation takes place:

$$3FeS_2=Fe_3S_4+2S$$

G. M. Gouyard, of Denver, Colo., has published[2] results of experiments on this subject, showing that in roasting a mixture of pyrite, blende and galena for magnetic sulphide of iron the magnetic concentrates run lower in lead and zinc than when magnetic oxide is produced.

Conversion of Ferric Oxide into Magnetic Oxide.—The production of magnetic oxide of iron, Fe_3O_4, is effected by carbon according to the following reactions:

$$6Fe_2O_3+C=4Fe_3O_4+CO_2$$

or the ferric oxide may be reduced first to ferrous oxide, and the latter converted to magnetic oxide by combination with an atom of oxygen from

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[1] Trans. Am. Inst. Min. Eng., X, 305.

[2] Proc. Colorado Scientific Society, 1897.

the air, which is known to take place when FeO is heated to redness in the air, the reactions being expressed thus:[1]

$$2Fe_2O_3+C=4FeO+CO_2$$
$$3FeO+O=Fe_3O_4$$

When siderite is calcined it is decomposed according to the reaction,

$$FeCO_3=FeO+CO_2$$

and the molecule of FeO is subsequently converted to Fe_3O_4 by taking oxygen from the air. The roasted carbonate is always strongly magnetic, but in execution of the process the temperature must be regulated carefully to avoid sintering the ore, which because of the fusibility of ferrous oxide and silica may easily happen. According to Le Chatelier[2] the decomposition of ferrous carbonate takes place at 800° C.

The bisulphide of iron may be changed into the magnetic oxide if it be roasted carefully at dull red heat. Practically, however, it is difficult to obtain all of the iron in that form and it is generally necessary after the sulphur has been removed to introduce some carbonaceous matter so as to reduce the ferric oxide to magnetic oxide. When the iron is present originally as ferric oxide, as in limonite ore, a similar reduction by carbon is necessary. Siderite on the other hand is converted into magnetic oxide by a simple heating. The last process, not requiring so much delicacy in manipulation,[3] has found more general application.

Practice at Monteponi.—At Monteponi, Sardinia, an ore consisting of hemimorphite and limonite, with a gangue of dolomite and clay, is separated magnetically in a plant designed by E. Ferraris, director-general of the Societá di Monteponi. The ore is first roasted in a revolving cylinder furnace with the addition of 2 to 3% coal slack to reduce the ferric oxide to magnetic oxide. The product after cooling is elevated to a trommel, which separates it into six classes of 1, 2, 3, 4, 5 and 6 mm, size, respectively, which are delivered to corresponding magnetic machines. The design of the latter is shown in Fig. 27. They consist essentially of a horseshoe electro-magnet, beneath the poles of which travel two belts, the directions of the latter being at right angles. The poles of the magnet are 50 mm. long

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[1] These reactions are expressed merely as typical, the reduction not being necessarily effected by carbon, but perhaps by carbon monoxide or carbonic monoxide and carbon together.

[2] Thonind. Ztg., 1886, p. 429.

[3] In attempting the reduction of ferric oxide, by carbon or carbon monoxide, when it is mixed with zinc oxide, carbonate or silicate, it is necessary not only to obtain the proper reduction of the ferric oxide but also to avoid reducing the zinc compounds, which would lead to a loss of zinc by volatilization.

and have an elliptical cross-section, of which the axes are 0·06×0·15 m. They are wound with copper wire of 0·5 sq. mm. section. The machine shown in Fig. 27 was an experimental one, and those finally installed had two magnets over the lower, or feed, belt, and, of course, a second upper, or discharge, belt, to correspond to the second magnet. These machines require 100 watts of current—i. e., two ampères under a tension of 50 volts. In the following description the feed belt is designated as *a,* and the discharge belt as *b,* which letters are not shown in the engraving.

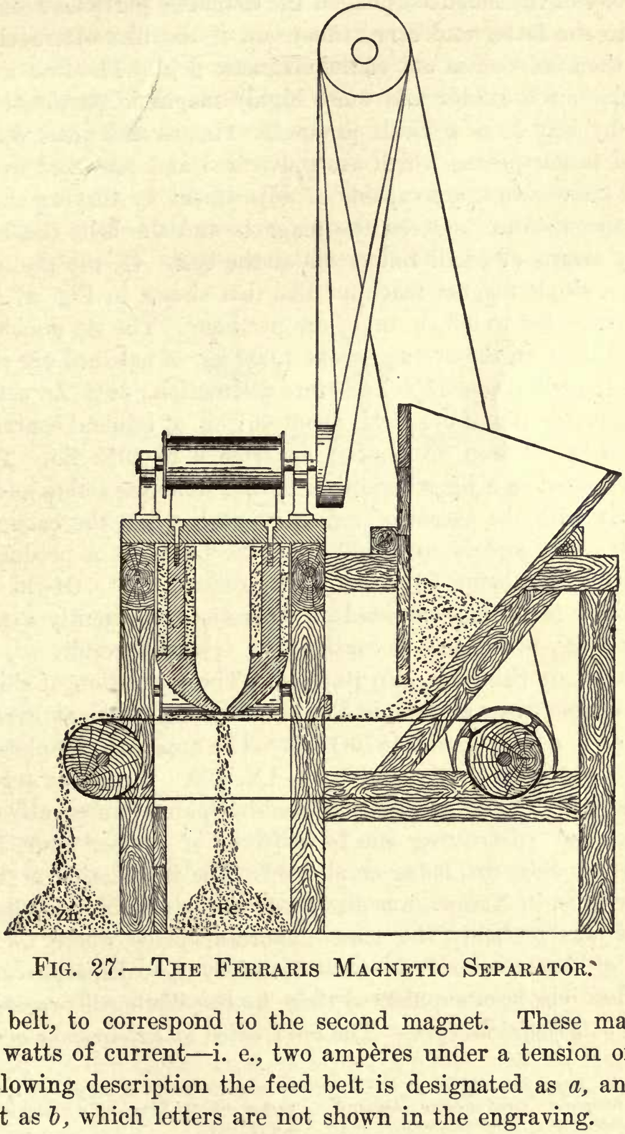

FIG. 27.—THE FERRARIS MAGNETIC SEPARATOR.

The Monteponi plant comprises six separators. The sized ore is delivered on the belt *a*, which carries it under the poles of the magnets at a distance of 20 to 50 mm., the distance being adjustable. The minimum speed of this belt is 0·5 m. per second. The belts *b*, which are interposed between *a* and the poles of the magnets, prevent the magnetic particles from becoming attached to the latter and carry them out of the line of travel of belt *a*, dropping them as soon as out of the magnetic field. The first magnet over belt *a* picks out the finer and more highly magnetic particles, which are delivered by belt *b* as a clean product. The second picks out a mixed product of larger pieces, which are pulverized and reworked over the machines. The machines are capable of adjustment by varying the magnetic force, or the distance between the magnets and the belt, the latter being effected by means of small rollers under the belt. In the preliminary experiments a single magnet machine, like that shown in Fig. 27, was found able to separate 0·2 to 0·3 cu. m. of ore per hour. The six double machines as installed treat on the average about 1,000 kg. of calcined ore per hour.

The ore treated assays 22% Zn before calcination; 30% Zn after calcination. One metric ton of ore yields about 667 kg. of mineral containing 40% Zn and 333 kg. of iron ore containing from 4 to 10% Zn. The former product is treated on a jig, wherein the burned dolomite slakes and is washed off, together with the excess of reduction coal, while the calamine passes through the sieve and is recovered from the hutch as a product assaying 48% Zn, its weight being 50% that of the roasted ore.[1] Of the 300 kg. of zinc contained in a ton of roasted ore there is consequently a recovery of 240 kg., or 80%, which may be considered a favorable result.

SEPARATION OF SIDERITE AND BLENDE.—The separation of siderite from blende by magnetism was done in the Lill dressing works at Przibram, Bohemia, as early as the decade 1870-1880. The apparatus employed was described in *Trans. Am. Inst. Min. Eng.*, IX, 420. A similar separation of siderite and oxidized zinc ore was practiced in Spain at an equally early date, machines capable of treating one to two tons of ore per hour, devised by Doctor Werner Siemens, being employed.[2] The installation at Friedrichssegen, near Ems, in Nassau, was also made as early as 20 years ago. Until recently it was probably the most important place where the magnetic separation of blende and siderite was effected by the calcination method, but the latter has now been supplanted there by the Wetherill process.

Practice at Friedrichssegen.—The ore treated at Friedrichssegen is a mill

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[1] E. Ferraris, Oest. Zts., 1898, p. 347.

[2] Werner Siemens, Gesammelte Abhandlungen, 1881, p. 537 et seq. Reference may be made also to a paper on magnetic separation by G. Prus, in Le Génie Civil, 1890, XVII, 337.

product assaying from 11 to 15% Zn (in the form of blende) and 18 to 23% Fe (in the form of siderite). This was heated to redness in a mechanical furnace of the McDougall type, which put through from 20,000 to 25,000 kg. of ore (according to the size of the particles) in 24 hours, with a coal consumption of 1,200 kg. The plant comprised two furnaces, each of which required the attention of one man, who also trammed the calcined ore to the cooling floor. When the ore had cooled to 50° C., or lower, it was elevated to a trommel, which divided it into sizes, namely, larger than 4 mm., 2 to 4 mm. and smaller than 2 mm. The stuff larger than 4 mm., which was due to fritting together of particles during the calcining, went to a set of rolls, by which it was reduced to 4 mm., whence it was raised again to the sizing screen; the stuff from 2 to 4 mm. in size and smaller than 2 mm. fell into separate bins, whence it was drawn to the primary magnetic separators. There were 12 of the latter arranged in three groups of four each, the four machines of each group being set in pairs and the pairs in series. The arrangement of the four machines constituting a group is shown in Fig. 28. Two groups took the coarser mineral and two the finer. The ore diverted to a group was divided equally between machines *A* and *B*, which made two products, one enriched in zinc and the other enriched in iron. The iron product of both machines was led to *D* and the zinc product to *C*. The two lower machines made a zinc product with 38 to 42% Zn and 6% Fe at the most, a mixed product, and an iron product, which still contained 6 to 8% Zn. The mixed product was retreated by a group of two machines. The iron product went to another group of four

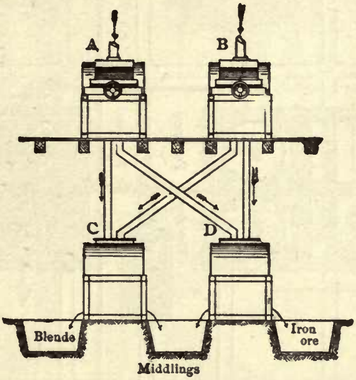


FIG. 28.—ARRANGEMENT OF MAGNETIC SEPARATORS AT FRIEDRICHSSEGEN, GERMANY.

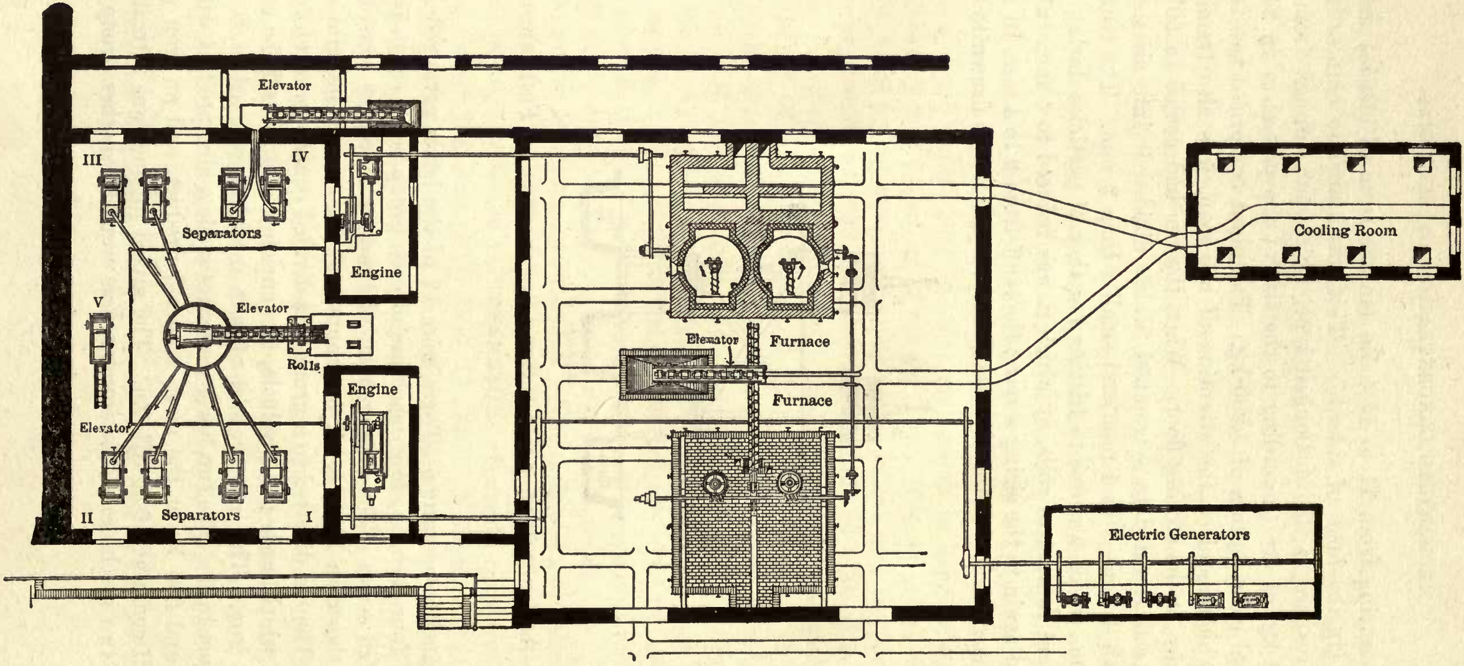
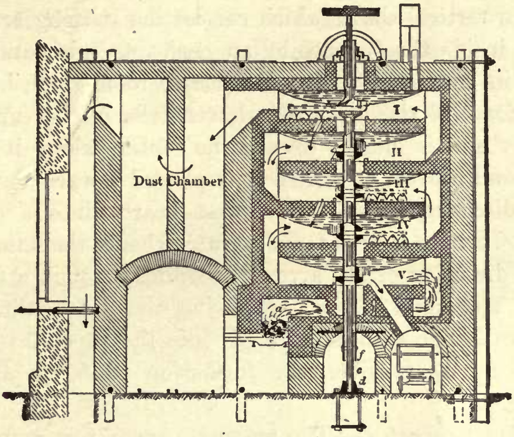


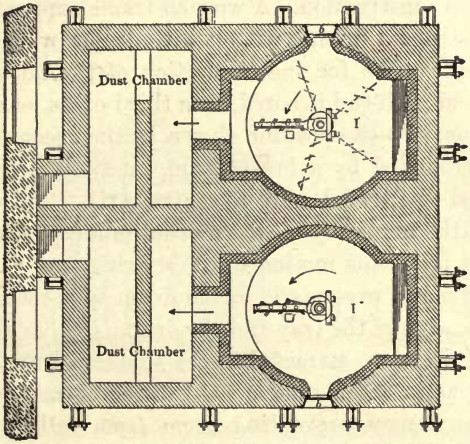
FIG 29.—PLAN OF MAGNETIC SEPARATING WORKS AT FRIEDRICHSSEGEN.

machines, which yielded a final product containing 40% Fe and 3 to 4% Zn; that represented the entire loss of zinc in the process.

It will be observed that the plant comprised 18 separators, of which 12 were employed in making the primary separation and six in reworking between products. The general arrangement of the plant is shown in the accompanying engraving, Fig. 29, which will be readily understood from the foregoing description.



Vertical Section



Horizontal Section

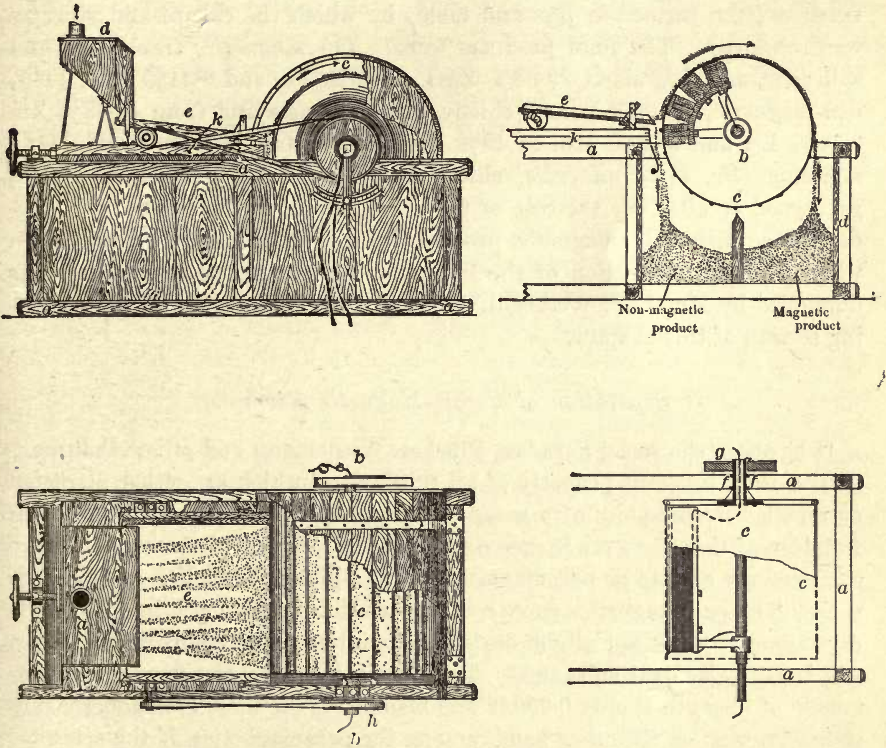
FIGS. 30 AND 31.—CALCINING FURNACE USED AT FRIEDRICHSSEGEN.

Calcining Furnace.—The calcining furnace used at Friedrichssegen requires no extended description, since it differs from the Herreshoff furnace and others of the McDougall type only in dimensions and structural details. It has two series of five superimposed hearths, which are about 6 ft. in diameter. The vertical shaft, which carries the stirring arms, is protected from the heat inside the furnace by an enclosing tube, and is driven by a worm gear at its upper end. In each hearth-room there is a stirring arm which moves forward the ore. The latter falls on the uppermost hearth at its periphery and is plowed toward the center, where it falls through a hole to the second hearth, on which it is plowed toward the periphery, and so on, being discharged from the lowest hearth into a car standing to receive it. The furnace is fired from a grate, whence the flames pass over the hearths in the direction of the arrows, escaping through a dust chamber to the chimney. The speed of the plows is regulated according to the size of the ore particles. The admission of air into the furnace is governed so as to prevent, so far as possible, the formation of ferric oxide instead of magnetic oxide.

The Wenström Separator.—The magnetic separators employed at Friedrichssegen, which are shown in Figs. 32 to 35, were of the Wenström type and very simple in construction. A wooden frame supports a stationary axis *b*, to which is fastened a casting wound with copper wire so as to form four electromagnets, the wires for the connection of the latter passing through the axis *b*, which at each end is bored for a third of its length. The magnets which are stationary in the position shown in the accompanying engraving, Fig. 33, are surrounded by a brass drum, on the exterior of which brass flanges are brazed on parallel with its central axis. The drum is rotated in the direction of the arrow by means of the pulley *g*, while a pulley *h*, on the opposite side, transmits motion to the shaking tray *e*, which receives ore from the hopper *d* and presents it to the drum in a thin, even sheet. The space between the edge of the tray and the drum is adjustable. In operation, the magnetic particles are attracted to the surface of the drum, where they are held so long as in the magnetic field, being thence carried over into a separate bin, the flanges preventing them from falling back. The non-magnetic particles fall directly from the edge of the tray into a bin. The capacity of a single machine is from 300 to 500 kg. per hour. The drum makes 36 r. p. m. and the shaking tray from 180 to 200 throws per minute. The power required is 0·125 h. p. and the electric current about 325 watts per machine. The Friedrichssegen plant comprised five dynamos, each of which delivered 20 to 25 amperes of 65 volts. One man attended to the entire plant of separators.

According to C. Blömeke,[1] the Wetherill machines, which have been introduced at Friedrichssegen, have been found to do twice the work of the Wenström machines, while they have the further advantage of occupying less room and not requiring the ore to be roasted.

Separation of Franklinite and Willemite.—In 1892, Mr. G. G. Convers, superintendent of the Lehigh Zinc Co.'s works at South Bethlehem, Penn., installed there an experimental plant of Wenström separators for the treatment of the mixed ore of New Jersey. The ore, which consisted of 51·92% franklinite, 31·58% willemite, 12·67% calcite, 0·52% zinkite, and 3·31% tephroite and other silicates, was first crushed to pass a 10-mesh sieve, and was then mixed with 20% of its weight of anthracite coal (buckwheat size) and passed through a brick-lined revolving cylinder, heated by gas from a Taylor producer. The heat of the furnace was regulated so



Figs. 32 to 35.—The Wenström Magnetic Separator.

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[1] Oest. Zts., 1898, p. 147.

that the ore issued from it at a bright red. The hot ore was conveyed into a revolving cooler, around the surface of which cold water was sprayed. After cooling, it was sifted to remove unburned coal (which was used subsequently on the grates of the zinc oxide furnaces, whereby the small quantity of zinc retained by it was recovered), while the ore itself was collected in bins, whence it was led to three Wenström magnetic separators, arranged in series. These machines were run so as to make a clean non-magnetic product of willemite, zinkite, calcite and silicates, and a magnetic product consisting chiefly of franklinite. The latter was sent directly to the oxide furnaces; the former to jigs and tables by which the calcite and silicates were removed. The final products were: (I) *magnetic,* franklinite and willemite, assaying about 29·66% ZnO, 37·20% Fe, and 9·34% Mn; (II), *non-magnetic, heavy minerals,* chiefly willemite, assaying from 46·38% Zn, 3·76% Fe, and 6·68% Mn. to 48% Zn, 2% Fe, and 7% Mn.; and (III) *non-magnetic, light minerals,* chiefly calcite and silicates. The process gave good results; but the cost of the roasting and the uncertainty of producing a uniformly magnetic product led to further experiments, from which a direct separation of the minerals, without previous roasting, was developed by Mr. J. P. Wetherill, whose process is described in the following section of this chapter.

Separation of Feebly-Magnetic Minerals.

It has been shown by Faraday, Plücker, Wiedemann and others that magnetism is an inherent property of all substances; which are either attracted or repelled by the poles of a magnet, though in most substances the manifestation of that property is exceedingly feeble. Those substances which are attracted are said to be paramagnetic; those which are repelled are diamagnetic. The paramagnetics show a wide variation in magnetic intensity; the diamagnetics show but slight deviation in intensity as compared with air, which is the neutral substance. Taking air as unity, the diamagnetic intensity of bismuth is only 0·99982 and bismuth is the most diamagnetic substance known; on the other hand, among the paramagnetics, if the attractability of steel be assumed as 100,000, that of magnetite is as high as 65,000, while siderite is only 120, hematite 93 to 43 and limonite 72 to 43. There is little data as to the relative magnetic attractability of various minerals,[1] and in any case such figures would not be of great value owing to the variable influence of impurities.

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[1] Plücker has given the following data as to the magnetic permeability of various minerals compared with metallic iron, which is valued at 100,000: magnetite, 40,227; siderite, 761; hematite, 714; limonite, 296; haussmannite (Mn_3O_4), 167.

For practical purposes it may be considered that all minerals containing manganese and iron, even garnet, are paramagnetic, except pyrite, which if paramagnetic is so feebly so that it cannot be separated from the diamagnetics, while quartz, calcite, dolomite and the zinc minerals (except franklinite and ferruginous blende) are diamagnetic. Pure blende is absolutely diamagnetic, while ferruginous blende is more or less paramagnetic. This difference is of great practical importance. In effecting a separation of pure blende from pyrite, as must be done with some of the Joplin ore, it is necessary to subject it to a preliminary roasting, because both the blende and the pyrite are diamagnetic. The blende of Leadville, Colo., on the other hand is ferruginous and sufficiently paramagnetic to enable it to be removed from the pyrite, which is mixed with it, without any preliminary treatment. According to Langguth[1] the magnetic permeability of blende varies according to its tenor of FeS or MnS.

Although the weak magnetism of many minerals besides magnetite and pyrrhotite was common knowledge for many years, it remained for J. Price Wetherill, general manager of the Lehigh Zinc Co., of South Bethlehem, Penn., to invent a practical machine developing such high magnetic intensity that advantage could be taken of this property for the commercial separation of various minerals, which either could not previously be converted into magnetic forms at all or could be thus converted only with difficulty and by delicate manipulation. Although the Wetherill machines are applicable to a great variety of metallurgical processes, they were designed primarily for the treatment of the mixed zinc ore of New Jersey, and in accomplishing that to perfection, besides furnishing a means whereby the mixed sulphide ore of Broken Hill and Leadville can be separated, and whereby iron can be largely removed from any zinc ore, they have constituted one of the most important contributions to the metallurgy of zinc in recent years. Their efficiency is so great that they are bound to be used in any new installation for the separation of blende and siderite, or calamine and limonite, instead of the simple magnetic machines described in the first section of this chapter.

THE WETHERILL SEPARATOR.—The Wetherill magnetic separators[2] are designed on the principle of developing a peculiarly high magnetic power, sufficient to act on minerals of such low magnetio permeability as limonite, hematite, siderite, etc. They were made originally of two types: (1) espe-

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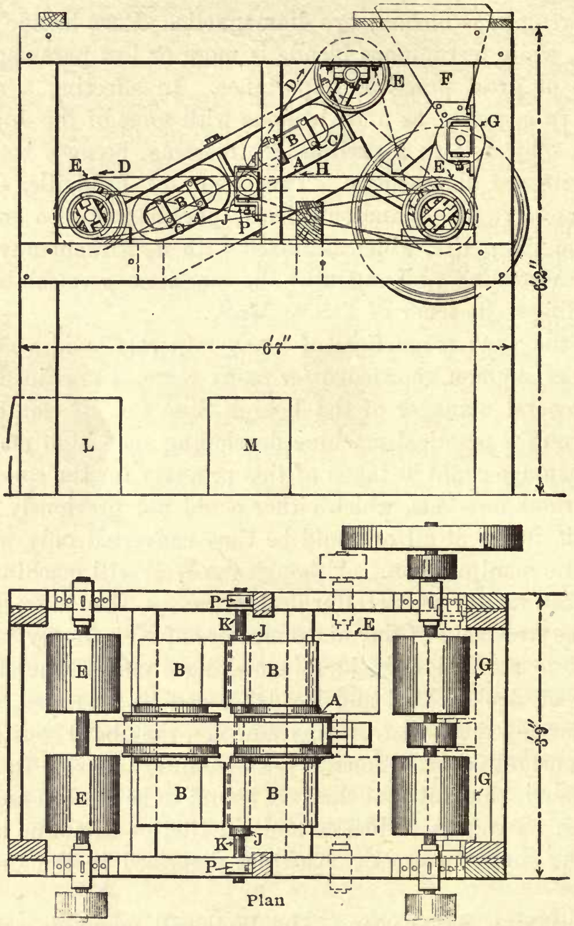
[1] Fascicule de l'Electrochemie, No. 23, Dec. 7, 1899.

[2] United States patents, Nos. 555,792 (March 3, 1896) and 555,794 (March 3, 1896). These patents are controlled by the Wetherill Separating Co. of New York, in the United States and Canada; and by the Metallurgische Gesellschaft, of Frankfurt am Main, in foreign countries.

cially for the treatment of fine ores and those from which rich magnetic heads are particularly sought; and (2) especially for comparatively coarse ores and those from which a clean non-magnetic product is desired.



FIGS. 36 AND 37.—PLAN AND ELEVATION OF WETHERILL MAGNETIC SEPARATOR, TYPE NO. 1.

The first type of machine is shown in Figs. 36, 37 and 38. This machine consists essentially of two magnetic cores and bobbins *AA,* and four pole pieces, *BBBB,* which are pointed in the manner shown and are adjustable by the bolts *C,* so that they may be moved nearer together or further apart, as may be desired. For substances of very low magnetic permeability two of the pole pieces are dispensed with and a solid yoke is used instead; for substances of higher magnetic permeability, such as garnet, franklinite, siderite, etc., the yoke may be removed and two pole pieces substituted for it, as shown. A belt *D,* shown in Figs. 36 and 38, driven in the direction of the arrow by the pulley *E,* shown in Fig. 36, passes between the negative and positive poles. The ore is fed from a hopper, *F,* by means of a feed roller *G,* upon a belt, *H,* which carries it in a layer, say $1/8$ to $^{3}/_{16}$ in. thick, to and over the pulley *J,* which is of a small diameter and rotates on a brass axle *K,* so arranged that it may be raised or lowered by the adjustable bearing *P.* The ore is thus delivered in close proximity to the space between the poles and the magnetic particles are withdrawn and lifted into the highly intensified field existing at that point. They are removed by the horizontal belt *D,* and carried into a receptacle *L.* The non-magnetic tails fall from the belt *H* into the receptacle *M.* The intensity of the magnetic attraction can be accurately adjusted by changing the distance of the feed belt from the pole points, or by changing the distance between the pole points, or by changing the ampèrage of the current.

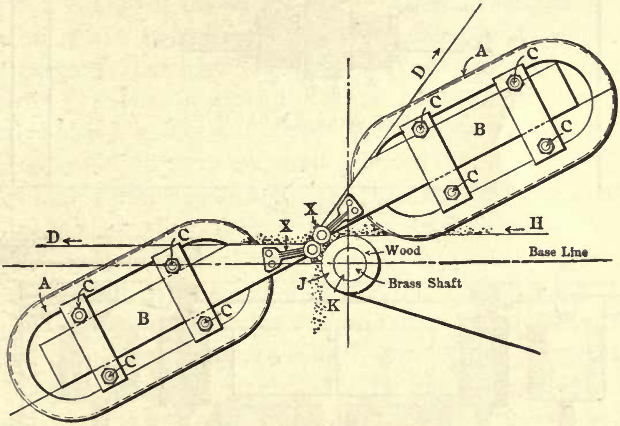
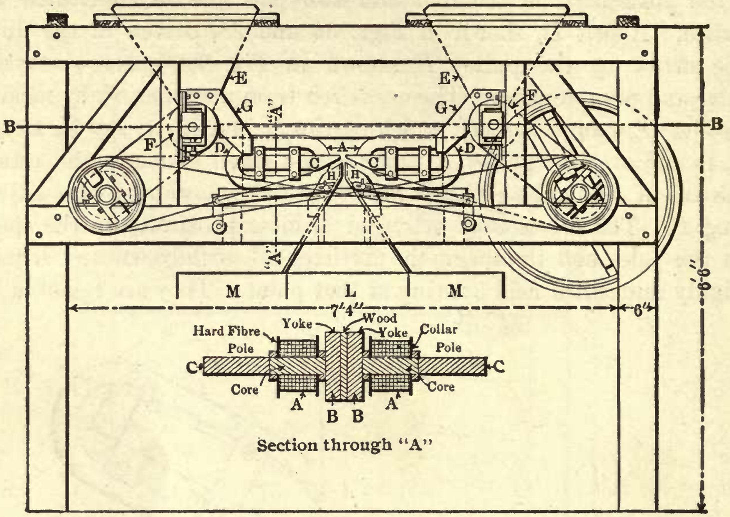


FIG. 38.—SECTION OF POLE PIECES OF WETHERILL MAGNETIC SEPARATOR.

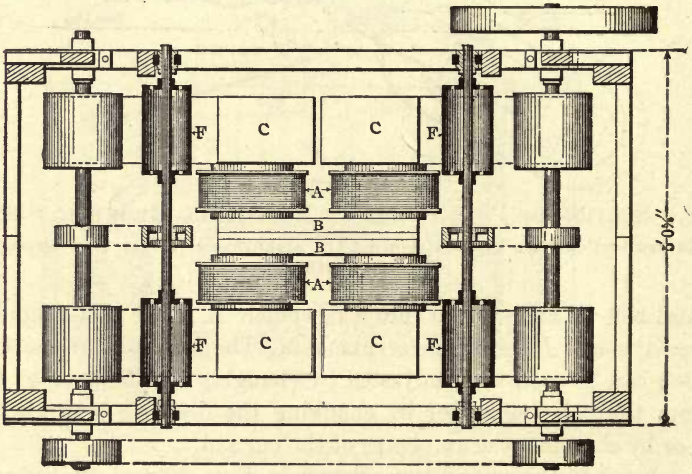
An enlarged section from Fig. 36, showing the arrangement of the pole pieces and the feed belt.

In machines of the second type, shown in Figs. 39 and 40, the magnetic particles adhere to the feed belt as it rounds the pole points, while the non-magnetic material falls away from it. The machine consists of magnets with cores and bobbins *A,* yokes *B,* pointed pole pieces *C,* belts *D* passing around the pole pieces in the manner shown, ore hoppers *E,* feed rollers *F,* and chutes *G,* which deliver the ore in a thin layer on the belts *D.*

The belts travel in the direction of the arrows and deliver the ore directly into the opening between the pole pieces, the distance apart of which is adjustable. Two shutters *HH*, one beneath the point of each pole piece, are adjustable so that the magnetic particles which adhere slightly to the pole pieces



Side Elevation



Section through "B"

FIGS. 39 AND 40.—SIDE ELEVATION AND HORIZONTAL SECTION OF WETHERILL MAGNETIC SEPARATOR, TYPE NO. 2.

are carried to one side by the moving belts *D* into the receptacles *MM* on either side of the shutters, while the non-magnetic fall through the space between the shutters and into the receptacle *L*. The separation may be regulated by varying the position of the shutters, the speed of the belts, the distance between the pole points, or the volume of current.

The machines described above were the earlier forms, as illustrated by Messrs. H. A. J. Wilkens and H. B. C. Nitze in *Trans. Am. Inst. Min. Eng.*, XXVI, 351 et seq. These have been superseded by others, which although operating on the same general principle show considerable improvement in their mechanical features. The main credit for these improvements is due to Mr. Lewis G. Rowand and Mr. Max Schiechel, the former having been connected with the exploitation of the machines in America and the latter having been in charge of the operations in foreign countries.[1]

The machines now in use may be divided into two types: (1) those in which the magnetic particles are lifted and carried away from the non-magnetic; and (2) those in which the course of the moving ore is influenced magnetically in such a way as to carry the various ingredients in separate directions, according to their magnetic permeability.

The most highly developed machine of the first class is illustrated in Fig. 43, while its method of operating is shown diagrammatically in Figs. 41 and 42. This has a horizontal carrying belt which passes between the poles of a pair of magnets, as represented in Fig. 42, the pointed pole pieces PP^1 of the upper magnet *M* being placed close to the flat pole pieces *LL′* of the lower magnet *M′*. The discharge belts *T* and *T′* pass around the poles of the upper magnet, as shown in Fig. 41. In Fig. 42 only one pair of magnets is shown, but generally this type is arranged with two or three pairs, the latter number being shown in Fig. 43. The standard machine of this type has an 18 in. carrying belt. The current employed for three pairs of magnets is approximately 5+15+30 amperes at 110 volts; the mere driving of the belts does not require more than 0·5 h. p. The capacity of the machine ranges from 600 lb. to 12,000 lb. per hour. Such a machine costs $3,850 f. o. b. New York, besides which the owners of the patents demand a certain royalty per ton of product for the license to use it.

A machine of the second class, which is used in Europe and Australia for ores that are only slightly magnetic, is illustrated in Fig. 44. In this there are three magnetic coils and pole pieces. The pole point *P*, which does the active work of separation, is of opposite polarity to the points P^1 and P^2. Around the pole point *P* passes a thin linen belt *F*, on which the ore is fed from the hopper *H* by means of the feed roller *R*. When the ma-

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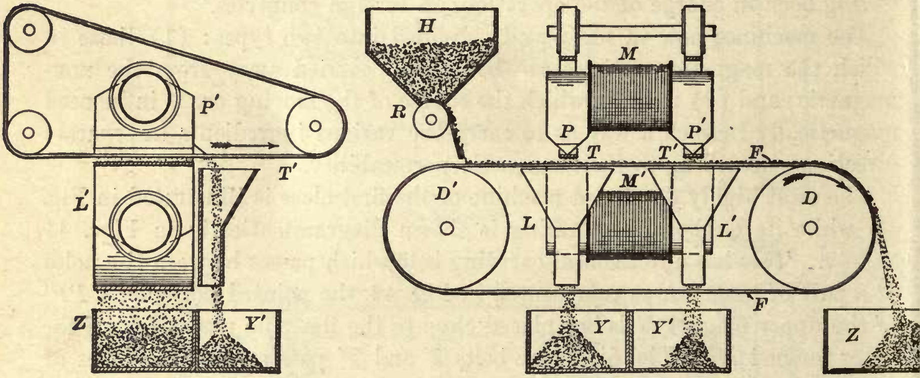
[1] H. A. J. Wilkens, The Mineral Industry, X, 776.

terial reaches the pole point *P*, the magnetic particles adhere to the feed belt *F* as it rounds the point, while the non-magnetic are thrown off by the momentum imparted to them by the belt. By means of the shutter *S*, which can be adjusted with a small set screw, the two grades of ore are accurately separated. This machine operates successfully on materials of very slight magnetic permeability at the expense of a comparatively low magnetic current, it being only necessary to hold the material for a moment against the pole point as compared with the necessity of actually lifting the materials in machines of the type shown in Figs. 41 to 43. There is some wear of the feed belt while rounding the pole point *P*, as well as that of the pole point itself. The proper adjustment of the shutter *S*, as well as that of the whole machine, is a somewhat delicate matter, and its use is advisable only when the ore is so slightly magnetic that it is impossible to make satisfactory separation with the machine shown in Figs. 41 to 43.



FIGS. 41 AND 42.—MODERN FORM OF WETHERILL SEPARATOR WITH HORIZONTAL BELT.

A three pole machine which combines certain of the features of the two types described above has been used in Europe to a considerable extent. In this the feed belt passes around a roller below the field of the three magnets, which are themselves surrounded by a take-off belt. As the ore reaches the magnetic field, the non-magnetic material is dropped by the feed belt as it passes over the roller, while the magnetic is lifted by the magnets to the take-off belt and is carried forward thereby. In operation the take-off belt is needed more to keep the pole points clear than to hold and carry the ore.

Theory of the Wetherill System.—It will be observed that the Wetherill machines do not differ essentially in mechanism from certain of the simple magnetic separators which have been used for various purposes. Their

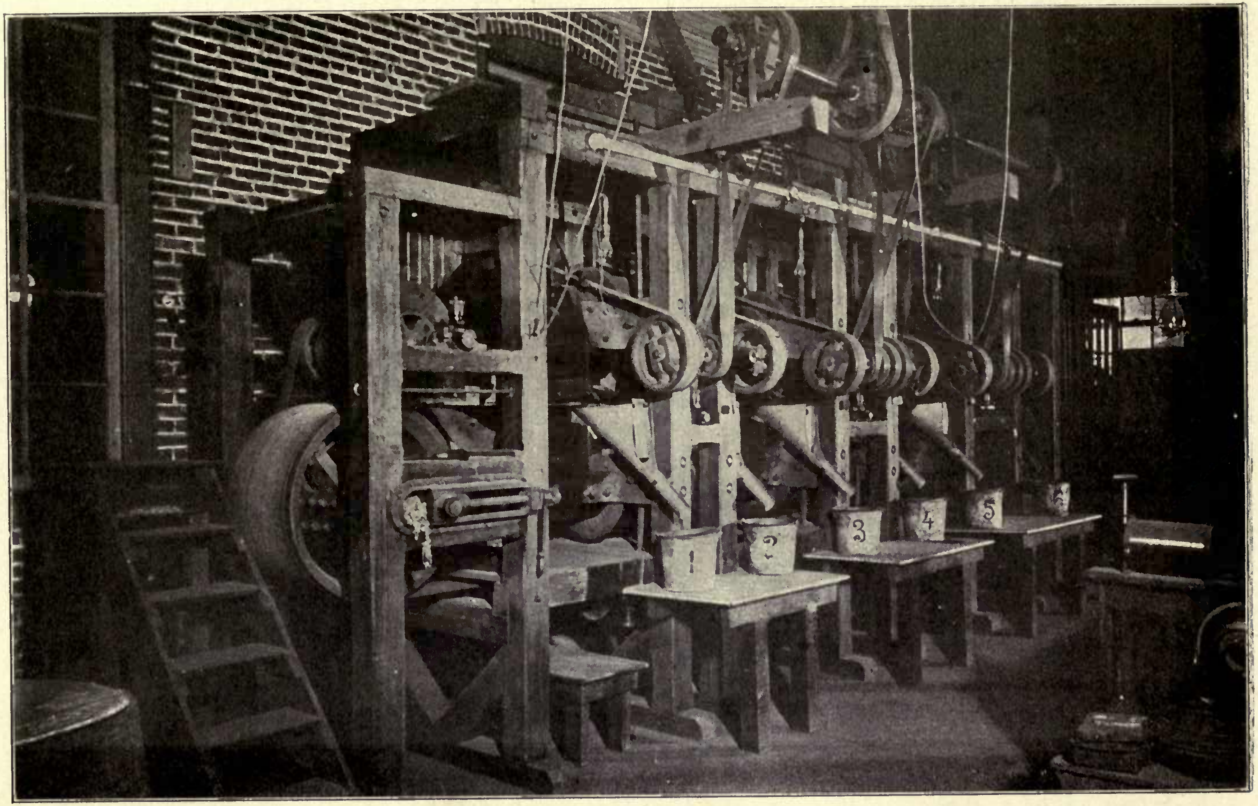


FIG. 43.—GENERAL VIEW OF WETHERILL MAGNETIC SEPARATOR.

novelty and the principle upon which they are designed are in the production of an intense magnetic field, which is obtained by the almost perfect condensation in the pole points of the vast number of magnetic lines generated in the large and correctly proportioned cores, yokes and pole pieces. Professor William A. Anthony expressed concisely the difference between the principles of the Wetherill and other magnetic separators as follows:[1] "Instead of small magnet cores and large air gap area, there should be large magnet cores and small air gap area—large magnet cores to reduce the resistance and permit of the production of a large number of magnetic lines, and a small air gap area through which all those lines must pass, and in which, therefore, they are crowded together, which is the same thing as saying that the magnetic force is intensified." This condensation is so complete in the Wetherill machines that the current of one bluestone cell, such as is used in telegraphy, is too powerful to permit a satisfactory separation of magnetite,[2] and for that purpose its strength must be reduced by resistance coils, in order to prevent tangling and allow the belts to draw the ore away from the pole points. The magnetic intensity of the poles can be regulated with such delicacy, by varying the current, that from a mixture of garnet, monazite and rutile the monazite can be removed at one operation, after which by a slight variation in the current the garnet and rutile can be separated.

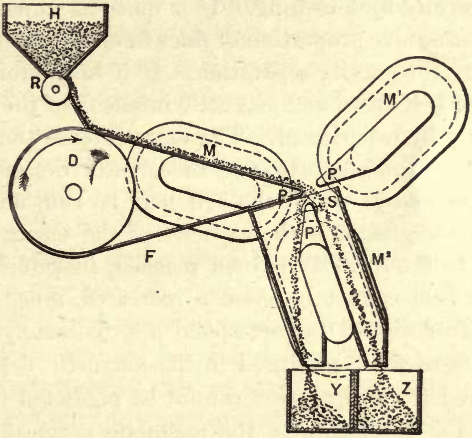


Fig. 44.—Modern Form of Wetherill Separator, Three Pole Type.

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[1] Cassier's Magazine, March, 1898, p. 433.

[2] H. A. J. Wilkens and H. B. C. Nitze, Trans. Am. Inst. Min. Eng., XXVI, 361.

The volume of current required by the Wetherill magnets is comparatively small. In the separation of the franklinite from the New Jersey zinc ore, three to eight ampères and 16 to 30 volts are necessary; for limonite and pyrolusite 10 to 15 ampères are required. The capacity of the machines is from 600 to 12,000 lb. per hour, depending upon the number of pole pieces, the width of the carrying belt, the speed of the belt, the volume and intensity of the current, and the size and character of the material operated upon.

Conditions of Practical Application.—In the employment of the Wetherill process for the separation of minerals it is important to subject the ore to a pulverization sufficiently fine to free the component minerals, which must then be sized carefully by screening. It is quite as important to avoid the production of an excessive proportion of fines in crushing ore for this process as it is in crushing for gravity separation. It it impossible to determine in advance the proper belt-speed and magnetic intensity of the separators, which can be settled only by experiment. The more intense the concentration of the lines of force in the magnets, and the greater the quantity of electric energy, the more speed at which the belt may be run, and the greater the speed of the belt the greater is the capacity of the separator. In practice, however, a limit to the capacity is soon reached, inasmuch as the intensity of the magnetic field cannot be greatly increased, since the poles of the electromagnets are designed to correspond approximately to a determined maximum of lines of force produced in the magnetic field. The quantity of current required for a separation cannot be predicted in advance, moreover, because of the variability in the magnetic permeability of the same mineral from different deposits.

PRACTICAL RESULTS OF THE WETHERILL SYSTEM.—The Wetherill system is now in use for the treatment of zinc ores at Franklin Furnace, N. J., Austinville, Va., Warren, N. H., Washington, Ariz., Kokomo, Denver and Cañon City, Colo., at Ems, Gladbach, Hamborn, Lohmansfeld and Friedrichssegen, in Germany, and at Melbourne and Broken Hill, N. S. W. It has been tried on a working scale at Joplin, Mo., but the plant there is not at present in operation. The results obtained at some of those places are summarized briefly in the following paragraphs:

Austinville, Va.—In separating a mixture of calamine and limonite obtained from the jigs at the works of the Wythe Lead and Zinc Co. at Austinville, Va., 100 parts of ore, assaying 18·60% Fe and 29·57% Zn, yielded 67 parts of zinc product, assaying 3·41% Fe and 41·40% Zn, and 33 parts of iron, assaying 49·45% Fe and 5·58% Zn.

Mine Hill and Franklin Furnace, N. J.—The largest and most important installation of the Wetherill process has been made by the New Jersey Zinc Co. at Mine Hill, where a plant of 500 tons per day capacity has been in operation since 1896, while a new mill of 1,000 tons per day capacity at Franklin Furnace was completed in 1901. Both of these are for the separation of the mixed franklinite and willemite obtained from the mines near by, which have been described in a previous chapter.

At the Mine Hill works the ore passes first over a grizzly with 1·5 in. spaces, the undersize going to an inclined screen with 0·5 in. holes and the oversize to a Blake crusher, which delivers its product to a screen with 1 in. holes. The undersize from the latter screen is received on a picking belt, which also takes the oversize, this arrangement being to reduce the wear of the belt, inasmuch as the coarse ore falls upon a cushion of fine ore. The oversize from the 0·5 in. screen is delivered to the same belt. Clean waste is picked from the belt, which conveys the enriched ore to a Blake duplex crusher. The product of the latter passes to two trommels with 0·25 in. holes, from which the oversize goes to the rolls and the undersize, together with the material which passed through the 0·5 in. screen, is delivered to an Edison dryer. The latter is a tower, 3 ft. square and 24 ft. high, with zigzag shelves, set at 45°, down which the ore slides while the products of combustion from a fireplace pass over it.

The dry ore is delivered to four trommels, covered with eight-mesh wire cloth, from which the oversize is crushed finer by rolls and then goes with the undersize to six trommels, covered with 10-mesh wire cloth. The oversize from the 10-mesh screens is passed through a four-mesh screen to remove nails, splinters of wood and other rubbish, and after being crushed finer by rolls is returned to the 10-mesh trommels. The material which passes through the 10-mesh screens is received in storage bins, whence it is drawn to the magnetic separators as required.

In the separating department the ore passes first over six Wetherill double machines, which yield franklinite concentrate and mixed tailings. The latter go to two trains of sizing trommels, one on each side of the mill. Each train consists of four trommels covered respectively with wire cloth of 16, 24, 30 and 50-mesh. They make five sizes of ore, which are delivered to storage bins, whence they are drawn to separators on each side of the mill. Each size passes over a series of three machines, the second taking the tailings of the first and the third those of the second, the concentrate of each machine being franklinite. The machines which treat the finest size are of the sloping type; the others are horizontal.

On each side of the mill there are four Harz jigs, of which Nos. 1, 2 and

4 have four compartments, and No. 3 has three compartments. These receive the tailings from the first four series of magnetic machines and make a concentrate, consisting of willemite and zinkite, and tailings consisting chiefly of calcite and quartz. The concentrate is dried in a revolving cylinder with longitudinal ribs inside, which is heated by direct firing. The tailings from the fifth magnetic machines are so rich in willemite and zinkite that they do not require a further concentration by gravity.

According to J. P. Wetherill,[1] the cost of treating 4,812 long tons of ore in February, 1897, was 74·54c. per ton, of which 53·32c. was for labor, 8·77c. for coal and 12·45c. for repairs, renewals and miscellaneous supplies. Out of 30,311 tons separated up to April 1, 1897, there were produced 20,455 tons of franklinite, or 67·48%, 7,271 tons of willemite and zinkite, or 23·99%, and 2,585 tons of tailings. The franklinite assayed 29·47% Fe, 13·57% Mn, and 22·94% Zn. The mixed willemite and zinkite assayed 2·20% Fe, 5·15% Mn, and 48·96% Zn. The grade of the latter varied from 46·5% to 53% Zn, according to the quantity of zinkite contained in the ore. The tailings averaged 4·19% Zn.[2] In 1900 the two mills treated 160,640 long tons of ore, which yielded 148,917 tons of concentrates, or nearly 93%.

In mill No. 2, near Franklin Furnace, the Edison system of fine crushing was first employed, but it was found to produce too high a proportion of fines and ordinary rolls were substituted for the Edison kind. This mill has 17 machines, each with three pairs of magnets, and treats 70 tons of ore per hour at a cost of about 40c. per ton.

Joplin, Mo.—At the Empire Zinc Co.'s smeltery at Joplin, Mo., Wetherill machines were employed for the removal of iron from certain concentrated blende which was considerably higher in pyrites than the average of the Joplin district. The ore was first roasted dead, as in the ordinary preliminary to distillation, the iron sulphide being thereby converted to oxide, which is susceptible to the magnetic influence of the Wetherill machines. The roasted ore delivered to the machines assayed 8% Fe. It was passed over two machines in series at the rate of 10 tons per 24 hours, the result being a zinc product assaying only 2% Fe, which went to the distillation furnaces, and an iron product assaying 15% Zn, which was thrown away.

Denver, Colo.—At the works of the Colorado Zinc Co. mixed sulphide ore from Leadville and Kokomo is treated. The ore is crushed to 30-mesh size by means of a Gates crusher and rolls, and after an hydraulic classification is washed on eight Wilfley tables. The latter produce from Leadville ore a galena concentrate assaying about 50% Pb, 13% Fe, 9% Zn and

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[1] Eng. & Min. Journ. July 17 and 24, 1897.

[2] These analyses show an average tenor of 27·5% Zn in the crude ore in the actual milling of 30,311 tons.

10 oz. Ag. per ton, and a pyrites-blende assaying about 21·5% Fe, 30% Zn and 5% Pb. The pyrites-blende is dried and then is fed to the Wetherill magnetic separators, of which there are two; they make a blende concentrate assaying about 50% Zn, 12% Fe and 1% Pb (the iron being chiefly contained in the blende as monosulphide), and pyrites tailings which contain about 7 or 8% Zn and 10% Pb. The tailings from the separator are mixed with the galena concentrate from the Wilfley tables and sold to the lead smelters. The Wetherill separators treat about 24,000 lb. of material per 24 hours apiece. They are of the design shown in Fig. 43. The belt is 18 in. wide and there are three magnets, which require respectively 7, 11 and 22 ampères of current, a total of 40 ampères, at 115 volts.

Lohmannsfeld, Germany.—An interesting installation of the Wetherill process has been made at Lohmannsfeld, in the Siegen district, Germany, at which place a deposit of ore consisting of galena, siderite and blende, with a gangue of quartz and quartzite is worked. The siderite (spathic iron ore) contains up to 12% of manganese. By the ordinary process of mechanical separation it was impossible to obtain from this ore any concentrate but galena of various classes, while blende and spathic iron could be got only by an expensive system of hand sorting. The third and fourth compartments of the Harz jigs furnished always a mixed product containing from 2 to 3% of gangue and 15 to 22% of blende, according to the tenor of blende in the crude ore, the remainder being spathic iron. The quantity of this mixed product ranged from 600 to 750 metric tons per month. In order to produce a marketable blende it was necessary to roast the mixture to transform the ferrous carbonate into magnetic oxide, which could then be removed by ordinary magnetic separators. The company operating at Lohmannsfeld not being able to install roasting furnaces at its mines, was obliged to sell its mixed product under very unfavorable terms, the average price realized at the mine being from 12 to 15 marks per metric ton. Experiments in separating the ore crushed to 3 mm. size by means of the Wetherill machines gave such favorable results that it was determined to make an installation of them.

The crude mineral sent to the Wetherill plant is the middlings from the Harz jigs of the wet-dressing works, varying in size from 1 to 10 mm., and containing from 5 to 20% of water. The arrangement of the plant is shown in the accompanying engravings, Figs. 45 to 48. The mineral is dried while being moved forward by means of a screw conveyor, the trough of which is made double, of cast iron, exhaust steam from the engine being passed between the two sections. The speed of the screw *A* is such that the mineral remains in the trough 30 minutes, while in the trough *B* it remains 25 minutes. These conveyors discharge the dried mineral into the primary

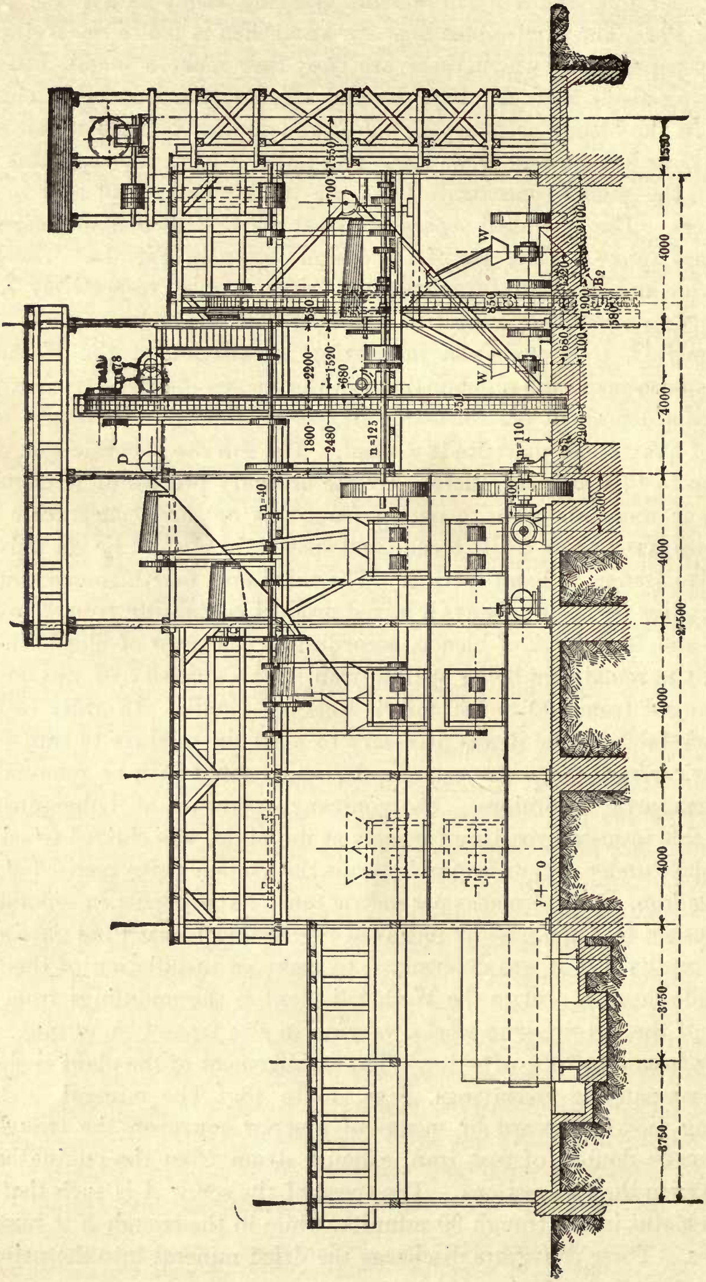

FIG. 45.—LONGITUDINAL ELEVATION OF WETHERILL MAGNETIC SEPARATING PLANT AT LOHMANNSFELD, GERMANY.
Scale, 1:200.

trommels, A_1 and A_2, by which it is separated into two classes. That which is finer than 3 mm. falls into the boot of the elevator C_1, which raises it to the main system of classifying trommels; that which is larger than 3 mm. passes to two sets of rolls, whereby it is crushed, falling thence into the pit B_2, whence the elevator C_2 raises it to the trommel A_2, to separate it into mineral smaller than 3 mm., which falls into the pit B_1, and mineral larger than 3 mm., which is returned to the rolls.

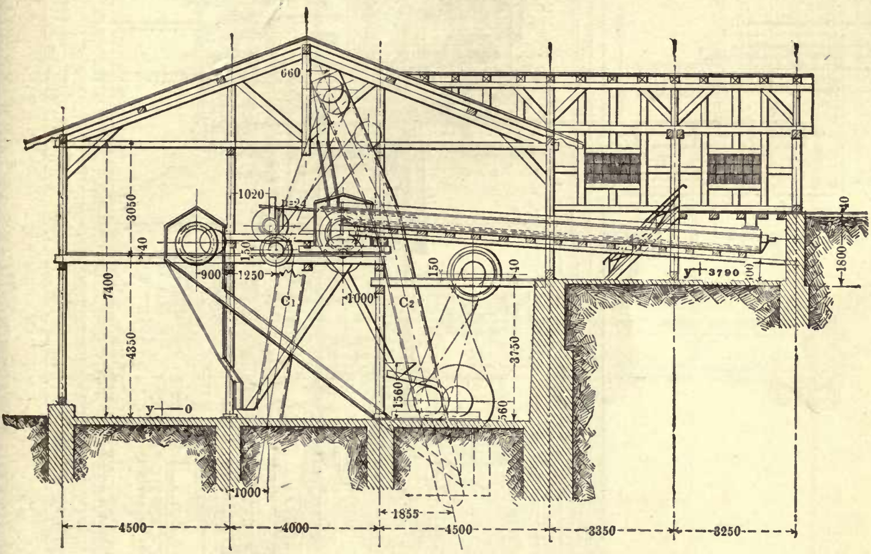

FIG 46.—TRANSVERSE SECTION OF WETHERILL MAGNETIC SEPARATING PLANT AT LOHMANNSFELD, GERMANY.
Scale, 1 :200.

All the mineral crushed to pass a 3-mm. screen is collected therefore in the pit B_1, whence the elevator C_1 raises it to the train of sizing trommels, to reach which, however, it has to be conveyed by a horizontal belt *D*, over which it set a powerful electro-magnet to pick out strongly magnetic articles such as rivet heads, small pieces of iron and steel, etc.

The sizing trommels separate the mineral into four classes, namely, 3 to 2 mm., 2 to 1·4 mm., 1·4 to 0·75 mm., and that which is smaller than 0·75 mm. The separated classes pass thence over three series of Wetherill separators of the two-pole and three-pole types, arranged two in a series. In the first member of each series, pure spathic iron is separated by a feeble current. The diamagnetic product passes to the lower machine where the intensity

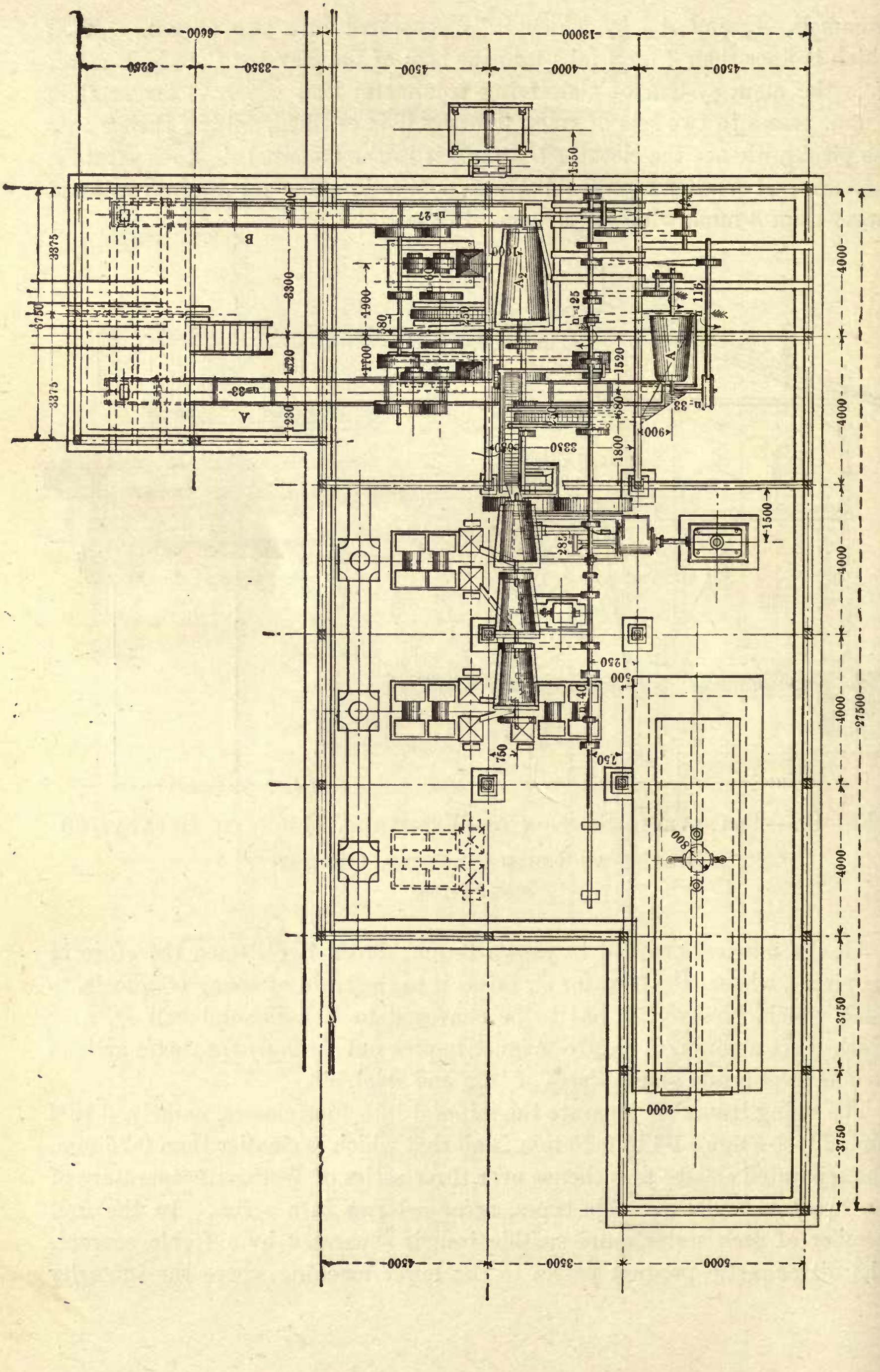

FIG. 47.—PLAN OF WETHERILL MAGNETIC SEPARATING PLANT AT LOHMANNSFELD, GERMANY.

of the magnetic field is greater, enabling a mixed product of blende and spathic iron to be separated as the paramagnetic, while pure blende remains as the diamagnetic. The blende of Lohmannsfeld is diamagnetic even in magnetic fields of the greatest intensity, because of the absence of iron and manganese in its composition. The width of the points of the poles of the Wetherill machines used at Lohmannsfeld is 340 mm. The belt speed of the upper machine is 40 m. per minute; of the lower 25 m. The current employed is of 65 volts electromotive force; the upper machine of the two-pole type works with 12 ampères; the lower with 14 to 16 ampères. For machines of the three-pole type the corresponding figures are five ampères for the upper and eight ampères for the lower.

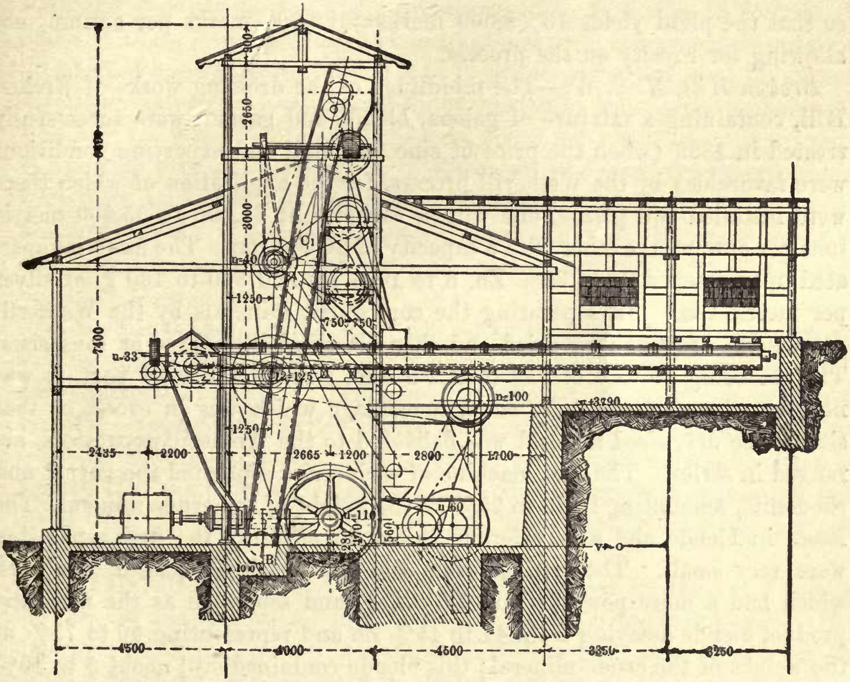

FIG. 48.—TRANSVERSE SECTION OF WETHERILL MAGNETIC SEPARATING PLANT AT LOHMANNSFELD, GERMANY.

The plant is capable of separating 3 to 3·5 metric tons of crude mineral per hour, the crew required for its operation being as follows: One foreman, five 16 to 18-year-old boys, one machinist (engineman) and one stoker. The cost of treatment (year's average) is 1·4 marks per metric ton of crude mineral, no amortization of the plant being reckoned. The plant cost about 100,000 marks. Allowing 20% for amortization, =20,000 marks per annum on 8,000 tons (the capacity of the plant) the total cost of treatment is 3·9 marks per ton. The value of the ore sent to the works being 12 to 15 marks, the total cost of production is 15·9 to 18·9 marks. The value of the blende recovered from a ton of crude mineral varies from 32 to 35 marks, so that the plant yields 16×8,000 marks=128,000 marks per annum, not allowing for royalty on the process.

Broken Hill, N. S. W.—The middlings of the dressing works of Broken Hill, containing a mixture of galena, blende and garnet, were successfully treated in 1899 (when the price of zinc was high and exporting conditions were favorable) by the Wetherill process, for the application of which there were installed two plants, one with a capacity of 30,000 to 35,000 metric tons per annum, the other with a capacity of 20,000 tons. The material operated upon assayed 25 to 30% Zn, 8 to 10% Pb and 300 to 400 g. of silver per metric ton.[1] In separating the component minerals by the Wetherill process the ore was first dried and then classified into three or four sizes. The ore being already of less than 2·8 mm. size for the most part, it was necessary to crush only the small percentage which was in excess of that size. The dry, sized mineral was delivered to the Wetherill separators, arranged in series. The first machine of each series separated the garnet and rhodonite, amounting to 15 to 25% of the weight of the crude mineral. The losses in blende and argentiferous galena in effecting the first separation were very small. The tailings from the first machines passed to others, which had a more powerful magnetic field and separated as the magnetic product blende assaying from 41 to 45% Zn and representing 50 to 75% of the weight of the crude mineral; this blende contained still about 8 to 10% Pb and 350 to 400 g. of silver per ton. As diamagnetic product there was obtained a mixture of quartz with 10 to 20% Pb, 10 to 20% Zn and 300 to 500 g. of silver per ton, this product constituting about 20% of the weight of the original mineral; it was subjected to a further process of gravity separation in the ordinary manner, yielding a concentrated product of argentiferous galena. The total recovery of zinc varied from 80 to 90% of that which was contained in the crude middlings. The cost of the process was said to be a little less than the cost of wet concentration in the ordinary manner as practised at Broken Hill.

The figures quoted above were taken from a paper by M. Smits, read be-

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[1] This material was considerably richer than the average of the "zinc middlings" made by the Broken Hill dressing works, which according to T. J. Greenway (in The Mineral Industry, IX, 752) contain 5 to 8% Pb, 15 to 25% Zn, and 5 to 10 oz. silver per ton. Data as to the assay of the crude ore will be found on p. 229.

fore the Congrès International des Mines et de la Métallurgie of the Exposition of 1900 at Paris and reprinted in the *Bulletin de la Société de l'Industrie Minérale,* vol. XIV, 1900. According to T. J. Greenway, a recognized authority on the dressing of the Broken Hill ore, the separation of the ore in the Wetherill plants was made in three classes, namely: I, non-magnetic material consisting chiefly of quartz, blende containing a large portion of lead, and free galena, the whole averaging 18 to 20% Pb, 18 to 22% Zn, and 8 to 12 oz. silver per ton. II, weakly magnetic concentrates containing 8 to 10% Pb, 35 to 42% Zn, and 8 to 12 oz. silver per ton; and II, strongly magnetic material consisting chiefly of rhodonite and garnet and assaying 4 to 6% Pb, 6 to 8% Zn, and 4 to 6 oz. silver per ton.

The zinc product was shipped to European smelters; the other products were piled up for future treatment. The Wetherill plants at Broken Hill are now (1901) closed down, blende concentrates of the grade made not affording any profit over and above the cost of production under the existing market conditions.

It is obvious that the Wetherill process at Broken Hill was only partially successful, the unsatisfactory results being ascribable to the exceedingly difficult character of the ore. "The above mentioned operations, as well as exhaustive experiments carried on by some of the mining companies, have rendered it evident that the average blende concentrate obtainable from the Broken Hill sulphide ore is unavoidably a complex product containing from 30 to 40% Zn, from 6 to 12% Pb, and from 6 to 12 oz. silver per ton" (Greenway, loc. cit.).

La Trieuse Separator.—This machine was used in 1892 at Laurium, Greece, for the removal of iron from a roasted pyrites and blende. It is described as having had two powerful electro-magnets, with poles terminating in finger-like ends, mounted in a frame and swung by a rod, with a universal joint, from the end of a counterpoised lever over a tray of ore. The current could be adjusted to suit the degree of roasting to which the ore had been subjected. The iron oxide having been picked up by the magnet, the latter was swung to one side over a receptacle into which the mineral was dropped by shutting off the current. To work 300 to 1,000 kg. of ore, assaying 22% Zn, in 10 hours, required 2·5 to 3 ampères and 40 volts.[1]

Cleveland-Knowles Separator.—This is a new magnetic separator of the high intensity type, in which it is aimed to prevent magnetic leakage except across the air gap, with the consequent increase of magnetic intensity at that point, which should be directly proportional to the decrease of leak-

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[1] Hugues Daviot, Comptes Rendus Mensuels del'Industrie Minérale, Sainte Etienne, May, 1893; The Mineral Industry, II, 827.

age at other points. According to Mr. W. P. Cleveland, one of the inventors of the machine, to whose courtesy these notes are due, that result is attained by providing a return path for the lines of force, from pole to pole of the magnet, of a greater permeability than air, wherefore all lines of force will pass through the conductor in preference to leaking through the air against a greater resistance, and will pass across the magnetic field with all or nearly all of the initial intensity.

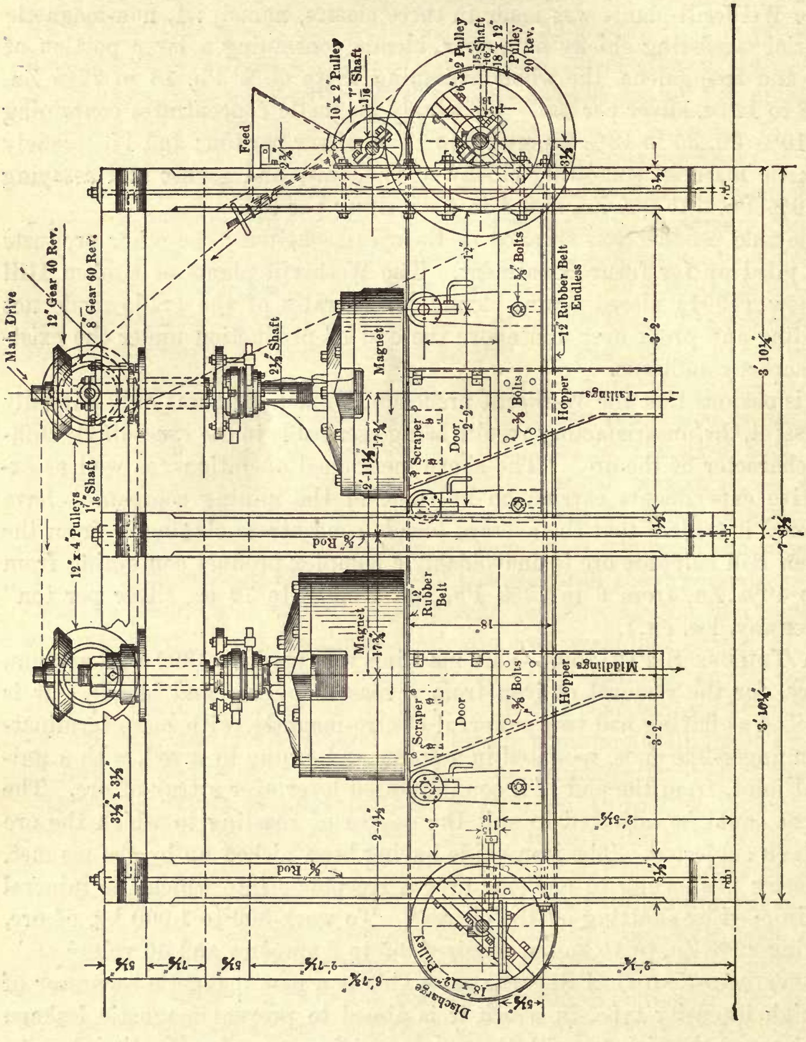


FIG. 49.—CLEVELAND-KNOWLES MAGNETIC SEPARATOR.
Side Elevation. Scale, ½ in.=1 ft.

In construction, the magnet is iron-clad, i.e., the coil is entirely surrounded by one of the poles itself. The magnetic field is therefore endless, being in the form of a circle, one pole concentric with the other and the space between them filled with a diamagnetic material. Two of these magnets, termed respectively the "rougher" and the "cleaner" are revolved in a horizontal plane above an endless conveying belt, upon which the ore to be separated is evenly spread. In passing beneath the magnets, each of which is placed so as to offset the belt on one side, the magnetic mineral is attracted and carried to one side of the belt, where it is removed from the magnet by a scraper and is dropped into a collecting chute. A current of 0·5 ampère

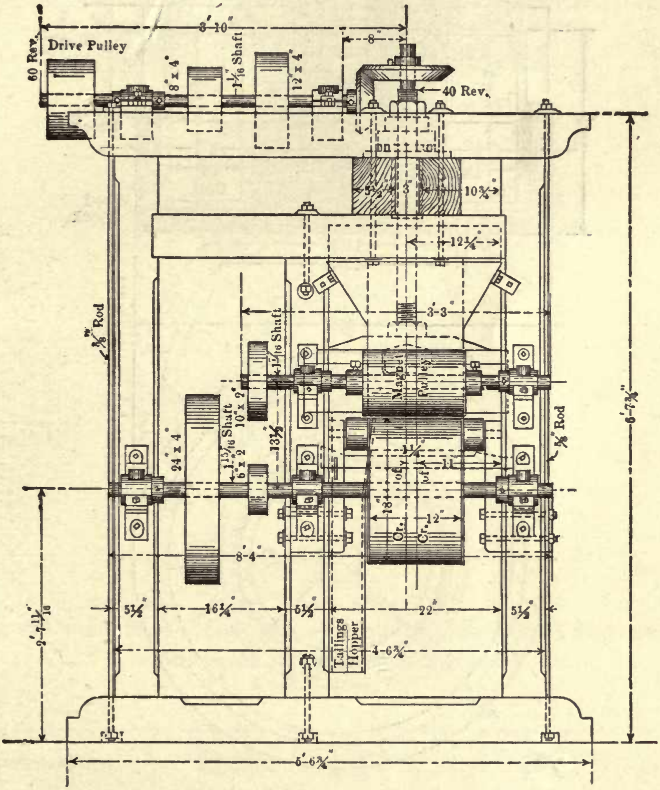
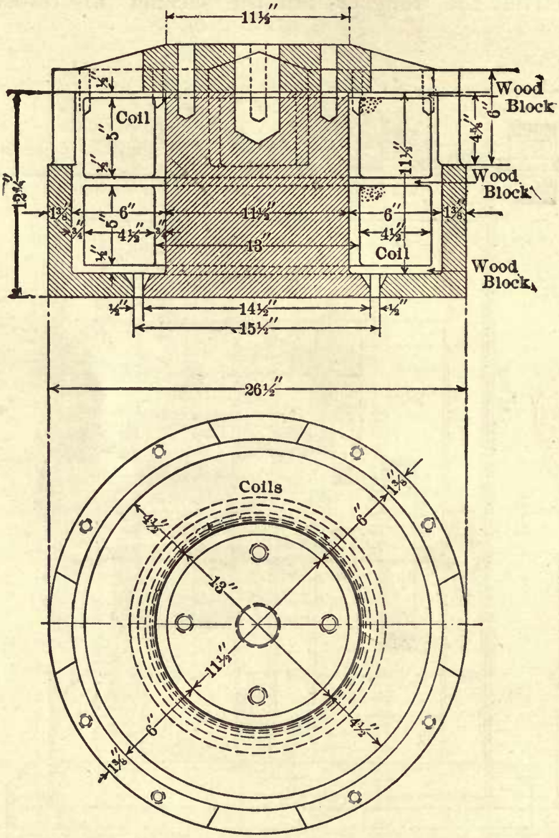


FIG. 50.—CLEVELAND-KNOWLES MAGNETIC SEPARATOR.

END ELEVATION.
Scale, ½ in.=1 ft.

is employed for the "rougher" magnet and one of 6·5 ampères for the "cleaner."

The Cleveland-Knowles separator has been in use since September, 1900, at the works of the Joplin Separating Co., at Joplin, Mo., where about 40



FIGS. 51 AND 52.—CLEVELAND-KNOWLES MAGNETIC SEPARATOR.

LOCATION OF COILS IN MAGNETS.
Scale, 1 in.=1 ft.

tons per 24 hours of blende-pyrites concentrates are treated. The mixture is first subjected to a light[1] preliminary roasting, in order to render the

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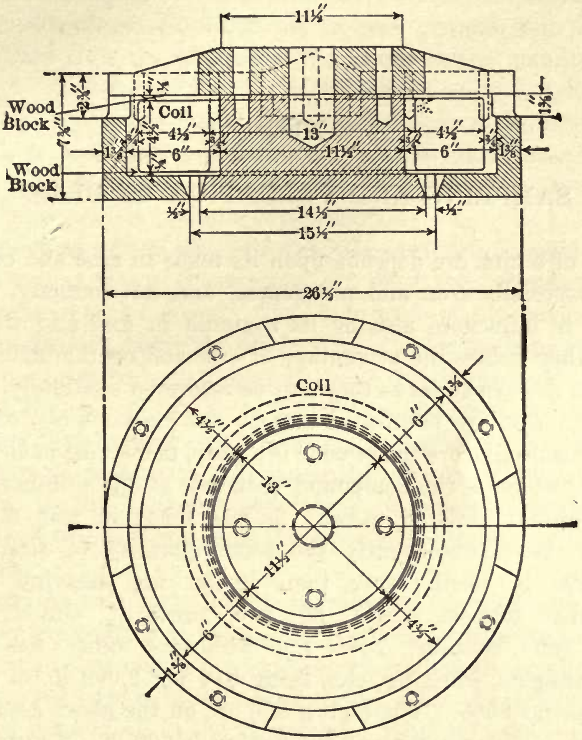
[1] Doctor William B. Phillips as the result of a series of experiments for the separation of pyrite from blende produced by a mine in Sevier County, Arkansas, came to the conclusion that it was unnecessary to remove more than one-third of the sulphur of the ore in the roasting process preliminary to magnetic separation; and that there was in fact a positive disadvantage in strongly roasting the ore, which led not only to a

pyrite magnetic, and is then passed to the separators. The results of some work that has been done are shown in the following table:



FIGS. 53 AND 54.—CLEVELAND-KNOWLES MAGNETIC SEPARATOR.

LOCATION OF COILS IN MAGNETS.

Scale, 1 in.=1 ft.

Source of Ore.	Quantity Treated.	Assay of Crude.		Assay of Concentrate.	
		% Zn.	% Fe.	% Zn.	% Fe.
Galena, Kan	10·5 tons	53·97	5·00	63·25	1·33
Joplin, Mo	43·0 "	47·19	9·84	59·45	1·00
" "	27·0 "	45·25	9·67	58·35	1·29
" "	7·0 "	51·56	6.13	57·20	2·40
Oronogo, Mo	22·0 "	44·60	12·39	62·15	1·02

The Cleveland-Knowles separator is controlled by the Magnetic Separating Co., of Joplin, Mo. It is illustrated in Figs. 46 to 51.

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diminished yield of zinc concentrate but also to a lower tenor of zinc in it (Eng. & Min. Journ., Nov. 30, 1901, p. 711). The kind of separator, with which these experiments were made, was not stated by Doctor Phillips.

XII.

SAMPLING AND VALUATION OF ORES.

The value of a zinc ore depends upon its tenor in zinc and objectionable impurities, especially iron and manganese, and its character, oxidized or sulphide; it is influenced also by its contents in lead and silver, always in so far as they reduce the percentage of zinc and contaminate the spelter and in certain cases in so far as they may be recovered separately. Obviously the value of a zinc ore is affected also by the market price of the metal. The most valuable zinc ore imaginable is a pure, dense zinc oxide, containing 80·344% Zn, which is the compound reducible at the minimum cost, and with the minimum loss of metal. As the tenor in zinc falls the cost of smelting rises; consequently the same quantity of zinc in an ore assaying 70% is worth more than in an ore assaying only 50%. For example, suppose there be two ores of similar character, except that one contains 70% Zn, while the other has only 50% the cost of freights, smelting, etc., being $10 per 2,000 lb. of ore and the recovery of metal 80%. The high grade ore on the above assumption will yield 1,120 lb. of zinc per ton and the low grade 800 lb. Assuming that the product is worth 5c. per lb., delivered at the smelting works, the value (without reckoning amortization, profit, etc.) of the high grade ore would be $56—$10=$46 and the value to the miner of the zinc in it would be $46÷1,400=3·286c. per lb.; similarly, the value of the low grade ore would be $40—$10=$30 and $30÷1,000=3·0c. per lb. of zinc. Obviously, the decrease in the value of the zinc in an ore with decrease in the grade of the latter is greater the higher the cost of carriage to the smelter's works and the cost of smelting per ton of ore. In the above example the cost of smelting per ton of ore and recovery of metal have been assumed the same for the high grade as for the low grade, though practically there would be a difference in favor of the higher grade.

The cost of smelting a calcined or roasted ore is increased moreover by the presence of those impurities (sulphur, iron and manganese), which may cause an increased loss of zinc and perhaps a greater consumption of fuel and reduction material. In referring the cost of smelting to terms of crude ore much will depend upon the relative cost of calcining calamine and roasting blende, the percentage of weight lost in each process, and the value that may be realized from the sulphur of the blende in the manufacture of sulphuric acid. The effect of other impurities such as lime, fluorspar, arsenic and antimony, which may contaminate the products and otherwise interfere in the process, must also be considered. It is essential therefore in establishing the value of a zinc ore to determine its composition; as a preliminary to which a representative sample is necessary.

In Europe most of the zinc ore which is smelted is bought on the basis of its zinc tenor as shown by sampling. In the United States comparatively little is bought in that manner. This is because of the peculiar conditions of the Joplin district, where the production of ore is made by a host of small miners, who expect cash immediately upon the delivery of their product, or rather at the end of the week of delivery and in many cases desire to realize on less than carload lots. It would be extremely difficult in view of the conditions of production to change the custom of buying and selling the ore so as to put it upon a strict basis of sample and assay, although during the last two or three years there has been a tendency to do so as far as practicable.

Besides serving as a basis for the purchase of ore a sample is important as a basis for the control of the smelting process, wherefore even if the ore be not bought in that manner everything received at the smeltery should be sampled there for a guide in the metallurgical work. It is only recently that this has been done at all in the Kansas and Missouri smelting district, but most of the large concerns now employ chemists and sample the ore received more or less carefully. Systematic sampling may also be made of advantage in the operation of dressing works, although it is only in rare instances that the design provides for doing so, and the introduction of a cheap and efficient system presents considerable difficulty. The only way whereby the efficiency of a milling or metallurgical process can be accurately determined is a comparison between the valuable contents of the crude material treated and those of the products obtained. Deductions from any other data are likely to be misleading. If losses were always determined accurately, many results which are considered satisfactory would prove not so, unsuspected leaks would be found, and by closing them the efficiency of the process would be increased.[1]

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[1] **An excellent series of articles, entitled " Notes on Sampling," was published in the Mining Reporter (Denver), Oct. 24, Nov. 28, Dec. 5, 12, 19 and 26, 1901 ; Jan. 16, 23 and Feb. 6, 1902.**

SAMPLING.

THEORY.—In theory the sampling of ore consists in a system of averaging, so that out of a large lot, say 50 or 100 tons, or even a ship's cargo of 2,500 tons or more, a small amount, e.g., 500 g. or 1 kg., which is further reduced by the analytical chemist to 1 g., may be obtained, which is perfectly representative of the whole lot. That this can be done with much exactness is proved by the concordance of the results of sampling the same lot of ore by two or more different methods. Whatever method of sampling be employed, the principle consists of the reservation of a certain proportion of the ore in different stages of the process. Its success depends upon taking a large proportion of the ore from many different places in the early stages, and the thoroughness of the mixture of the particles effected in the later. This mixture is accomplished first by simply crushing the ore, and afterward, as the reduction in bulk progresses, by turning it over by shoveling, or by mechanical devices.

Supposing a lot of 100 tons of ore consisting of pieces of mixed blende and quartz, all of 3 in. diameter, were to be sampled, the conventional method is to reserve every tenth shovelful in unloading the railway cars or wagons, whence there results a first sample of 10 tons, or 10% of the original quantity. This sample being so large and having been taken at regular intervals in the discharge of the ore, the same proportion from all parts, may be assumed to be thoroughly representative of the whole, which is the case as demonstrated by experience. If now the aforesaid sample were to be reduced further in the same manner, without any intervening steps, the final result would be one shovelful or one piece, which might or might not be representative of the original ore, according to the uniformity of the latter; in all probability it would not be representative, the error accumulating with each successive cutting out. Consequently, the theory of averages demands a breaking up of the first sample into smaller pieces before reducing it in bulk any further.

The idea may be illustrated by supposing that we had a mixture of 9,000 white marbles and 1,000 red ones, which it was desired to sample. By taking one tenth of them at regular intervals, precisely the same quantity each time, we should probably obtain 900 white and 100 red. A second cutting would give 90 and 10 respectively, if the result chanced to be perfect, while the third would give nine and one. A fourth would necessarily give either one white marble or one red one, but in either case the result would be incorrect. If, however, the nine white and one red had been each broken into tenths and the pieces had been thoroughly mixed the averaging might have gone on with as slight chance of error as at the third cutting. It would have been still better if they had been broken before the third cutting. If these figures were increased many fold the conditions would be somewhat analogous to those in sampling a lot of lump ore.

Crushing Required.—The size to which a lot of ore should be broken before each cutting out, in order to make it possible to obtain a correct average, may be calculated mathematically. Such a calculation is based on the principle that a certain ratio between the weight of the sample and the size of the largest particle having been assumed, it shall be maintained through every stage of the operation, because the volumes and weights of pieces of the same kind of ore are to each other as the cubes of their diameters. Most of the calculations of this kind that have been made have been for gold and silver bearing ores, which owing to the more irregular distribution of their values are more difficult to sample correctly than an ordinary zinc or lead ore. However, a couple of examples of such a calculation will serve to illustrate the principle.

Reed reckons[1] that with a silver ore averaging 50 oz. per 2,000 lb., containing pieces as high as 300 oz. per 2,000 lb., in cutting down from 100 tons to 10 tons, the pieces of ore may be as large as cocoanut size; from 10 tons to one ton, orange size; from one ton to 0·1 ton, walnut size; from 0·1 ton to 6 lb., pea size; and from 5 lb. to 0·5 lb., fine enough to pass a 20-mesh sieve. Argall states[2] that in sampling rich gold ores with automatic machines accurate results are obtained when the sample cut out is 20% of the weight of the ore when the average size of the ore is 1 in., 1·25% when the size of the ore is 0·25 in., 0·0785% when the size of the ore is 0·0625 in. (8-mesh), and 0·005% when the size of the ore is 0·0171 in. (30-mesh). In sampling material of 8-mesh size, therefore, a cutting out of 157 lb. from 20,000 lb. should be accurate, although in practical work a somewhat larger proportion would be taken as a matter of extra precaution.

In sampling ores for blast furnace smelting, it is objective to avoid breaking the pieces smaller than 2·5 to 3 in. size any more than is absolutely necessary, since that size is the best for the operation of the furnace. Nothing of that kind has to be considered in sampling a zinc ore, which, if it is to be reduced by the Belgian or Rhenish process, must be anyway crushed finely enough to permit of it being cut down directly to a sample of small bulk. The major part of the ore received by the zinc smelter is generally a concentrated product, already fine and mixed to a rather uni-

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[1] School of Mines Quarterly, VI, 357.

[2] In a paper on "Sampling and Dry Crushing in Colorado" read before the Institution of Mining and Metallurgy, Feb. 20, 1902.

form composition, so that a comparatively small proportion taken out by a single cutting will afford an accurate sample. If the material were absolutely uniform a random grab sample would be quite sufficient, but that condition is one that is hardly to be relied upon.

The zinc smelter does not therefore generally have to consider the more elaborate systems of sampling that are necessary with ores of more variable character and less uniform composition, but in this connection it is well to describe briefly the methods employed in other branches of metallurgy, which may be modified to suit the particular conditions. The sampling of ore and other products may be done either by hand or mechanically, or by a combination of both methods. Mechanical sampling is the cheaper and the more accurate, but sampling by hand is still employed to a considerable extent and is often necessary in determining the value of occasional lots of material for which the means for mechanical sampling might be inconvenient or unavailable.

SAMPLING BY HAND.—The method which is the cheapest, when the ore does not have to be crushed for another purpose, and at the same time is as efficient as any other, with the further advantage that it can be performed satisfactorily by inexperienced men, is known as fractional selection. It consists merely in shoveling over a pile of ore, putting aside an aliquot part, say every fifth or every tenth shovelful. This may be done as the cars or wagons are being unloaded. The chief precautions to be observed are to shovel always from the floor and work gradually into the pile from one or more points of attack—i.e., avoid shoveling around the pile at random, which might result in two shovelfuls reserved for the sample being taken from about the same place in the pile.

As in other methods of sampling, the percentage of the ore which may safely be reserved in the first cutting out depends upon the fineness and the uniformity of the mixture. If the ore be very fine, two or more cuts may be made before further crushing, but the ore should be well mixed each time before doing so. Otherwise it should be crushed finer before making the second cut, and so on. For the fine crushing of small samples machines of the coffee-mill type are generally employed. When the sample has been reduced to a small quantity, say 200 lb., the further cutting down is best done by means of a grid or riffle sampler, or the modification of it which is known as Jones' sampler.

The sample grinder shown in Fig. 55 has a muller at the head of a vertical shaft, which is revolved in close proximity to an annular die forming a continuation of the hopper into which the ore is fed. Below the hopper there is a cylindrical housing, which confines the crushed ore and discharges it through two spouts, below which pans are placed to catch it. The degree of crushing is regulated by raising or lowering the muller by means of the hand wheel, which controls the lever on which the shaft is stepped. The machine shown in Fig. 52 has tight and loose pulleys 16×4·5 in., weighs 775 lb. (900 lb. boxed for shipment) and requires 3 h. p. These machines are driven at 150 r. p. m. They cost about $100, f. o. b. makers' works.

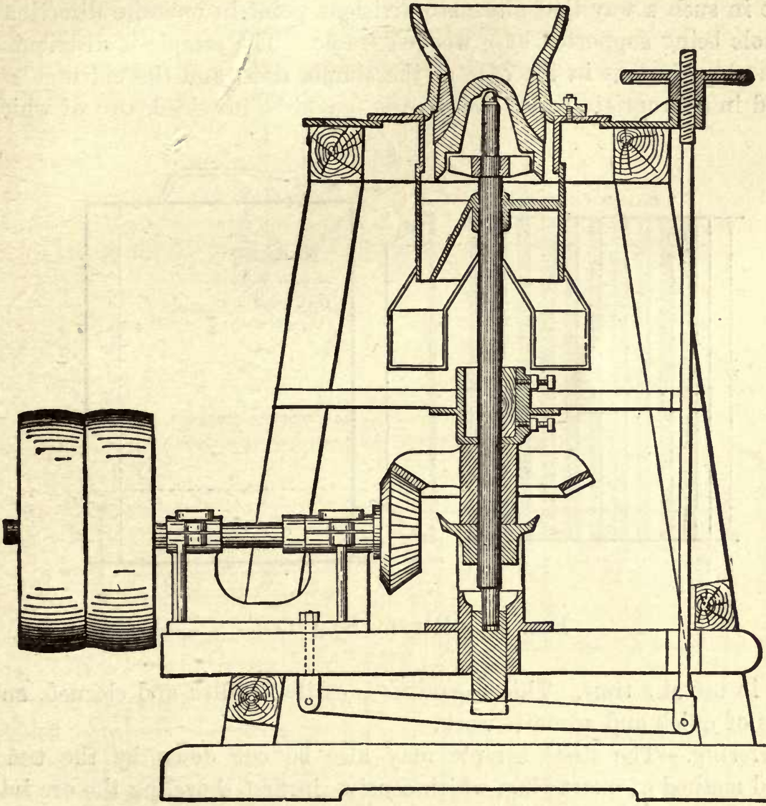

FIG. 55.—SAMPLE GRINDER.

Grid or Riffle Sampler.—The grid, or riffle, sampler consists of a series of riffles alternating with the same number of open spaces, the width of the riffles and open spaces being equal, 1 in. width being a common dimension. The fine ore being fed in an even sheet by means of a tray of the same width as the grid over all, the tray being drawn longitudinally over the riffles, one half will fall through the open spaces into another tray placed underneath, while the other half will be caught in the riffles.

Jones' Sampler.—This apparatus, which is shown in Figs. 57 to 59, is an improvement on the simple riffle and tray, but works on the same principle. It consists of a series of narrow, triangular hoppers, which are joined side by side in such a way that alternate divisions point in opposite directions, the whole being supported by a wooden frame. The sample is distributed over the top, just as in the case of the simple riffle, and the cuttings are received in appropriate pans. Four pans should be provided, two of which will be in use at a time. This apparatus is easily handled and cleaned, and permits of quick and accurate work.

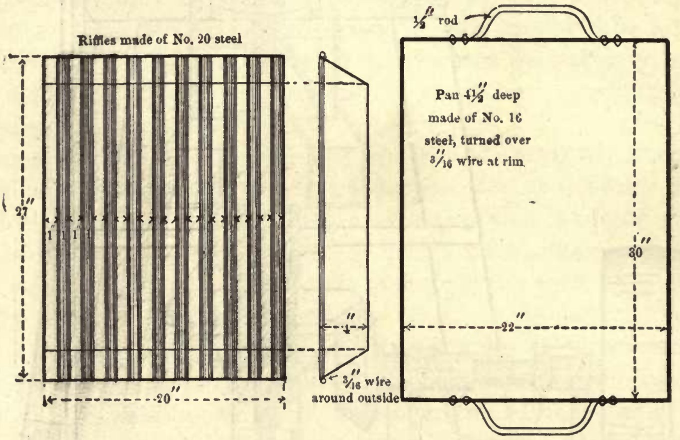

FIG. 56.—RIFFLE SAMPLER.

Quartering.—The first[1] sample may also be cut down by the time-honored method of quartering, which consists in first shoveling the ore into a large ring, each shovelful being thrown on the ring in regular order from left to right, or vice versa. The fine ore remaining on the floor inside the ring is swept to the center and is carefully gathered up and distributed evenly all around the ridge of the ring. The ore in the ring is then shoveled into the center in such a way as to form a true cone. In doing this, it is essential to shovel gradually around the ring, always moving in the same direction, and throw each shovelful precisely on the top of the cone, so that

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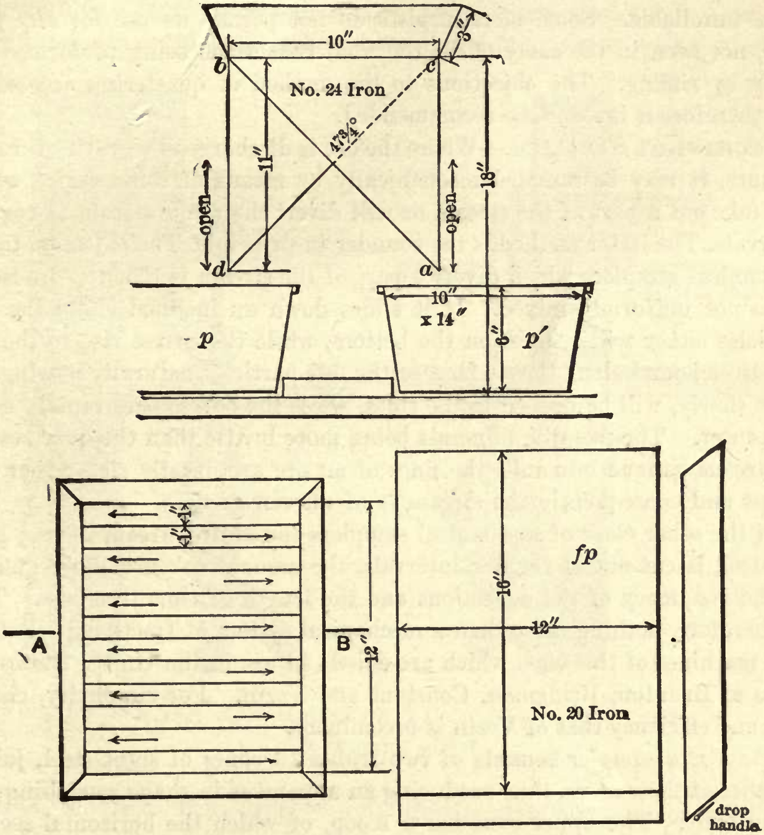
[1] By this is meant the sample obtained in reserving a certain percentage of the shovelfuls in the first handling of the ore. The direct quartering of a large lot of ore, say 50 or 100 tons, would be too cumbersome a process.

the particles of ore will distribute themselves equally over the pile. The object of ringing and coning is to mix the ore thoroughly.

The cone having been formed, it is pulled down with the aid of long-handle, round point shovels, beginning near the top of the cone, and walking around it, working it down from center to periphery until it becomes a truncated



FIGS. 57 TO 59.—JONES' SAMPLER.

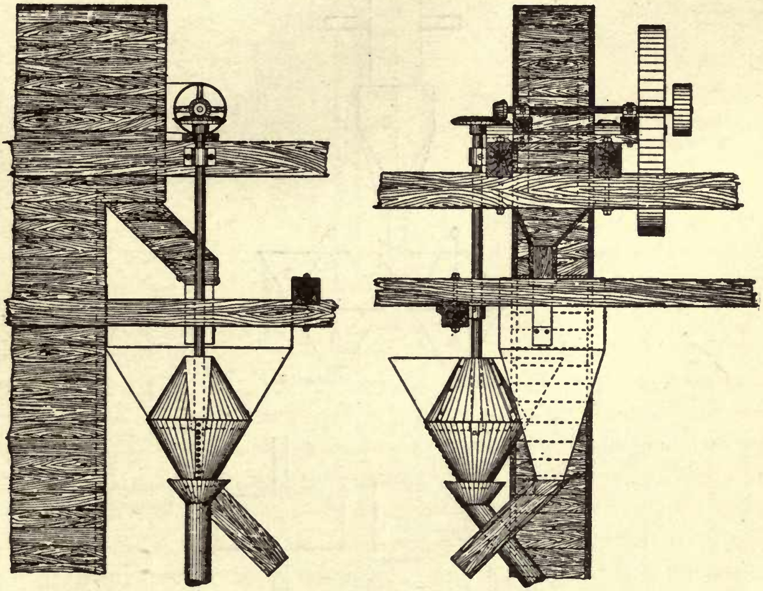
Fig. 57: Section on line AB of Fig. 58. **Fig. 58:** Plan. **Fig. 59:** Charging pan.

cone about 6 in. high. This is marked off into quarters by means of a rod with a sharp edge and the alternate quarters are shoveled away, whence the name "quartering." Care must be taken to sweep away the dust from the floor where the quarters have been removed. The remaining quarters are then ringed out and coned as before, and quartered down until the sample amounts to 50 or 100 lb., after which the further reduction is generally done by riffling. It is customary to employ two men at quartering, in which case they shovel on the ring and cone diametrically opposite to each other. They ought to sample 2,000 lb. down to 5 lb. in about two hours. The method of sampling by quartering is subject to serious errors, and although still employed in numerous works, has been discarded in others as being quite unreliable. Some metallurgists do not permit its use for any purpose, not even in the assay office, the final reductions being performed entirely by riffling. The objections to the method of quartering are sound, and therefore it is not to be recommended.

MECHANICAL SAMPLING.—Where the ore is discharged by gravity through a chute, it may be sampled mechanically by means of some device which will take out a part of the stream or will divert the whole stream at regular intervals. The latter method is the sounder in principle. The objection to the mechanical samplers which divert a part of the stream is that the ore is apt to be not uniformly mixed. If it slides down an inclined chute the fine particles either will remain on the bottom, while the coarse rise to the top and often bound along the surface, or the fine particles, naturally moving the more slowly, will be pushed to the sides, while the coarse pass rapidly down the center. The metallic minerals being more brittle than the quartzose or calcareous gangue minerals, the fines of an ore are usually richer than the lumps and consequently the stream is of uneven grade.

In the other class of mechanical samplers the entire stream of ore, good and bad, is cut out at regular intervals, the amount taken being regulated by the frequency of the deflections and the length of time they last. This is, therefore, nothing more than a mechanical system of fractional selection. The machines of this class which are chiefly in use in the United States are those of Brunton, Bridgman, Constant and Vezin. For simplicity, cheapness and efficiency that of Vezin is preëminent.

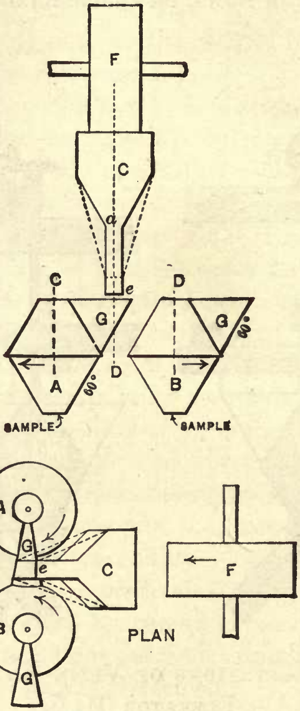
The Vezin sampler consists of two truncated cones of sheet steel, joined together at their bases, thus producing an apparatus in shape something like a can buoy. The upper cone has a scoop, of which the horizontal section has the shape of a sector of a circle. The machine is fixed on a vertical spindle, by which it is revolved. It is placed so that the scoop at each revolution cuts through the stream of ore to be sampled. The ore thus caught slides down into the lower cone and thence through a spout to a receiving bin. The upper cone, which carries the scoop, serves simply to prevent any stray pieces of ore from falling into the lower one. It is preferable to present the ore to the scoop by an inclined chute, say at 58°, rather than as a stream falling vertically. The frequency of the samples depends upon the number of revolutions of the scoop, 10 r. p. m. being an ordinary speed. The size of the sample depends upon the relation of the scoop to the circle of which it is a sector., e.g., if the diameter of the circle be 24 in. and it be desired to cut out one tenth the sector must be 3·54 in. at the arc. The width of the sector, however, must always be four times the diameter of the largest piece of ore. When the ore to be sampled is so fine that it may be cut down to a small percentage without any further crushing, two samplers are commonly arranged in series, a suitable mixing device being interposed between the two. The second sampler thus receives the sample, thoroughly mixed, which is cut out by the first.



FIGS. 60 AND 61.—ELEVATIONS OF VEZIN AUTOMATIC SAMPLER AND ELEVATOR HEAD.

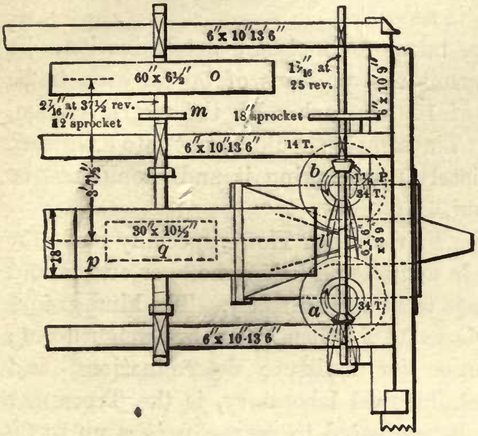
The Vezin sampler and its method of installation are shown in the accompanying engravings. The machine is supplied by the F. M. Davis Iron Works Co., of Denver, Colo., but can be built in any machine shop from drawings which can be obtained from its inventor, H. A. Vezin, of Denver, Colo. The cost of a machine installed is usually about $200.

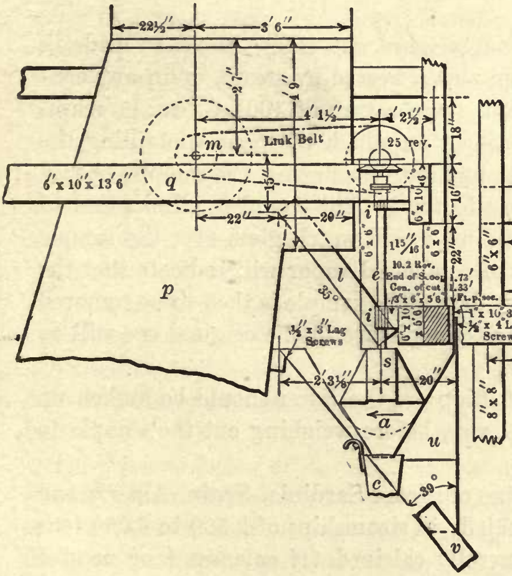
Arrangement of Mechanical Samplers.—The installation of any mechanical sampling apparatus involves an elevation of the whole quantity of ore in order to establish a falling stream in which the machine may be interposed. If the ore has to be crushed it will in all probability have to be raised anyway in order to screen it. A belt and bucket elevator is the means usually employed. With such an arrangement the elevation of a good many tons of ore to a considerable height is by no means expensive. Even under unfavorable conditions the cost of elevation to 30 ft. ought not to exceed 0·5c. per ton. In the case of a zinc ore already crushed very fine,

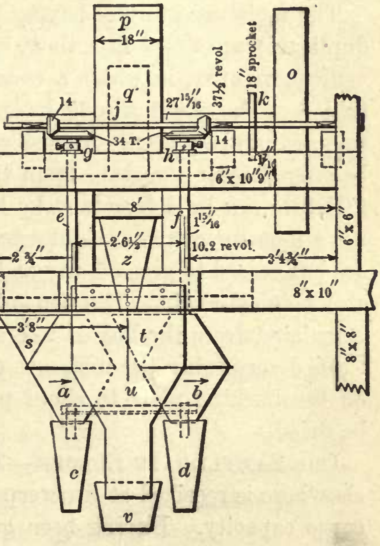


FIGS. 62 AND 63.—ELEVATION AND PLAN SHOWING ARRANGEMENT OF VEZIN AUTOMATIC SAMPLERS IN DUPLICATE.

an automatic sampling system would consist simply of a belt and bucket elevator and the cutting out machines. However, unless it were necessary to elevate the ore to storage bins, or for some other purpose in which the sampling process intervened merely as an incident, the cost of sampling mechanically in the manner suggested above might be more than that of sampling by hand as the wagons were unloaded. If on the other hand the ore had to be elevated anyway, so that the cost of handling the ore could be charged properly to another process, the cost of sampling mechanically would be cheaper than by hand. In a works where a considerable quantity of ore has to be sampled, the cost per ton is insignificant.







FIGS. 64-66.
ARRANGEMENT OF VEZIN AUTOMATIC SAMPLERS IN DUPLICATE.

FIG. 64.—PLAN.
FIG. 65.—SIDE ELEVATION.
FIG. 66.—FRONT ELEVATION.

DETERMINATION OF MOISTURE.—In buying ores it is important to determine the percentage of hygroscopic moisture which they contain, especially if they be raw concentrates, which are always wet, while even calcined ores will contain as much as 1% of water if they have been exposed to the atmosphere for any considerable length of time. The moisture sample should be taken immediately after the weighing of the ore. A grab sample is made to answer for this purpose. The method is conceded to be inaccurate, but by common agreement it is accepted.

A rather large sample should be taken for moisture determination. It should be at least sufficient to permit two portions of 50 oz. each to be weighed out, and preferably two of 100 oz. each. By the metric system, 1 or 2 kg. are convenient weights. The sample should be put into a tin can, with closely fitting cover, immediately upon taking it and should not be exposed until it is to be weighed out.

The scales commonly used are the Fairbanks or Howe, specially made for this purpose, which are graduated in ounces and half ounces, or ounces and twentieths, and enable the percentage to be read directly. The kind graduated in ounces and twentieths reads to 0·1% when 50 oz. are weighed out; it costs $15. An excellent balance for moisture determinations, and other experimental work in a metallurgical laboratory, is the Troemmer solution scale, costing $17, which is graduated to grams, weighs up to 20 kg., and is extremely accurate.

The moisture samples having been weighed out (they should be made in duplicate) are dried in a sheet iron closet, heated by steam, or in any convenient manner, in which a constant temperature of 100° C. can be maintained. The closet should have shelves on which the pans containing the samples can be placed. In order to facilitate the drying, the samples should be stirred with a spatula from time to time. The complete elimination of moisture can be determined by holding a cold watch glass over the sample for a moment or two; the absence of condensed vapor will indicate that the ore has ceased to give off any and is dry. The sample is then to be removed and weighed again. The percentage of moisture in the original ore will be calculated from the loss of weight.

In determining the moisture of lump ore the latter should be broken up on the bucking plate to about pea size, before weighing out the sample to be dried.

ORE SAMPLING IN EUROPE.—Zinc ore from Sardinia, Spain, Algeria and elsewhere is received at Antwerp chiefly in steamships of 2,500 to 3,000 tons cargo capacity. Having been generally calcined (if calamine) or roasted (if blende) at the mines where it was produced, it arrives at Antwerp in the condition of fines. It is unloaded into railway cars on the wharf, whence it is despatched directly to Liège, Stolberg and elsewhere. The method of discharging cargoes is crude. The ore is hoisted from the ship's hold in bags or baskets, which are weighed separately by means of beam scales. The railway wagons carry about 10,000 kg. each and a basket of ore weighs about 50 kg. Generally eight baskets per wagon, or one in 25, are reserved as a sample, the sample basket being chosen at random by agreement of the representatives of the buyer and seller, or more commonly the agent to whom both parties have entrusted their interests in this part of the business. The sample baskets are carried to a shed, where their contents are mixed, quartered down and divided into three parts for analysis in the usual manner. Settlements are made in Antwerp on that basis.

Valuation of Zinc Ores.

There are various formulæ employed by European smelters to determine the value of zinc ore, all of them being based on the London quotation for spelter, the tenor of zinc in the ore, the loss of zinc in smelting and the cost of carriage and smelting. The factors assumed to represent loss in treatment and cost of treatment are of course arbitrary, and vary according to the character of the ore and requirements of individual smelters. The price of zinc at Antwerp, Liège, Hamburg, Breslau and elsewhere is governed by the London market. The price of zinc at London is accepted as reported in the "London Commercial Report" of the *Public Ledger,* or otherwise as may be agreed, the average price of the month of the ship's arrival being commonly taken. The terms of settlement are also a matter of mutual agreement, but the buyers generally offer 80% of the value of the consignment on ship's arrival, and the remainder after agreement of weights, assays and the basis price of spelter.

In the United States there is comparatively little ore sold by the use of formulæ which introduce all the factors affecting the value of the mineral, but there is an increasing tendency toward the adoption of that method. Whether the transactions are based on an arbitrary bid or on a sliding scale, however, the governing factor is the price of prime Western spelter at St. Louis.

The determination of the value of zinc ore by means of a formula which constitutes an automatic sliding scale, based on the tenor of zinc in the ore and its value in that crude form, is by all means the fairest and generally the most satisfactory method to both seller and buyer.

Custom of the Joplin District.—In the Joplin district of the United States the buying and selling of ore is customarily effected by weekly transactions. The old method of completing these is still in use to a considerable extent. The smelters who consume the Joplin ores have agents in the district, who visit weekly the various mills and after a visual examination of the concentrated ore displayed in the bins make a bid of so much per ton for the week's output. The miner sells to the highest bidder and transactions are settled by cash payment on Saturday. The smelter's bid is for the ore in the mill bins, so the cost of carting, usually 25 to 50c. per ton, falls on him. Another local peculiarity is that no determination is made of the moisture contents of the ore and the settlement is made on the wet weight, though if the ore be excessively wet a certain arbitrary deduction in weight may be made by mutual agreement. Ordinarily the coarse concentrates of the district contain 2 to 3% water, while the slime (locally called "sludge") holds 10 to 18% when fresh. This crude system of buying and selling ore is the outgrowth of the local conditions of mining and smelting, in neither of which has skilled technical advice and supervision been employed to a considerable extent until recently.

Since 1899 blende concentrates have been settled for in the Joplin district to some extent upon the basis of their tenor in zinc, as shown by sample and assay. The system of sampling employed consists in taking a few shovelfuls of the ore after it has been loaded in the railway car, which are mixed together and quartered down to about two pounds. This is divided into three parts, one of which is assayed by the buyer and one by the seller, the third being reserved against accident or disagreement. If the assays agree within a few tenths of a per cent, the average is taken as the basis of settlement. Moisture is determined in the usual manner. The selling of ore on an assay basis has led to an improvement in the grade of the product of those miners who have adopted the custom and obtain thereby a guide as to the efficiency of their mill work. Sometimes the ore is sampled in the bins at the mill by boring with a tube through the bed of ore in numerous places.

The calamine ores of Aurora and Granby are not yet sold upon an assay basis. They fetched $10@$14 per ton f. o. b. cars in 1901, their tenor in zinc ranging from 40 to 48%. Their value fluctuates more or less according to the price of spelter. In 1899 when spelter was quoted at 6c. calamine assaying about 44% Zn commanded $25 per ton f. o. b. cars.

SLIDING SCALES.—Contracts for ore are frequently made in Europe, particularly for standard ores of well-known composition, on a sliding scale up and down from an arbitrary basis. In such cases a certain value is established for a certain grade of ore at a certain price for spelter at London and that basis price is changed to correspond with fluctuations in the price of the metal in the London market, an addition or deduction being made for each unit of zinc above or below the standard. This method is illustrated in the following offer for blende concentrates, which was made by a Belgian smelter in the early part of 1902. A basis price of 132·5 fr. per 1,000 kg. of ore ex ship at Antwerp, when spelter is £18 per 1,000 kg. at London, is established. For each unit of zinc below 50 there is a deduction of 5 fr.; for each unit above 50 the price is increased 4 fr.; consequently the value of ore assaying 60% Zn would be 172·50 fr. when spelter is at £18, London. An addition or deduction of 12·50 fr. is made to or from the basis price for each variation of £1 in the price of spelter. Thus an ore assaying 50% Zn would be worth 120 fr. at £17, while an ore assaying 60% Zn would be worth 160 fr. The London quotation of spelter in pounds sterling per 2,240 lb. is divided by 1,015 to obtain the equivalent per 1,000 kg. For a unit of iron in excess of 4% a deduction of 3 fr. is made from the value of the ore; for each unit in excess of 5% the deduction is 5 fr. Such contracts involve naturally the cost of smelting and the loss in treatment, but it is extremely difficult to formulate them so that they will be equally fair to both parties in all fluctuations of the price of metal and variations in the grade of the ore.

The effect of the varying factors which determine the value of an ore is allowed for in numerous formulæ, which greatly simplify the calculation. These formulæ represent algebraically the percentage of zinc in the ore, the loss in smelting, the market price of spelter at London, Breslau or St. Louis and the cost of carriage on ore, smelting and the smelter's profit. Some of these formulæ are analyzed as follows:

I. Buyers of zinc ore in Antwerp often use the formula

$$V = \frac{(T - TL) \times (P - 2{\cdot}50)}{10} - S,$$

in which

V = value of ore in francs per 1,000 kg.
T = units of zinc in ore.
L = loss in treatment expressed decimally or fractionally.
P = price of spelter in francs per 100 kg.
S = cost of carriage and smelting, and smelter's profit, commonly referred to as the "returning charge."

If the price of spelter be £18 at London and exchange 25·20, P will be 44·68. Assuming $T = 50$, $L = 0{\cdot}20$, and $S = 60$ fr., the value of an ore assaying 50% Zn would be 108·72 fr. per metric ton ex ship at Antwerp.

This formula introduces the number of units of zinc recoverable in smelting the ore, represented by the expression $T - TL$, which is equivalent to 0·8 T. The market price of spelter is represented by the expression $P - 2{\cdot}50$, in which P is the equivalent in francs per 100 kg. of the London price quoted in pounds sterling per 2,240 lb.; a deduction of 2·50 fr. per 100 kg. is made to cover freight on spelter from the works, commissions, insurance, etc. Inasmuch as the tenor of zinc in the ore is represented by units of 10 kg. each, and since that number is multiplied by the value of 10 units of spelter, it is necessary to divide the product by 10 before deducting the returning charge. The latter will vary according to the grade and character of the ore, the requirements of the smelter, etc. It will be relatively low in this formula, because the smelter will realize a profit on the zinc recovered over and above what he pays for according to the expression $T-TL=80\%$.

II. Another formula introducing a modification to discriminate between the value of calamine and roasted blende is the following:

$$a \text{ For calamine: } V = \frac{(0{\cdot}95\ P \times 0{\cdot}8T)}{10} - S.$$

$$b \text{ For roasted blende: } V = \frac{[(0{\cdot}95\ P\ \times (0{\cdot}8\,T - 1)]}{10} - S.$$

in which the symbols have the same meaning as in formula No. 1. The value of a calamine ore under the same conditions as in formula No. 1 would be determined by formula No. 2 to be 109·78 fr., while roasted blende of the same tenor in zinc would be worth 105·54 fr.

Formulæ II*a* and II*b* are identical with I, save that the expression 0·95 P is substituted for P—2·50 to allow for the charges on the spelter product, and an additional unit is deducted from 0·8 T in the case of the roasted blende to compensate for the lower extraction of zinc from that class of ore.

III. In Europe at the present time zinc ore is frequently purchased on a sliding scale based on the formula

$$V = [0{\cdot}95\ P \times \frac{(T-8)}{100}] - S.$$

in which

V = value of ore per ton of 2,240 lb. ex ship at Swansea, Antwerp or Hamburg.
P = average price of spelter (good ordinary brands) at London during month of delivery of the ore.
T = number of units of zinc contained in the ore as shown by assay.
S = returning charge.

This constitutes a scientific and equitable basis. The discount from the London price of the metal corresponds more or less to the cost of delivering at London the smelter's product—i.e., 0·95 P represents the value of the metal at the works where produced. The expression T—8 covers the loss in smelting, and takes into account the proportionately higher loss in smelting low grade ores. If T=60, the smelter pays for 86⅔% of the zinc in the ore; if T=50, he pays for 84%. The costs of carriage of the ore from the seaport to the works, the cost of smelting and the smelter's profit are covered practically by the returning charge, S, inasmuch as there will be little or no margin over T—8. The returning charge will vary, of course, with the character of the ore, market conditions, etc. In 1900, a bid for raw blende assaying 60% Zn was £3 10s. per 2,240 lb.; another bid made a returning charge of only £3. When spelter was worth £22 at London, a returning charge of £3 10s. made the value of 60% ore=£7 7s. 4d. ex ship at Swansea, Antwerp or Hamburg=$35·848 per 2,240 lb. (reckoning exchange at $4·866)=$32 per 2,000 lb.

This formula is the same as II*a*, save that the percentage of zinc paid for in the ore is represented by T—8 instead of 0·8 T. In order to give the value of the ore in francs per metric ton it would be made

$$V = \frac{(0{\cdot}95\,P \times (T-8)}{10} - S.$$

in which P is the value of spelter in francs per 100 kg.

In order to obtain the value of ore in dollars per 2,000 lb. the modification of the formula would be

$$[20(T-8) \times P] - S,$$

in which P is the value of spelter in cents per pound.

Inasmuch as so high a percentage of zinc is paid for under this formula, the returning charge has to be made large enough to cover all costs, besides the smelter's profit.

IV. Ore buyers in Swansea, Wales, used to employ a rule,[1] which is reduced to the following formula:

$$V = W\,[P\,(0{\cdot}8\,\frac{T-1}{100-C}) - S] - L.$$

in which

V = value of ore per 2,240 lb.
W = percentage of ore remaining after calcination, expressed decimally.
P = price of spelter per 2,240 lb. at London, less £1.
T = tenor of zinc in ore, expressed in units.
C = loss in calcination expressed in units.
S = smelting charge, considered to be £2 10s.
L = cost of calcination, assumed to be 5s.

Assuming an ore assaying 40% Zn, which will lose 30% of its weight in calcination, its value when spelter is worth £17 at London will work out by the above formula as follows:

$$\frac{40-1}{100-30} = \frac{39}{70} = 0{\cdot}557.$$

0·557 × 0·8 = 0·446.
0·446 − 0·01 = 0·436.
0.436 × £16 = £6.976 = £6 19s 6d.
£6 19s 6d − £2 10s = £4 9s 6d.
0·70 × £4 9s 6d = £3 2s 8d.
£3 2s 8d − 5s = £2 17s 8d = net value of the ore per ton.

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[1] H. D. Hoskold, Trans. Fed. Inst. Min. Eng., V, i and ii, 93.

An analysis of this formula shows it to be less complicated than appears at first sight. The expression (T—1)÷(100—C) gives the percentage of zinc in the calcined ore, on which the value of the metal is calculated and against which the smelting charge is deducted. The smelter receives an allowance of one unit and pays for only 80% of the remainder, which allows for the difference in loss between ores of different grades and leaves besides a certain margin of profit. It is proper to make a fixed deduction from the London price rather than a percentage deduction, inasmuch as the freight charges at least remain constant, irrespective of the selling price. The value having been calculated on the calcined ore the result must be reduced to terms of raw ore by considering the relation between the weight of the raw and calcined ore, which is done by introducing the factor W.

V. In Upper Silesia calamine used to be bought by the formula[1]

$$V = \frac{(T \times P \times 0{\cdot}66) - 101.}{100}$$

in which

V = value of ore per 100 kg.
P = price of zinc at Breslau per 100 kg. less 0·5 mark.
T = units of zinc in ore.

If spelter be worth 32·6 marks per 100 kg. at Breslau, which is equivalent to approximately £18 per 2,240 lb. at London, the value of an ore assaying 20% Zn would be [(20×32·1×0·66)—101]÷100=3·227 marks per metric centner=32·27 marks per metric ton=$6·97 per 2,000 lb. Similarly, the value of an ore assaying 10% Zn would be 1·109 marks per centner=11·09 marks per metric ton=$2·40 per 2,000 lb.

In the case of blende a deduction of 7·5 marks per metric ton=$1·62 per 2,000 lb. is made to cover the cost of roasting.

VI. A contract for blende concentrates produced in Joplin, Mo., assaying about 60% Zn and not more than 2% Fe was effected in November, 1899, on the following basis, f. o. b. mines:

$$V = [16T \times (P - 0{\cdot}20)] - S$$

in which

V = value of ore in dollars and cents per 2,000 lb.
T = units of zinc in ore.
P = price of prime Western spelter at St. Louis in cents per pound.
S = charge for carriage and smelting of ore, etc.

In this formula the factor 16 represents the product of the number of pounds in a unit, namely 20, by the assumed percentage of recovery of zinc namely 0·80. The factor S varied within certain limits, according to

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[1] Kosmann, Oberschlesien, sein Land und seine Industrie, p. 153.

the value of spelter and also whatever percentage of iron there might be in the ore over 2%. P—0·20 represented the value of spelter at the works in Kansas. This formula is substantially the same as No. 1.

VII. A schedule to determine the price of ore, known as the Paxton scale, was adopted in 1899 by the Missouri and Kansas Zinc Miners' Association, which corresponded to the formula

$$V=700\ P$$

in which

V = value of ore per 2,000 lb. assaying 60 % Zn.
P = value of prime Western spelter at St. Louis in cents per pound.

Thus if the price of spelter were 6c. per lb., the price of ore would be $42 per ton; with spelter at 5c. per lb. ore would be $35, and so on. This formula was quite impractical, inasmuch as it made the value of zinc in ore bear a constant relation to the value of the marketable metal, while no account was taken of the fact that the cost of smelting remains unaffected by fluctuations in the value of the metal, wherefore its proportion in the cost of production increases as the cost of the ore goes down. For example, if the cost of carriage, smelting, etc., be $10 per ton of ore and 80% of the zinc be recovered, 2,000 lb. of 60% ore will yield 960 lb. of metal, which at 6c. per lb. would be worth $57·60. The cost of production if the ore were bought according to the schedule of the Miners' Association would be $42 for ore, plus $10 for freight and smelting, a total of $52.[1] If the price of spelter were 5c. per lb. the value of the product would be $48 and the cost $35+$10=$45. If the price were 4c. per lb. the value of the product would be $38·40 and the cost $28+$10=$38. Consequently, the smelter who had made a profit of $5·60 per ton of ore when spelter was worth 6c. would make only 40c. with spelter at 4c., and at a lower price would suffer a loss. Such a schedule to determine the price of ore is obviously unscientific; after a few months of trial the Missouri and Kansas Miners' Association was compelled to abandon it.

The formula, V=700 P, determined only the value of ore assaying 60% Zn. For ores of other grades, $1 per ton was added or deducted for each unit of zinc above or below 60, but when the tenor in zinc fell below 53% a deduction of $1·50 was made. Iron and lead were each penalized $1 per unit in excess of 1%, up to 4% Fe and 3% Pb.

VIII. The Paxton schedule of 1899 was subsequently modified by its author. As a matter of interest, the revised schedule submitted by George

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[1] These figures are merely illustrative and are not to be taken as representing the actual cost of smelting or recovery of metal in Missouri and Kansas.

B. Paxton, of Joplin, Mo., under date March 22, 1900, is herewith presented. This schedule was intended for clean, merchantable blende, free from iron, lead or excess of moisture. According to Mr. Paxton, it is based on the actual cost of smelting the various grades of ore, with the addition of a small profit to the smelter. The price of spelter at St. Louis, Mo., is assumed as the basis.

Lead and iron in the ore were penalized as follows: For each unit of lead, 25 lb. were to be deducted from the weight of the ore—i.e., if 2,000 lb. of ore assaying 1% Pb were offered for sale, the smelter would pay for only 1,975 lb. according to the above schedule. For 1% Fe 45 lb. were deducted, and for each unit in excess of 1% 55 lb. additional. An ore assaying 6% Fe would therefore be taxed 320 lb. A special contract was required for ore assaying more than 6% Fe.

The prices established by this schedule are considerably below those which have prevailed in the Joplin district during a good deal of the time since its publication. The recent practice in buying ore there has been to make a certain basis price for the grade assaying 60% Zn and pay for higher or lower grades at an advance or deduction of $1 per unit. The basis price varies according to the price of spelter, the competition among the smelters and other fluctuating conditions.

IX. A formula used in 1900 by Kansas and Missouri smelters for the purchase of Joplin ore was

$$V=(16T\times P)-S$$

in which

V = value of ore per 2,000 lb.
T = units of zinc in ore.
P = value of zinc per pound at St. Louis.
S = returning charge.

The returning charge included cost of carting, freight, smelting and commissions, lumped at $10·50, and the smelter's profit. The latter was made equivalent to 100 P on ore assaying 60% Zn or upward, and $100\,P+[10\,P\times(60-T)]$ on lower grade ore. Consequently, with spelter at 4c. the smelter would reckon a profit of $4 per ton on 60% ore and $4+ ($0·40×10) =$8 per ton on 50% ore.[1]

This formula is substantially the same as Nos. 1, 2*a*, 3 and 6. A recovery of 80% of the zinc in the ore is reckoned and on that basis the assay in units multiplied by 16 gives the number of pounds of zinc to be paid for. The necessary discount from the St. Louis value of the metal is included in the returning charge. The greater margin demanded by the smelter in the

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[1] W. George Waring, Eng. & Min. Journ., July 14, 1900.

PAXTON SCHEDULE OF ZINC ORE PRICES.

Price of Spe'ter per 100 lbs.	50%	51%	52%	53%	54%	55%	56%	57%	58%	59%	60%	61%	62%	63%	64%	65%
3·00	7·50	8·28	9·06	9·84	10·62	11·40	12·18	12·96	13·74	14·52	15·30	15·78	16·26	16·74	17·22	17·70
3·05	7·80	8·59	9·38	10·18	10·97	11·76	12·56	13·35	14·14	14·94	15·73	16·21	16·70	17·19	17·68	18·17
3·10	8·10	8·90	9·71	10·51	11.32	12·12	12·93	13·73	14·54	15·35	16·16	16·65	17·15	17·64	18·14	18·64
3·15	8·40	9·21	10·03	10·85	11.67	12·49	13·31	14·13	14·95	15·77	16·59	17·09	17·60	18·09	18·60	19·11
3·20	8·70	9·53	10·36	11·19	12·02	12·86	13·69	14·52	15·35	16·18	17·02	17·53	18·04	18·55	19·06	19·58
3·25	9·00	9·84	10·69	11·53	12·38	13·28	14·07	14·91	15·71	16·60	17·45	17·97	18·49	19·01	19·53	20·05
3·30	9·30	10·15	11·01	11·87	12·73	13·59	14·44	15·30	16·13	17·02	17·88	18·40	18·93	19·46	19·99	20·52
3·35	9·60	10·47	11·34	12·21	13·08	13·95	14·82	15·69	16·56	17·43	18·31	18·84	19·38	19·91	20·45	20·99
3·40	9·90	10·78	11·67	12·55	13·43	14·32	15·20	16·09	16·97	17·86	18·74	19·28	19·83	20·37	20·92	21·41
3·45	10·20	11·09	11·99	12·89	13·79	14·68	15·58	16·48	17·38	18·28	19·17	19·72	20·27	20·82	21·37	21·93
3·50	10·50	11·41	12·32	13·23	14·14	15·05	15·96	16·87	17·78	18·69	19·60	20·16	20·72	21·28	21·84	22·40
3·55	10·80	11·72	12·64	13·57	14·49	15·41	16·33	17·26	18·18	19·10	20·03	20·59	21·16	21·73	22·30	22·87
3·60	11·10	12·03	12·97	13·91	14·84	15·78	16·72	17·65	18·59	19·53	20·46	21·03	21·61	22·18	22·76	23·34
3·65	11·40	12·34	13·29	14·24	15·19	16·14	17·09	18·04	18·99	19·94	20·89	21·47	22·05	22·64	23·22	23·81
3·70	11.70	12·66	13·62	14·58	15·54	16·51	17·47	18·43	19·39	20·35	21·32	21·91	22·50	23·09	23·68	24·28
3·75	12·00	12·97	13·95	14·92	15·90	16·87	17·85	18·82	19·80	20·77	21·75	22·35	22·95	23·55	24·15	24·75
3·80	12·30	13·28	14·27	15·26	16·25	17·24	18·22	19·21	20·20	21·19	22·18	22·78	23·39	24·00	24·61	25·22
3·85	12·60	13·60	14·60	15·60	16·60	17·60	18·60	19·60	20·60	21·60	22·61	23·22	23·84	24·45	25·07	25·69
3·90	12·90	13·91	14·93	15·94	16·95	17·97	18·98	20·00	21·01	22·02	23·04	23·66	24·29	24·91	25·53	26·16
3·95	13·20	14·22	15·25	16·28	17·30	18·33	19·36	20·38	21·41	22·44	23·47	24·10	24·73	25·36	25·99	26·63
4·00	13·50	14·54	15·58	16·62	17·66	18·70	19·74	20·78	21·82	22·86	23·90	24·54	25·18	25·82	26·46	27·10
4·05	13·80	14·85	15·90	16·96	18·01	19·06	20·11	21·17	22·22	23·27	24·33	24·97	25·62	26·27	26·92	27·57
4·10	14·10	15·16	16·23	17·30	18·36	19·43	20·50	21·56	22·63	23·70	24·76	25·41	26·07	26·72	27·38	28·04
4·15	14·40	15·47	16·55	17·63	18·71	19·79	20·87	21·95	23·03	24·11	25·19	25·85	26·51	27·18	27·84	28·51
4·20	14·70	15·79	16·88	17·97	19·06	20·16	21·25	22·34	23·43	24·52	25·62	26·29	26·96	27·63	28·30	28·98
4·25	15·00	16·10	17·21	18·31	19·42	20·52	21·63	22·73	23·84	24·94	26·05	26·73	27·41	28·09	28·77	29·45
4·30	15·30	16·41	17·53	18·65	19·77	20·89	22·00	23·12	24·24	25·36	26·48	27·16	27·85	28·54	29·23	29·92
4·35	15·60	16·73	17·86	18·99	20·12	21·25	22·38	23·51	24·64	25·77	26·91	27·60	28·30	28·99	29·69	30·39
4·40	15·90	17·04	18·18	19·33	20·47	21·61	22·75	23·90	25·04	26·19	27·34	28·04	28·75	29·45	30·15	30·86
4·45	16·20	17·35	18·51	19·67	20·83	21·99	23·14	24·30	25·46	26·62	27·77	28·48	29·19	29·90	30·61	31·33
4·50	16·50	17·67	18·84	20·01	21·18	22·35	23·52	24·69	25·86	27·03	28·20	28·92	29·64	30·36	31·08	31·80
4·55	16·80	17·98	19·16	20·35	21·53	22·71	23·90	25·08	26·26	27·44	28·63	29·35	30·08	30·81	31·54	32·27
4·60	17·10	18·29	19·49	20·68	21·88	23·07	24·26	25·45	26·64	27·36	29·06	29·79	30·53	31·26	32·00	32·74
4·65	17·40	18·60	19·81	21·02	22·23	23·44	24·65	25·86	27·07	28·28	29·49	30·23	30·98	31·72	32·47	33·21
4·70	17·70	18·92	20·14	21·36	22·58	23·81	25·03	26·25	27·47	28·69	29·92	30·67	31·42	32·17	32·92	33·68
4·75	18·00	19·23	20·47	21·70	22·94	24·17	25·41	26·64	27·88	29·11	30·35	31·11	31·87	32·63	33·39	34·15
4·80	18·30	19·54	20·79	22·04	23·29	24·54	25·78	27·03	28·28	29·53	30·78	31·54	32·31	33·08	33·85	34·62
4·85	18·60	19·86	21·12	22·38	23·64	24·90	26·16	27·42	28·68	29·94	31·21	31·98	32·76	33·53	34·31	35·09
4·90	18·90	20·17	21·45	22·72	24·00	25·27	26·54	27·82	29·09	30·36	31·64	32·42	33·21	33·99	34·78	35·56
4·95	19·20	20·48	21·77	23·05	24·34	25·63	26·91	28·20	29·49	30·78	32·07	32·86	33·65	34·44	35·23	36·03
5·00	19·50	20·80	22·10	23·40	24·70	26·00	27·30	28·60	29·90	31·20	32·50	33·30	34·10	34·90	35·79	36·50
5·05	19·80	21·11	22·42	23·74	25·05	26·36	27·68	28·99	30·30	31·62	32·93	33·73	34·54	35·35	36·16	36·97
5·10	20·10	21·42	22·75	24·07	25·40	26·72	28·05	29·37	30·70	32·03	33·36	34·17	34·99	35·80	36·62	37·44
5·15	20·40	21·73	23·07	24·41	25·75	27·09	28·43	29·77	31·11	32·45	33·79	34·61	35·44	36·26	37·09	37·91
5·20	20·70	22·05	23·40	24·75	26·10	27·46	28·81	30·16	31·57	32·86	34·22	35·05	35·88	36·71	37·54	38·38
5·25	21·00	22·36	23·73	25·09	26·46	27·82	29·19	30·55	31·92	33·28	34·65	35·49	36·33	37·17	38·01	38·85
5·30	21·30	22·67	24·05	25·43	26·81	28·19	29·56	30·94	32·32	33·70	35·08	35·92	36·77	37·62	38·47	39·32
5·35	21·60	22·99	24·38	25·77	27·16	28·55	29·94	31·33	32·72	34·11	35·51	36·36	37·22	38·07	38·93	39·79
5·40	21·90	23·30	24·71	26·11	27·51	28·92	30·32	31·73	33·13	34·53	35·94	36·80	37·67	38·53	39·40	40·26
5·45	22·20	23·61	25·03	26·44	27·86	29·28	30·69	32·11	33·53	34·95	36·37	37·24	38·11	38·98	39·85	40·73
5·50	22·50	23·83	25·36	26·49	28·22	29·15	31·08	32·50	33·14	35·37	36·80	37·68	38·56	39·44	40·32	41·20
5·55	22·80	24·24	25·68	27·13	28·57	30·01	31·46	32·90	34·34	35·79	37·23	38·11	39·00	39·89	40·78	41·67
5·60	23·10	24·55	26·01	27·46	28·92	30·38	31·33	33·29	34·74	36·20	37·66	38·55	39·45	40·34	41·24	42·14
5·65	23·40	24·86	26·33	27·80	29·27	30·74	32·21	33·68	35·15	36·62	38·09	38·99	39·90	40·80	41·71	42·61
5·70	23·70	25·18	26·66	28·14	29·62	31·11	32·59	34·07	35·55	37·03	38·52	39·43	40·34	41·25	42·16	43·09
5·75	24·00	25·49	26·99	28·48	29·98	31·47	32·97	34·46	35·96	37·45	38·95	39·87	40·79	41·71	42·63	43·55
5·80	24·30	25·80	27·31	28·82	30·33	31·84	33·34	34·85	36·36	37·87	39·38	40·30	41·23	42·16	43·09	44·02
5·85	24·60	26·12	27·64	29·16	30·68	32·20	33·72	35·24	36·76	38·28	39·81	40·74	41·68	42·61	43·55	44·49
5·90	24·90	26·43	27·97	29·50	31·03	32·57	34·10	35·64	37·17	38·70	40·24	41·18	42·13	43·07	44·02	44·96
5·95	25·20	26·74	28·29	29·83	31·38	32·93	34·47	36·02	37·57	39·12	40·67	41·62	42·57	43·52	44·47	45·43
6·00	25·50	27·06	28·62	30·18	31·74	33·30	34·86	36·42	37·98	39·54	41·10	42·06	43·02	43·98	44·94	45·90
6·05	25·80	27·37	28·94	30·52	32·09	33·66	35·24	36·81	38·38	39·96	41·53	42·49	43·46	44·43	45·40	46·37
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6·25	27·00	28·62	30·25	31·87	33·50	35·12	36·75	38·37	40·00	41·62	43·25	44·25	45·25	46·25	47·25	48·25
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6·35	27·60	29·25	30·90	32·55	34·20	35·85	37·50	39·15	40·80	42·45	44·11	45·12	46·14	47·15	48·17	49·19
6·40	27·90	29·56	31·23	32·89	34·56	36·22	37·88	39·55	41·22	42·88	44·54	45·56	46·59	47·61	48·64	49·66
6·45	28·20	29·87	31·55	33·22	34·90	36·58	38·25	39·93	41·61	43·29	44·97	46·00	47·03	48·06	49·09	50·13
6·50	28·50	30·19	31·88	33·57	35·26	36·95	38·64	40·33	42·02	43·71	45·40	46·44	47·48	48·52	49·56	50·60

PAXTON SCHEDULE OF ZINC ORE PRICES (Continued).

Price of Spelter per 100 lbs.	50%	51%	52%	53%	54%	55%	56%	57%	58%	59%	60%	61%	62%	63%	64%	65%
6·55	28·80	30·50	32·20	33·91	35·61	37·31	39·02	40·72	42·42	44·13	45·83	46·87	47·92	48·97	50·02	51·07
6·60	29·10	30·81	32·53	34·24	35·96	37·67	39·39	41·10	42·82	44·54	46·26	47·31	48·37	49·42	50·48	51·54
6·65	29·40	31·12	32·85	34·58	36·31	38·04	39·77	41·50	43·23	44·96	46·69	47·75	48·82	49·88	50·95	52·01
6·70	29·70	31·44	33·18	34·92	36·66	38·41	40·15	41·89	43·63	45·37	47·12	48·19	49·26	50·33	51·40	52·48
6·75	30·00	31·75	33·51	35·26	37·02	38·77	40·53	42·28	44·04	45·79	47·55	48·63	49·71	50·79	51·87	52·95
6·80	30·30	32·06	33·83	35·60	37·37	39·14	40·90	42·67	44·44	46·21	47·98	49·06	50·15	51·24	52·33	53·42
6·85	30·60	32·38	34·16	35·94	37·72	39·50	41·28	43·06	44·84	46·62	48·41	49·50	50·60	51·69	52·79	53·89
6·90	30·90	32·69	34·49	36·28	38·08	39·87	41·66	43·46	45·25	47·05	48·84	49·94	51·05	52·15	53·26	54·36
6·95	31·20	33·00	34·81	36·61	38·42	40·23	42·03	43·84	46·65	47·46	49·27	50·38	51·49	52·60	53·71	54·83
7·00	31.50	33·32	35·14	36.96	38·78	40·60	42·42	44·24	46·06	47·88	49·70	50·82	51·94	53·06	54·18	55·30

case of the low grade ore compensates for the increased cost of treatment and lower recovery of metal from it.

VALUE OF LEAD AND SILVER BEARING ZINC ORES.—In general the presence of lead in a zinc ore detracts from its value, since the cost of smelting is likely to be increased by the more rapid destruction of retorts (and consequent greater loss of zinc), while the quality of the spelter produced is deteriorated by the lead which distils over with the zinc, and its value per pound is lessened to an extent which is not offset by the increase in the weight of metal produced. For those reasons a penalty is frequently imposed on lead in ore when it exceeds a stipulated limit, which in Kansas is sometimes fixed at 1%, the penalty being generally 50c.@$1 per unit for lead in excess of that amount. A penalty of $1 per unit is much too high, the extra cost entailed upon the smelter by a lead content in the ore being ordinarily a good deal less than that. Frequently lead is not taxed at all. The smelters of Belgium distil regularly mixtures of ore which are rather high in lead and recover a portion of that metal by jigging the residues drawn from the retorts and smelting the concentrate, whereby silver is recovered together with the lead. In the purchase of such ores the smelter of course reckons upon the value of the lead and silver recoverable, minus the additional cost of the process, and makes his price accordingly.

DEDUCTION IN VALUE ON ACCOUNT OF IRON.—Western smelters generally deduct $1 per unit (sometimes only 50c.) of iron in excess of a certain limit, which is fixed variously at 1% or 2%. In the Joplin district an ore containing 5% Fe and upward is marketed with more or less difficulty, the smelters generally preferring the ores of higher grade and purity. In Europe, where no ore so clean as that of Joplin is available, mixtures containing 5% to 10% Fe+Mn are habitually smelted. Manganese plays the same part in zinc smelting that iron does, and the units of manganese shown by assay are added to the units of iron in computing the value of an ore.

CONCLUSION.—Besides the contents of the ore in zinc, lead, silver, iron, manganese and other valuable or deleterious elements, numerous other factors enter into consideration in determining its value. The smelter aims for various metallurgical reasons to treat a product of a certain uniform composition, which must generally be prepared by making a mixture of various ores. Competition may necessitate that he has to pay a comparatively high price for the scarce, high grade material, wherefore he will scheme to recoup himself by buying the more plentiful, inferior ores at a relatively low price. The average cost at which he is able to secure his desired mixture will form the basis of his calculations.

The actual cost of smelting is determined by the experience in treating the mixtures which are regularly distilled. It varies naturally from time to time according to the wages for labor, cost of coal, and other conditions. In the cost of smelting, all such general charges as taxes, insurance, and the expense of administering the business must be included. Furthermore, there must be an allowance for amortization of the cost of the works, which will not last indefinitely without replacement of their parts at various intervals. The smelter expects to realize a profit over and above the cost of the ore and the expense of smelting it, including all charges, which will be a fair return upon the money invested in the business, and he looks for a return which will be sufficiently large to reimburse him for the numerous risks that must be taken in this, as in any other manufacturing enterprise.

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THE
ORE DEPOSITS
OF THE
United States and Canada

By James Furman Kemp, A.B.E.M.

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