AN

ELEMENTARY TEXT-BOOK

OF

METALLURGY.

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PREFACE TO THE SECOND EDITION.

THIS edition has been revised, and some slight alterations have been made, otherwise no change has been found to be necessary, as experience has shown the book to be suitable for the purpose for which it was intended. The Author has to thank friends for pointing out some errors, which have been corrected, and also his assistant, Mr. Hugh Campbell, for assistance in the revision.

GLASGOW, *August*, 1899.

PREFACE.

THIS little work is intended for the use of Students—both for such as are commencing the study of Metallurgy professionally, and for those who require a general knowledge of the subject as part of their Engineering education—and also for those who, being already engaged in Metallurgical Industries, desire some knowledge of the principles on which the processes they use are based.

Special attention is given to principles, select processes only are described, and no more details are given than are necessary to the understanding of the principles.

The Student who has carefully gone through this book should next read Prof. Roberts-Austen's *Introduction to the Study of Metallurgy*, after which he will be able to study with profit special treatises on that branch of the subject in which he is most interested.

A short Course of Practical Work is added for those who have access to a metallurgical laboratory.

GLASGOW, *December*, 1894.

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CHAPTER I.

INTRODUCTION—PROPERTIES OF THE METALS.

Nature and Scope of Metallurgy.—Metallurgy has been defined as "the art of extracting metals from their ores, and adapting them to the various processes of manufacture." According to this definition metallurgy is an art, not a science; a science is a collection of truths, an art is the application of these to useful purposes. A vast number of facts concerning the metals, their properties, and the means by which they can be extracted from their ores, have been discovered. These constitute what may be called the science of metallurgy, whilst the application of the facts to practical purposes is the art.

The facts, which thus form the science, do not belong to metallurgy alone, but mostly fall within the domain of chemistry; indeed, metallurgy has sometimes been regarded as a branch of applied chemistry.

A student who wishes to understand metallurgical operations, must have mastered at least the rudiments of chemistry, and should have some knowledge of physics, especially of heat, and of mechanics.

Ancient Metallurgy.—Metallurgy is the most ancient of the arts, and many of the facts on which the science of chemistry is based were discovered by the early smelters of metals. In the earliest historical times men were acquainted with six metals only—viz., gold, silver, copper, lead, tin, and iron; mercury was discovered early in the historical period. The methods for the smelting and purifying of these metals discovered by the ancients were in many instances very similar to those now in use.

The Metals.—Of the seventy (or thereabouts) known elements about fifty-six are regarded as metals, and possess certain properties in common. It is, however, quite impossible to say what properties are essential to metals, or to frame a satisfactory definition; indeed, whether certain of the elements are to be regarded as metals or not depends entirely on which property is selected as the characteristic of a metal.

Of the metals a few only are common, the rest are rare, so rare as to be, in many cases, mere chemical curiosities; only about seventeen are of any importance in the arts, and with these only is the metallurgist concerned.

The useful metals may, for convenience, be classified into the following six groups:—

1. IRON (including all forms of Iron and Steel).
2. THE HEAVY METALS.—Iron, Copper, Lead, Zinc, Tin.
3. THE RARER HEAVY METALS.—*Nickel, Cobalt, Manganese, Chromium.*
4. THE NOBLE METALS.—Gold, Silver, Mercury, *Platinum.*
5. THE METALS OF THE ALKALIES AND EARTHS.—*Sodium, Potassium, Magnesium, Aluminium.*
6. THE SEMI-METALS.—*Arsenic, Antimony, Bismuth.*

Those metals, the names of which are printed in italics, are of less importance, and will not be considered in this volume.

The value of the metals in the arts depends on the possession of certain properties, physical and chemical.

A. PHYSICAL PROPERTIES.

Density.—That the metals are heavy is a common statement, but one which is not always true, as some of them are lighter than water.

The most convenient way to state the relative weights of different bodies is to compare them with the weight of an equal bulk of some standard substance taken as unity. For practical purposes the standard selected is always water. The weight of any substance, the weight of an equal volume of water at 15·5° C. (60° F.) being taken as 1, is called its density or specific gravity.

Since 1 cubic foot of water weighs 1,000 ounces, the weight in ounces of a cubic foot of any other substance having a specific gravity S, will be 1,000 × S.

SPECIFIC GRAVITY OF SOME OF THE METALS.

Platinum,	21·50	Iron,	7·79
Gold,	19·50	Tin,	7·29
Mercury,	13·59	Zinc,	7·10
Lead,	11·45	Aluminium,	2·67
Silver,	10·50	Magnesium,	1·75
Copper,	8·96	Sodium,	·97
Nickel,	8·80	Lithium,	·59

These numbers represent the specific gravities of the pure metals; and many of them are only approximations. The specific gravities of the commercial metals are often somewhat different on account of differences in physical condition and the presence of impurities.

Mechanical treatment, such as hammering or rolling, usually increases the density by forcing the molecules more closely together, but in cases where it induces a crystalline structure the density may be lowered.

For some purposes density is of considerable importance. Mercury is used for barometers because, being very heavy, only a short column of it is required to balance the pressure of the atmosphere. The difficulty of making spurious gold coins depends on the very high density of gold, and where great lightness is required, as in small weights, &c., aluminium is often used.

Malleability "is the property of permanently extending in all directions without rupture, by hammering or rolling;" it is opposed to brittleness. This property is possessed to a very high degree by some metals, whilst others are devoid of it. The metals can be arranged in series in order of malleability, but no numerical value can be attached to them as there is no unit in which measurements of this property can be made. Placing the most malleable metals first, the order is—

Gold.
Silver.
Copper.
Aluminium.
Tin.
Platinum.
Lead.
Zinc.
Iron.

Such a list can never be more than an approximation, as the order may be modified by circumstances. Change of temperature has a very marked effect, the metals usually, though not always, becoming more malleable as the temperature rises; copper becomes very brittle near its melting point, and zinc is only malleable through a small range of temperature. The presence of impurities, also, has often a very marked effect on the malleability of metals, a small quantity of antimony in lead, for instance, rendering the metal quite brittle.

There is every gradation between extreme malleability and brittleness. The first evidence of lack of malleability in a metal is that when hammered or rolled it cracks at the edges, whilst if the malleability is very deficient the piece will break up.

Ductility.—This is the property of "permanently extending by traction" and by virtue of it metals can be drawn into wire, and is closely related to, though not by any means identical with, malleability. The method of wire drawing is very simple.

A draw plate, which is a plate of steel perforated with conical holes, the smaller end of which is the size the wire is required to be made, is used. A rod of metal of suitable size is taken and pointed either by hammering or filing till the point can be pushed through the hole; it is then seized by a pair of pincers and pulled until the wire is drawn through the hole, whereby it is diminished in diameter and increased in length. The reduction in diameter at each draught will usually be small, so that in making a fine wire the metal has to be drawn through many dies one after the other.

The order of the metals for ductility is—

Gold.
Silver.
Platinum.
Aluminium.
Iron.
Nickel.
Copper.
Zinc.
Tin.
Lead.

The ductility is much influenced by temperature; some metals drawing best hot, others cold. The soft metals usually draw better than the hard ones; except when the former are very deficient in tenacity, as in the case of lead.

Flow of Metals.—Metals seem at first sight to be perfectly rigid, but they will flow under very great pressure, the molecules having a certain amount of freedom of motion. On this depend malleability and ductility, and also the power possessed by some metals of being impressed by means of a die. In coining, striking medals, and some other operations, a blank or smooth piece of metal is placed in a press and engraved dies are forced by great pressure against it; the metal flows into every depression of the die and an exact copy is thus obtained.*

Annealing.—When metals have been rolled, hammered, or subjected to mechanical treatment they often become brittle. The brittleness can usually be removed by heating to redness and cooling. This operation is called annealing. The annealing temperature varies with different metals, and in some cases (*e.g.*, steel) the cooling must be slow, whilst in others (*e.g.*, copper) the rate of cooling is immaterial. The mechanical treatment seems to throw the molecules into a state of stress, and when the metal is heated the molecules, being to a certain extent freed, are enabled to settle themselves into a natural position and thus relieve the stress. The density of the metal is frequently altered by annealing, usually being diminished.

Tenacity.—This is the property of resisting being torn asunder, it depends on the cohesion of the molecules. It is one of the

* See Roberts-Austen, *Introduction to the Study of Metallurgy.*

most important properties of metals, for on it depends their value for structural purposes. The tenacity is measured by ascertaining the force required to tear asunder a bar of the metal of known sectional area.

The metal to be tested is cut into test pieces of suitable shape, either flat or round, the exact area of section of which is known. Various forms of test piece may be used, but whichever is selected care must be taken that the section is uniform over the portion where fracture may take place, or at any rate that there are no sudden changes of sectional area. Two marks are made on the bar, usually at a distance of eight inches apart. The bar is put into the machine and pressure is applied. The bar stretches and ultimately breaks. The pressure with which the bar breaks is called the "breaking weight," and the tensile strength is the breaking weight for each square inch of section. The principal elongation takes place near the point at which fracture occurs, so that the area of the fractured surface is very much less than the original sectional area of the test piece. The percentage by which the area diminishes before fracture is often measured, and is of value in determining the suitability of the metal for some purposes. The distance between the two marked points is measured, and the percentage by which the bar elongated before fracture is calculated. Thus, if the points were 8 inches apart on the original test piece, and 10 inches after fracture, the elongation would be 2 inches or 25 per cent. The original length of the test piece must always be stated, since almost the whole of the elongation occurs near the point of fracture.

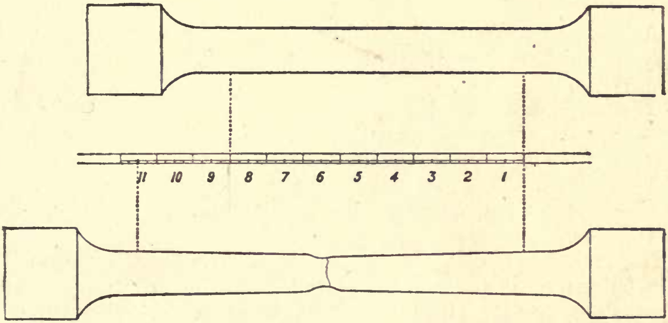

Fig. 1.—Steel Test Piece before and after fracture.
Tensile strength, 24 tons. Elongation, 31·25 per cent. on 8 inches.

The elongation is often of as much importance in judging of the quality of a metal as the tensile strength. As a rule, metals with very high tensile strength elongate very little, whilst the tough metals elongate very much. Anything which tends to harden a metal, as, for instance, rolling or wire drawing, usually tends to increase the tenacity; whilst anything which tends to soften it, as, for instance, annealing, tends to diminish the tenacity and increase its elongation.

The approximate relative tenacities of the metals may be taken as— *

Lead,	1
Tin,	1·3
Zinc,	2·0
Copper,	12 to 20
Cast iron,	7 to 12
Wrought iron,	20 to 40
Steel,	10 to 100

As a rule, tenacity diminishes as temperature rises, and it may be either increased or diminished by the presence of impurities.

Testing Machines.—Various forms of testing machines are in use. In all of these a small force is multiplied either by a system of levers or by hydraulic power.

Toughness.—This property is closely related to tenacity. It is the power of resisting fracture by bending, and is opposed to brittleness.

Hardness.—This is the property of resisting abrasion by scratching; on it depends the power of the metal to resist and wear away tools. The relative hardness of two substances is judged by rubbing them together, when the one which marks the other is the harder, or by comparing the ease with which they are scratched by a knife or file.

The following Table may be of use:—

Mercury,	Liquid.	
Sodium,	Soft as wax	(1).
Lead,	Scratched by the nail	(2).
Platinum, Copper, Gold, Silver, Tin,	Marked easily by a knife	(2-3).
Nickel, Iron, Antimony, Zinc,	Scratched by glass	(3-6).
Very Hard Steel,	Scratches glass	(over 6).

The numbers in the last column are the hardnesses according to Moh's scale, which is used by mineralogists.

Temperature modifies hardness very much, most metals becoming softer as the temperature rises. The presence of very small quantities of impurities may very much increase the hardness of a metal, as, for instance, antimony in lead.

Elasticity.—This is the property by virtue of which bodies

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* Thurston, *Materials of Engineering.*

after distortion tend to return to their original form. Popularly the term elasticity is used so vaguely as to be almost meaningless, it sometimes being taken to mean extensibility and at other times high elastic limit.

Elasticity is measured by the force per unit of area with which a body after it has been stretched, bent, or otherwise deformed tends to return to its original form, and this force is always equal to that which was necessary to cause the distortion, provided the distortion has not been too great.

If a bar of steel or any other metal (of 1 square inch section) be taken and a weight of 1 cwt. be attached to it, it will elongate a little; if another cwt. be added the extension will be doubled, and so on, the amount of extension being proportional to the extending force. If the elongation continued at the same rate, and the bar did not break, it would ultimately double its length. The weight which would be required to double the length of the bar if its elasticity remained perfect is called the Modulus of Elasticity or Young's Modulus. It need hardly be said that there is no metal which will actually double its length before fracture.

If after the first cwt. has been put on and the elongation noted, it be removed, the bar will return to its original length; if the 2 cwts. be put on the same result will be obtained; but if the process be continued, larger and larger weights being added, a point will be reached when the bar will no longer return to its original length when the weight is removed, but will have become permanently elongated, or "taken a permanent set." It has now been strained beyond its limit of elasticity. The limit of elasticity is that weight which, acting on a bar of 1 square inch sectional area, will just make it take a permanent set.

When the stretching force is less than the elastic limit, the bar will return to its original length when the stress is removed, and, therefore, within certain limits the elasticity of every metal is perfect.

Elasticity may be measured either in tension, by pulling the bar, or in compression.

	Breaking Weight.	Elastic Limit.	Modulus.
Cast iron, ...	11 tons	5 tons	7,000 tons.
Wrought iron, . . .	25 ,,	15 ,,	10,000 ,,
Mild Steel, ...	23 ,,	13 ,,	12,500 ,,
Steel, ...	42 ,,	22 ,,	11,200 ,,

Elasticity may be measured in the same testing machine used for measuring tensile strength, the weight being removed from time to time to see whether the elastic limit has been reached.

Crystallisation.—Many metals crystallise readily on cooling from fusion, and such are usually brittle; lead and tin, however, which are soft metals, crystallise readily. Metals precipitated by the electric current are usually in a very crystalline condition. A crystalline structure may sometimes be induced by vibration or by heating to a high temperature, whilst rolling and wire drawing tend to destroy it. Very crystalline metals are usually deficient in strength.

Colour.—The colour of the metals is not of much practical importance, except for ornamental purposes. The metals may be classified according to their colours.

1. White—	
α. Silver-white,	Silver.
β. Inclining to silver-white,	Platinum, aluminium, tin, nickel.
γ. ,, blue,	Antimony, lead, zinc.
δ. ,, pink,	Bismuth.
2. Gray,	Arsenic, pig-iron.
3. Yellow,	Gold, certain alloys.
4. Red,	Copper.

The colour is best seen on a polished surface.

Lustre.—Most metals reflect light very perfectly, and therefore have a high lustre; this, indeed, is so characteristic of the metals that such a lustre is often called metallic. It is not shown by metals in the rough or finely-divided condition, but may usually be obtained by polishing or burnishing. Mercury, though a liquid, has a perfect lustre, but in general it is best shown by the hard metals which will take a high polish.

Many minerals, such as galena and pyrites, have a metallic lustre.

Fusibility.—All the metals except mercury are solid at ordinary temperatures, but they can all be melted. The melting points of the common metals are:—

	Deg. C.	Deg. F.
Mercury,	−39	−39
Tin,	216	420
Bismuth,	254	490
Lead,	326	619
Zinc,	415	779
Silver,	954	1,749
Gold,	1,045	1,913
Copper,	1,054	1,929
Cast iron,	1,240	2,264
Platinum,	1,775	3,227

These numbers must only be taken as approximations; some of the higher ones are uncertain.

Metals may be classified according to their fusibility.

1. Fusible below redness,	Tin, lead, zinc.
2. Fusible at temperatures easily attainable,	Aluminium, copper, silver, gold.
3. Fusible at highest furnace temperature,	Nickel, manganese, iron.
4. Infusible at ordinary furnace temperatures,	Platinum and its allies.

In this connection Pouillet's scale of temperature may be of interest.

	Deg. C.	Deg. F
Incipient redness,	525	977
Dull red,	700	1,292
Cherry red,	900	1,652
Deep orange,	1,100	2,012
White,	1,300	2,372
Dazzling white,	1,500	2,732

These figures are only rough approximations.

Volatility.—This is the property of being converted into vapour by heat. Most metals are more or less volatile at high temperatures, even iron being volatilised in the electric arc, and in some cases the volatility must be taken into account, but for most purposes a metal that does not volatilise to any great extent at ordinary furnace temperatures may be considered as fixed. The metals which are sufficiently volatile to be distilled are arsenic, zinc, cadmium, sodium, and mercury.

Heat Conductivity.—The metals as a whole are good conductors of heat, though some are much better than others. The relative conducting powers may be taken as—

Gold,	100·0	Zinc,	36·0
Platinum,	98·4	Aluminium,	33·0
Silver,	97·3	Tin,	30·4
Copper,	87·8	Lead,	18·0
Iron,	37·4	Marble,	2·2

These numbers are only approximations, as the conductivity varies according to the purity of the metal and other circumstances. It is easy to pick out the good and bad conductors from the list.

Electric Conductivity.—The power of the metals to transmit an electric current varies much. As a rule, metals are as good conductors of electricity as they are of heat, but they vary much among themselves. The conducting power of the metals may be taken as—

Silver,	100	Steel,	14
Copper, pure,	99·5	Tin,	13·3
Copper,	90	Iron,	12·7
Gold,	71	Platinum,	9·77
Aluminium,	33	Lead,	7·76
Zinc,	25·9	Mercury,	1·61

As the temperature rises the conductivity falls or the resistance increases; and as the rate of increase is different with different metals the order will vary with the temperature. The resistance of iron doubles at 180° C., quadruples at 430° C., and becomes ninefold at 860° C.

The presence of impurities also has a very deleterious effect. This is specially noticeable in the case of copper, some commercial samples of which have a conductivity as low as that of iron.

B. CHEMICAL PROPERTIES.

Action of Oxygen.—Most metals have considerable affinity for oxygen, and will combine with it under suitable conditions.

Gold, silver, platinum, and mercury, which do not combine with oxygen either at ordinary temperatures or at a red heat, and the oxides of which are decomposed by heat alone, are called noble metals. All the other metals combine with oxygen at high temperatures, often burning brightly with the formation of oxides.

Most metals are not acted on by dry air, but in presence of air, moisture, and carbon dioxide, they rust or become tarnished, with the formation either of hydrate, as in the case of iron, or basic carbonates, as in the case of lead and copper.

Action of Acids.—Most metals are soluble in hydrochloric acid, hydrogen being evolved. Copper, lead, and silver are not acted on by hydrochloric acid, but dissolve in nitric acid with formation of nitrates and evolution of oxides of nitrogen. Tin and antimony are attacked by nitric acid, but instead of being dissolved are converted into insoluble hydrates. Gold and platinum are not attacked by any single common acid, but are dissolved at once by a mixture of nitric and hydrochloric acids, which is, therefore, called aqua regia. Sulphuric acid attacks most metals either in the cold, hydrogen being evolved; or with aid of heat, when sulphur dioxide is given off.

CHAPTER II

COMBUSTION, &c.

Sources of Heat.—Since almost all metallurgical operations are carried on by the aid of heat, the methods of producing this are of great importance to the metallurgist. Of all sources of heat there is but one available for practical use—that is, the combustion of fuel, though in very rare instances others, such as the electric current or the rays of the sun, may be used for special purposes.

Combustion.—When two substances enter into chemical combination heat is usually evolved. When this heat is sufficiently intense to produce light, combustion is said to take place, and one of the bodies is said to burn. In all ordinary cases the combustion is the union of some substance which we call a fuel with the oxygen of the air. The substance which combines with the oxygen is called a combustible and is said to burn; whilst the oxygen is called a supporter of combustion. These terms are not strictly accurate, since combustion is always a mutual action, in which both substances play an equal part; but they are convenient and are sufficiently correct for practical purposes. By a combustible is meant, practically, a substance which will combine with the oxygen of the air with sufficient energy to evolve light.

Conditions which Favour Combustion.—Combustion, being chemical action between two substances, can only take place under favourable circumstances. Certain conditions favour chemical action, and therefore combustion; whilst others hinder it or stop it altogether.

α. *The Combustible and the Air must be in Contact.*—This is so obvious that no explanation is necessary.

β. *The Temperature must be sufficiently High.*—For every combustible there is a temperature below which combustion cannot take place. If a jet of gas be turned on the gas escapes into the air, but does not ignite because the temperature is too low; apply a light or a piece of white hot metal, ignition takes place, and the combustion once started continues, because the heat evolved by the combustion is sufficient to keep up the temperature above the ignition point. The temperature at which ignition takes place varies with the nature of the combustible.

Some substances take fire at the ordinary atmospheric temperature, and can, therefore, only be kept if protected from contact with the air. Phosphorus takes fire at such a low temperature that for safety it is always kept under water. Ordinary fuels ignite at a bright red heat, whilst iron will burn if heated to whiteness.

If a substance which is burning be cooled below its ignition point it will be at once extinguished. A gas flame cannot pass through wire gauze, because the gauze being a good conductor carries away the heat so rapidly that the gas which passes through is cooled below its ignition temperature, and therefore cannot burn. Anything which cools a burning substance will check, and ultimately stop, combustion. The principal action of water in extinguishing a fire is by cooling, on account of the large quantity of heat absorbed by the conversion of the water into steam, though the displacement of the air by steam has also a very considerable effect.

γ. The Combustible must be in a Sufficiently Fine State of Division.—As a rule the finer the state of division of the combustible the more readily will combustion take place, because the more intimate will be its contact with the air. Very large lumps burn slowly, the surface exposed to the air being small. It often happens in great conflagrations that large bales of very combustible materials are only burned at the surface, because the air is unable to penetrate into the mass, whilst smaller masses of the same materials are completely destroyed. A fine powder of a combustible substance will not burn in mass because it lies so close that air cannot penetrate it; but if it be mixed with air, by means of a blast or otherwise, it will burn rapidly, or may even cause explosion.

Many substances which will not burn in mass will do so very readily if very finely divided. Iron and lead may be obtained in such a fine state of division that they take fire spontaneously on coming into the air.

Fineness of division is most perfect in gases, which, therefore, if combustible, burn very readily and when mixed with air form explosive mixtures.

Flame.—Combustibles burn in two very different ways. A piece of charcoal simply glows and smoulders away without any flame, whilst coal gas burns with a large brightly luminous flame. If the charcoal be burned in pure oxygen instead of in air it burns much more brilliantly, but still without flame; but if it be burned in mass, as in a blast furnace, a pale-blue flame is produced. Most solid fuels burn without flame, but all gases burn with flame.

Consider a jet of a gas, say hydrogen, burning in the air. Just at the top of the tube is a dark zone of cold unburnt gas escaping, and round outside the flame is the air, also cold; between the gas and the air is the zone, where the air and gas are mixing and combustion is taking place. This zone is the flame.

Flame, then, is burning gas, and solids never burn with a flame unless they are first converted into gas, in which case, of course, it is the gas which burns. Coal burns with a flame because the heat distils gas out of it, leaving a residue of coke. Charcoal in mass seems to burn with a flame, because carbon monoxide, which is a combustible gas, is formed by imperfect combustion of the carbon.

Luminous Flames.—Hydrogen burns with a flame which is very hot, but almost non-luminous; whilst the flame of coal gas is very luminous. This difference is due to a difference in the nature of the gases. Coal gas consists of a complex mixture of hydrocarbons (compounds of carbon and hydrogen), hydrogen, and other gases. When these burn, if combustion be complete, water and carbon dioxide are produced, both of which are gases at the temperature of the flame. If air be supplied in sufficient abundance to ensure this complete combustion at once, as in the Bunsen burner, the flame is non-luminous. Under ordinary circumstances the flame is luminous. The gas escaping from the burner is cold and forms a dark zone, exactly as in the case of hydrogen. Between this zone and the air the gases are mixing, and combustion is taking place forming a flame. Towards the centre of the flame the quantity of air which is present will be small and combustion will, therefore, be imperfect. When hydrocarbons burn in a limited supply of air, combustion is at first only partial, and the high temperature produced dissociates a portion of the hydrocarbons with separation of carbon. These separated particles being very intensely heated glow brightly, and thus impart luminosity to the flame; as they pass into the air they too burn, and produce a very hot, but only slightly luminous, mantle, which surrounds the luminous zone of the flame.

Smoke.—If the combustion of the carbon, separated during the early stages of the combustion of a gas, be prevented, it will be deposited as smoke or soot. If a cold surface be depressed into a luminous gas flame the gases in contact with it will be cooled, and soot will be deposited on it. The combustion of the carbon is frequently prevented either by cooling, by contact with a cold surface, or in any other way, or by a deficient supply of air, the result in either case being smoke. Smoke is always produced by the imperfect combustion of fuels containing carbon, and anything which tends to check the combustion will tend to cause smoke. In order that the production of smoke may be avoided, it is essential that the burning gas should be thoroughly mixed with air before its temperature is reduced below the ignition point of the separated carbon.

Domestic Fire.—As an illustration of the phenomena of combustion, the case of an ordinary domestic fire may be considered. Suppose the grate to be full of glowing coke, air enters by the bars and the coke burns, producing carbon dioxide and carbon monoxide, $C + 2O = CO_2 \cdot C + O = CO$. The carbon dioxide produced coming in contact with the red-hot carbon is reduced to carbon monoxide, $CO_2 + C = 2CO$, and the carbon monoxide coming to the surface of the fire, and being hot, burns with its characteristic pale-blue flame, $CO + O = CO_2$. Now suppose some fresh coal to be thrown on the fire, the blue flames disappear, and as the coal gets hot dense smoke is given off. If after a time the fire be stirred or a light be applied, the escaping gases take fire and burn with a bright smoky flame. The carbon monoxide, which burnt with a blue flame before the coal was put on, must still have been formed afterwards, but in passing through the cold coal it would be cooled below its ignition temperature, and, therefore, would not take fire, the combustible gases given off from the coal would likewise be at too low a temperature to ignite, and, therefore, would also escape unburnt up the chimney till the heat from below penetrated the mass of coal and ignited them. The actual amount of solid carbon carried away in smoke is small, but the conditions which allow of the production of smoke are just those which allow of the escape of combustible gases unconsumed, and, therefore, the possibility of considerable loss of heating power.

Draught.—If a fire is to be kept burning the products of combustion must be carried away and fresh air must be supplied. Where carbon is the fuel used the carbon dioxide produced occupies the same bulk as the oxygen in the air consumed, whilst when hydrogen burns, the water formed (as steam) occupies twice the bulk of the oxygen.[*] Hence at the ordinary temperature the products of combustion would occupy a slightly larger volume than the air used but would be considerably heavier. As, however, the gases leave the fire at a very high temperature they will be much lighter than the surrounding air. Gases expand $\frac{1}{273}$ of their volume at 0° C. for each 1° C. rise of temperature, so that at 273° the gas would double its volume and

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[*] $C + \underset{\square\square}{2O} = \underset{\square\square}{CO_2}$ $2H + \underset{\square}{O} = \underset{\square\square}{H_2O}$ $C + \underset{\square}{O} = \underset{\square\square}{CO}$.

therefore halve its density. The hot gases tend to rise up the chimney and thus produce a draught which removes the products of combustion and allows fresh air to flow in to take their place.

The draught depends on the difference in weight between the column of hot gas inside the chimney and that of the cold air outside. Obviously the higher the temperature of the gas in the chimney, and the taller the chimney, the greater will be this difference and the better will be the draught. It must be remembered that as all the heat carried away by the escaping gases is lost the temperature should be kept as low as possible without impairing the draught. Anything which tends to cool the chimney gases, such as the influx of cold air, loss of heat by radiation, and anything which tends to hinder their free flow will tend to diminish the draught, and thus impede combustion.

Heat of Combustion.—The unit which is taken in all measurements of heat is "that quantity of heat which will raise 1 lb. of water 1° C. (from 4° to 5° C.), or if British units be used 1° F. (from 39·2° to 40·2° F.).

The amount of heat which is evolved in any chemical reaction is fixed and definite, and is quite independent of the rate at which the reaction takes place. 1 lb. of carbon in burning to form carbon dioxide will evolve 8,080 units of heat. This is expressed by saying that "the calorific power of carbon burning to carbon dioxide is 8,080 units," or, more shortly, C P of carbon = 8,080. The carbon may smoulder away or it may be burned by a blast of air or oxygen; but, provided carbon dioxide be formed, the amount of heat evolved for each pound of carbon consumed will be the same. When the carbon burns to carbon monoxide the amount of heat evolved is much less, 1 lb. of carbon only giving 2,400 units of heat, or the calorific power of carbon burnt to carbon monoxide is 2,400. So that when carbon is burnt to carbon monoxide, only about one-third of the heat is produced which it could evolve if completely burned to carbon dioxide, therefore incomplete combustion may cause a loss of two-thirds of the heat of the fuel. When hydrogen burns it evolves a large amount of heat, 1 lb. giving 34,000 units. In this case there is no intermediate product, and when combustion is imperfect some of the hydrogen is left unconsumed

Calorific Power of Fuel.—The calorific power of a fuel is the number of units of heat which would be evolved by the combustion of 1 lb. of it. Almost all fuels contain carbon and hydrogen, and by assuming that in any fuel the constituents will give on combustion the same amount of heat which they would do if they were in the free condition, the calorific power of the fuel can be calculated from its chemical composition. This assumption is not absolutely correct, but the results obtained by means of it are near enough the truth for all practical purposes.

Suppose the fuel to contain no other combustible than carbon, then if c' = the amount of carbon in 1 lb. of the fuel the calorific power (C P) will be $c' \times 8{,}080$.

If the fuel contains both carbon and hydrogen, then if h' is the quantity of hydrogen in one part

$$\text{C P} = c' \times 8{,}080 + h' \times 34{,}000.$$

Example.—A fuel contains 70 per cent. of carbon and 30 per cent. of hydrogen. What will be its calorific power?

$$\begin{aligned}\text{C P} &= \cdot 7 \times 8{,}080 + \cdot 3 \times 34{,}000 \\ &= 5{,}656 + 10{,}200 = 15{,}856.\end{aligned}$$

If the fuel contain oxygen the calculation becomes a little more complex. The oxygen in the fuel is present, not in the free gaseous condition, but in combination with the other constituents of the fuel, either carbon or hydrogen. This carbon or hydrogen being already combined with oxygen, cannot combine with more; that is, it cannot burn. It is always assumed that the oxygen present in a fuel is combined with hydrogen, and as hydrogen combines with eight times its own weight of oxygen it is evident that the oxygen will have rendered non-combustible one-eighth of its own weight of hydrogen, Assuming a fuel to contain carbon, hydrogen, and oxygen, using c' and h', as before, and o' standing for the quantity of oxygen in 1 lb. of the fuel, then

$$\text{C P} = c' \times 8{,}080 + (h' - \tfrac{1}{8}\, o') \times 34{,}000.$$

Example.—A fuel contains 60 per cent. carbon, 24 per cent. hydrogen, and 16 per cent. oxygen, calculate its calorific power.

$$\begin{aligned}\text{C P} &= \cdot 6 \times 8{,}080 + \left(\cdot 24 - \frac{\cdot 16}{8}\right) \times 34{,}000 \\ &= 4{,}848 + 7{,}480 = 12{,}328.\end{aligned}$$

The amount of hydrogen present in excess is that required to combine with the oxygen present—*i.e.*, $(h' - \frac{1}{8}\, o')$ is called the *available hydrogen.*

Other constituents may usually be neglected, ash, nitrogen, and other non-combustible materials having no effect, except that the more there is of them the less combustible matter can there be in the fuel.

Evaporative Power.—In estimating the value of a fuel engineers usually state, not the number of units of heat which the combustion of 1 lb. of the fuel will evolve, but the number of pounds of water at 100° C. (212° F.) which would be converted into steam by it. This is called the *evaporative power* of the fuel. The amount of heat required to convert 1 lb. of water at 100° C. into steam at the same temperature is 537° C. (or 967° F.) units, and this is called the *latent heat of steam.*

In the case of a fuel containing carbon only, the evaporative power would be $\frac{c' \times 8{,}080}{537}$, which would give for pure carbon 15·05 lbs. In the case of hydrogen, or fuels containing it, another factor must be taken into account. Each pound of hydrogen burnt will form 9 lbs. of water, which will have to be evaporated, and will, therefore, use up heat which would otherwise be available for evaporating the water in the boiler. Further, in calculating calorific powers it has been assumed that the fuels were at 0°, so that each pound of water they contain or yield on combustion would have to be heated to 100° before it could be evaporated. The evaporative power of hydrogen would, therefore, be

$$\text{E P} = \frac{34{,}000 - (100 + 537) \times 9}{537} = 52{\cdot}6 \text{ lbs.}$$

The evaporative power of any fuel containing carbon, hydrogen, oxygen, and water may be obtained from the formula.

$$\text{E P} = \frac{c' \times 8{,}080 + \{(h' - \tfrac{1}{8}o') \times 34{,}000\} - (9h' + w') \times 637}{537}$$

the symbols c', h', and o' being used as already explained, and w' being the quantity of water contained in 1 lb. of the fuel.

Experimental Determination of Heating Power.—This is always done in the laboratory by burning a weighed quantity of the fuel in such a way that the heat evolved can be absorbed in a known weight of water, the rise of temperature of which can be measured. The instrument most commonly used is "Thomson's Calorimeter." The fuel is burned by a mixture of potassium chlorate and nitrate, which readily parts with its oxygen. The fuel mixed with the oxidising agent is put into a copper cylinder or furnace, which is placed on a stand; a short piece of slow match is placed on the top of the mixture and lighted; a cover is rapidly placed over and is held firmly by springs. The whole is put into a measured quantity of water, the temperature of which has been noted. Combustion soon takes place, the products of combustion escape by the holes at the bottom of the cover, and passing through the water impart their heat to it. The rise of temperature is then noted and the calorific power is calculated.

Example.—

Weight of water, ...	2,000 grammes.
Weight of fuel, ...	2 ,,
Temperature of water before experiment, .	15° C.
,, ,, after ,, . .	23·5° C.
Rise of temperature, ...	8·5° C.
Add 10 per cent. for loss of heat, . . .	·85
	9·35

The 2 grammes of fuel have heated 2,000 grammes of water 9·35°, and the calorific power of the fuel is, therefore, 9,350.

By using 2,000 grammes of water and 2 grammes of fuel, the rise of temperature × 1,000 gives the calorific power. If 1,074 grammes of water and 2 grammes of fuel be used, the number of degrees rise of temperature is the evaporative power of the fuel.

Centigrade and Fahrenheit Degrees.—The calorific power is very often given in Fahrenheit or British units. The amount of heat required to raise 1 lb. of water 1° F. is $\frac{5}{9}$ of that required to raise the same weight 1° C., so that 9 British units are equal to 5 Centigrade units.

Therefore

$$\begin{aligned} \mathrm{CP}^f &= \tfrac{9}{5}\,\mathrm{CP}^c \\ \mathrm{CP}^c &= \tfrac{5}{9}\,\mathrm{CP}^f \end{aligned}$$

where CP^c is the calorific power of any fuel in Centigrade units, and CP^f is the same in Fahrenheit units.

CHAPTER III.

FUELS.

Classification of Fuels.—All fuels in common use, with the possible exception of mineral oils, are of vegetable origin.

Fuels may be classified in various ways, the most convenient being according to the state in which they are used—solid, liquid, or gaseous—each class being subdivided into natural or prepared fuels, according as they are used in the condition in which they occur in nature, or undergo more or less preparation before use.

1. Solid Fuels:
   - *a.* Natural—
     1. Wood.
     2. Peat.
     3. Coal.
   - *β.* Prepared—
     1. Charcoal.
     2. Coke.
     3. Briquettes.
2. Liquid Fuels:
   - *a.* Natural—
     - Mineral oils.
   - *β.* Prepared—
     - Distilled oils.
3. Gaseous Fuels:
   - *a.* Natural—
     - Natural gas.
   - *β.* Prepared—
     - Coal gas.
     - Producer gas.
     - Water gas.

WOOD.

Wood was the first, and for many ages the only, fuel with which man was acquainted. It is still used in a few well-wooded districts in minor metallurgical operations; but it has long been abandoned in most countries, except for a few very special purposes, and even for these it is usually first converted into charcoal. Wood is the more or less hard material which constitutes the stems, branches, and roots of trees and shrubs. All vegetable tissue consists essentially of cellulose ($C_6H_{10}O_5$) with smaller quantities of other materials. As the plant grows older its tissues become hardened, and the cells become filled up by the deposition of other matters mostly richer in carbon and hydrogen than cellulose, and some of them containing nitrogen.

Wood always contains large quantities of water. When the plant is living the tissues are permeated with liquid sap, which is in constant circulation. This is greatest in amount in spring, when the growth is most vigorous, and least in winter. When the tree is felled it will retain this liquid, and, therefore, freshly-felled wood is very moist, often containing 50 to 60 per cent. of water, the quantity varying with the nature of the tree, its age, and the season of felling.

The wood is barked—*i.e.*, stripped of its bark—and left exposed to the air; it then loses much of its water and becomes air dried, in which condition it may retain from 15 to 25 per cent. of water.

Composition of Wood.—Cellulose contains, carbon 44·44, hydrogen 6·17, and oxygen 49·39 per cent., and, therefore, has no available hydrogen. The average composition of wood may be taken as—

	Dry.	Air dried.
Carbon, ...	50	40
Hydrogen, ...	6	4·8
Oxygen, ...	41	32·8
Nitrogen, ...	1	·8
Ash, ...	2	1·6
Water, ...	...	20·0
	100	100·0

All woods contain a certain amount of inorganic material, which is left as a residue or ash when the wood is burned. The amount of ash varies somewhat with different kinds of wood and in different parts of the tree, and is usually between 1 and 2 per cent. The bark and leaves contain more than the branches, and these more than the stem. The ash consists principally of carbonates of lime, potash, and soda, with small quantities of silica, oxide of iron, and phosphoric and sulphuric acids. The composition of the ash of wood is of little importance to the metallurgist.

Properties of Wood.—Wood is apparently lighter than water; this is due to its being very porous, for the materials of which it is composed are all heavier than water, and if the cells and spaces become filled with water the wood becomes "water-logged," and will then sink. Air-dried wood has a specific gravity varying from ·39 to over 1, with an average of about ·6, oak being about ·7075. Wood free from air spaces has a specific gravity of about 1·25.

Wood as a Fuel.—Wood is not a good fuel for most purposes, for several reasons:—

1. It contains, when air dried, a large quantity of water, which not only diminishes the percentage of combustible material, but which has to be evaporated, and, therefore, absorbs heat which might otherwise be used.

2. It contains a large amount of oxygen, and, therefore, very little available hydrogen, so that its calorific power is low. Taking the average analysis of dry wood on p. 20, the available hydrogen $\left(h' - \frac{1}{8}o'\right)$ is $\left(\cdot06 - \frac{\cdot41}{8}\right) = \cdot009$.

The calorific power would be—

$$\cdot5 \times 8{,}080 + \left(\cdot06 - \frac{\cdot41}{8}\right) \times 34{,}000 = 4{,}040 + 306 = 4{,}346$$

and the evaporative power—

$$\frac{\cdot5 \times 8{,}080 + \left(\cdot06 - \frac{\cdot41}{8}\right) \times 34{,}000 - (637 \times 9 \times \cdot06)}{537}$$

$$= \frac{4{,}040 + 306 - 344}{537} = 7\cdot4.$$

Taking the average sample of air-dried wood the results would be much lower, thus—

$$CP = \cdot4 \times 8{,}080 + \left(\cdot048 - \frac{\cdot328}{8}\right) \times 34{,}000 = 3{,}232 + 238 = 3{,}470$$

and

$$EP = \frac{\cdot4 \times 8{,}080 + \left(\cdot048 - \frac{\cdot328}{8}\right) \times 34{,}000 - (\cdot20 + (\cdot048 \times 9))\,637}{537}$$

$$= \frac{3{,}232 + 238 - 402}{537} = 5\cdot7.$$

3. It is very bulky, therefore occupies much space for storage, and requires large fireplaces for its combustion.

On the other hand, it has three advantages:—

1. It kindles very readily.
2. It burns with a very long flame.
3. Its ash is small in quantity, and comparatively free from deleterious substances.

PEAT.

Peat is the remains of certain low forms of plants which thrive in moist situations. These live and die, and their partially-decayed remains form the soil on which the next crop grows, so that the mass accumulates year after year till it may attain considerable thickness, and form a peat moss or peat bog. The newest peat will be at the top, and the oldest will be at the bottom; the latter may have undergone considerable change, whilst the former will have a composition very like that of the fresh plants. The peat near the top is usually light brown in colour, light and porous; that deeper is much darker in colour, often nearly black, harder, and more compact, and the individual plants of which it is composed are much less easy to distinguish. The composition of peat naturally varies with its age. It always contains a large quantity of water, sometimes as much as 60 or 70 per cent., and when air dried it still retains a considerable amount. The ash is often very high, as earthy material is readily deposited among the fibres from the water which is in continual circulation through the mass. The following analyses will give an idea of the composition of peat, but individual samples vary so much that it is quite impossible to give an average:—

	1.	2.
Carbon,	61·04	54·02
Hydrogen,	6·67	5·21
Oxygen and Nitrogen,	30·46	30·38
Ash,	1·83	9·73

The density of peat varies from about ·25 to ·9. Peat is a poor fuel, it has all the disadvantages of wood, and in addition has in many cases a very high ash, and falls to pieces as it burns.

COAL.

Coal is the only fuel of much practical importance in this country. It is the fossilised remains of plants which flourished many ages ago, and which have since been buried beneath accumulations of sedimentary matter, often of immense thickness. In most cases, though probably not in all, the forests, the remains of which now constitute the coal-beds, grew on the sites where the coal is now found.

It is very difficult to give a definition of coal which shall be accurate and sufficient, but the following, given by Dr. Percy, will answer all practical purposes:—"Coal is a solid, stratified mineral substance, black or brown in colour, and of such a nature that it can be economically burned in furnaces or grates;" or, perhaps, as being shorter, "Coal is any mineral substance used as fuel which is mainly made up of the remains of plants."

There are an enormous number of varieties of coal, and these vary so much and pass so gradually one into another that an accurate classification is impossible. Coals may be classified into four groups:—

1. Lignite, or brown coal.
2. Bituminous, or true coal.
3. Cannel coal.
4. Anthracite.

Lignite.—This may be regarded as partially mineralised coal, and, therefore, as being intermediate in composition between wood and the true coals. It is usually brown in colour, though sometimes quite black, and often retains quite distinctly the woody structure. The only British deposit of this material is that at Bovey Tracey in Devonshire, but there are abundant lignite deposits on the Continent.

An analysis of Bovey lignite given by Percy is—

Carbon,	66·31
Hydrogen,	5·63
Oxygen,	22·86
Ash,	2·27

Lignites are of more recent geological age than the true coals, the Bovey beds being probably Oligocene. Lignite is a poor fuel, owing to the large amount of oxygen which it contains, and is of very little importance metallurgically.

Bituminous or True Coals.—All the coals in common use in this country belong to this group. They are characterised by a black colour, and by burning with a more or less luminous and smoky flame. They occur in beds belonging to the carboniferous period, interstratified with shales.

When a coal is heated in a closed vessel gas and other volatile matter is given off and a residue or coke is left.

It is difficult to frame a satisfactory classification of coals owing to the great number of varieties, and to the fact that the common descriptive names are used very loosely, and with different meanings in different districts. The following, based mainly on the percentage of non-volatile matter left on distillation, will probably answer all practical purposes:—

1. Non-caking coals, long flame,	50 to 60 per cent. coke.
2. Long flame, gas coal,	60 ,, 68 ,, ,,
3. Coking coal, furnace coal,	68 ,, 74 ,, ,,
4. ,, short flame,	74 ,, 80 ,, ,,
5. Anthracitic or steam coal,	80 ,, 90 ,, ,,

1. *Non-caking Coals, Long Flame.*—This class includes the splint and other similar coals used for blast furnace work in Scotland and Staffordshire. They burn with a long smoky flame. The coke left on distilling a powdered sample in a closed vessel is pulverulent or only slightly coherent. The specific gravity,

unless the ash is very high, is about 1·25; the colour is brown or black; they often have a more or less shaly appearance, and are very hard. The amount of oxygen is large, the ratio of oxygen to hydrogen being from 4 to 1 to 3 to 1, and the calorific power is about 8,000 to 8,500 units.

2. *Gas Coals.*—These mostly have a fine black colour and a specific gravity of about 1·3; they burn with a long luminous flame, and the coke left on distillation, though solid, is porous and friable. They yield a very large amount of gas, often as much as 17 to 20 per cent., or 10,000 to 11,000 cubic feet per ton; the ratio of oxygen to hydrogen is from 3 : 1 to 2 : 1; and the calorific power about 8,500 to 8,800.

3. *Coking Coal proper, or Furnace Coal.*—These coals are black and shining; they yield on distillation from 68 to 74 per cent. of coke. On heating the coal softens, apparently fuses, swells up and yields a hard compact coherent coke, in which all trace of the original particles of coal is lost. The yield of gas is 15 to 16 per cent., the ratio of oxygen to hydrogen is about 1 : 1; and the calorific power is about 8,800 to 9,300.

4. *Short Flame Coking Coals.*—These coals are also black and shiny. On heating in a closed vessel they behave very much like the preceding group; they yield from 74 to 80 per cent. of a good coherent coke, give from 12 to 15 per cent. of gas, and burn with a much shorter flame than the preceding varieties. The oxygen ratio is about 1 : 1, and the calorific power is from 9,300 to 9,600.

5. *Anthracite Steam Coals.*—These coals are bright black; they soil the fingers very slightly, if at all; they have a specific gravity of about 1·35 to 1·4. They burn with a very short flame and produce very little smoke. On distillation they yield 80 to 90 per cent. of a very friable coke, and 8 to 12 per cent. of gas. The calorific power is 9,200 to 9,500, and the amount of oxygen they contain is very small.

Anthracite.—This may be taken as the extreme form of the coal series and occurs mainly in districts where the beds have been subjected to metamorphism. It has a fine black colour, a metallic lustre, sometimes showing iridescence (peacock coal), and does not soil the fingers. It has a close uniform texture. and often breaks with a conchoidal fracture. On distillation it gives off little or no gas; the residue is quite unchanged, showing no signs of coking, and amounts to from 90 to 98 per cent. of the weight of the coal. Anthracite is extremely difficult to ignite, very frequently crumbles to pieces as it burns, and gives a very high local temperature.

Cannel Coal.—This variety differs very much from all other

Composition of various Fuels.

	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.
Moisture,	20	15	49·45	7·18	8·11	1·35	33·7	2·32	10·06	50·8	·54
Volatile,	52	58		35·12	33·94	31·95		15·01			66·54
Coke,	28	27	50·55	57·7	57·95	66·7	66·3	82·67	89·94	49·2	32·92
Fixed carbon,	25	22	36·30	54·9	53·98	64·63	64·98	78·77	85·48	47·75	12·17
Ash,	3	5	14·15	2·8	3·97	2·07	1·32	3·90	3·46	1·45	20·73
Sulphur,	...	...	2·61	·88	1·15	·74	·88	·59	·83	1·76	·69

1. Wood (oak).
2. Peat (Irish).
3. Bovey lignite.
4. Splint coal.
5. Splint coal.
6. Newcastle gas coal.
7. Garscube coking coal.
8. Anthracite coal (blind coal).
9. Anthracite.
10. Florence cannel.
11. Boghead „

kinds of coal, so much, indeed, that its mode of formation is usually regarded as being also different. It is close, compact, and strong, and breaks with a smooth or conchoidal fracture. The lustre is usually dull, it does not soil the fingers; and often does not resemble ordinary coal in appearance. It yields 8,000 to 12,000 cubic feet of highly-illuminating gas per ton, whence its name cannel or candle coal. The residue left on distillation contains usually but little fixed carbon, and a very large amount of ash. Cannel coal does not coke, and on burning often cracks and falls to pieces with a crackling sound, whence the name parrot coal, which is sometimes given to it. As an extreme case of a cannel coal may be taken the Boghead Cannel or Torbanite, which yields on distillation about 15,000 cubic feet of gas per ton.

These coals may be regarded as being shales which have become saturated with coaly matter, and they pass gradually into mere bituminous shales. They are used for gas making, but very rarely as fuels.

Ash of Coal.—All coal when burned leaves a certain amount of inorganic material or ash. This is partly the remains of the ash of the plants of which the coal has been formed, and is partly due to added mineral matter. The ash consists mainly of silica, alumina, and oxide of iron, and is, therefore, quite different in composition to the ash of wood.

Coal frequently contains iron pyrites (called coal brasses) scattered through it, sometimes in thin layers, sometimes in nodules or other forms. When the coal is burned the sulphur burns away and the iron is left as red oxide,

$$2FeS_2 + 11O = Fe_2O_3 + 4SO_2.$$

A red ash usually indicates that the coal contains a considerable quantity of pyrites, and, therefore, is sulphurous. The absence of redness can, however, only be taken as an indication of the absence of pyrites when the total quantity of ash is low, as the red colour is readily hidden by a large amount of white earthy residue. Sulphur is also often present in fuel in the form of calcium sulphate, and probably also in organic combination. The ash always contains phosphoric acid as calcium phosphate.

Water and Gas in Coal.—Coal, as raised from the pit, contains a considerable quantity of water, part of this it loses on exposure to the air, retaining, however, 4 or 5 per cent. Freshly won coal also contains occluded in it a considerable quantity of gas, which is slowly given off on exposure to the air.

CHARCOAL.

When wood is heated without access of air it is decomposed, gaseous and liquid products of decomposition are given off, and a black porous residue of charcoal is left. Charcoal is usually regarded as being carbon; it is, however, by no means pure, but in addition to the ash always contains small quantities of hydrogen and oxygen. As an average about—

Carbon,	93
Oxygen,	1·5
Hydrogen,	2·5
Ash,	3

may be taken to be the composition of dry charcoal.

Good charcoal should ring when struck, it should show clearly the annual rings of growth of the wood; and should lose nothing but water on ignition. The specific gravity of charcoal is about ·2, varying according to the nature of the wood from which it is obtained. It is very porous, the actual density of the carbon of which it is composed being 2.

Charcoal absorbs water and gases very readily. It burns without flame, but smoulders away, leaving a perfectly white residue of ash, and once ignited it should continue to burn till it is completely consumed.

Charcoal Burning.—Charcoal for use as a fuel is always specially prepared for the purpose. The method which was used in this country, and which is still in use on the Continent in countries where wood is plentiful, is that of burning in heaps.

A dry level spot is selected, and in it are driven three vertical stakes, so arranged as to form an equilateral triangle, and these are kept apart by short cross pieces. These three stakes form the chimney round which the wood is to be piled. The wood is barked, cut into pieces of uniform length of 2 feet 6 inches or so, and these are arranged round the chimney in rings, each ring being made to slope a little more than the one within. On the top of these is arranged another layer so as to form a second tier, and above this is placed a layer of brushwood or small branches. Round the bottom of the pile are driven a series of Y sticks, short distances apart, and on these are laid cross pieces; the whole surface of the heap is then covered with a layer of turf, grass side inwards, resting on the cross pieces. Lighted brushwood is put down the central chimney to start combustion, the wood soon takes fire, and when well alight the top of the chimney is closed with turf, so that combustion can only take place very slowly. Large quantities of steam are evolved, and this partially condensing on the inner side of the cover runs down and escapes at the bottom of the heap. This stage, owing to the large amount of water produced, is called the "sweating" stage, and lasts one or two days. The cover is then pressed firmly down with a pole; if large hollows are found a portion of the cover is removed, the hollows are filled up with brushwood, and the cover is replaced as quickly as possible; the opening round the bottom of the pile is then closed and the whole is left to itself for several days. At the end of this period openings are made round the bottom of the heap and also through the cover, near the top, say at A. From these upper openings clouds of dense yellow smoke escape, which gradually become thinner and are at length replaced by a pale bluish haze. These openings are now closed, and another ring opened lower down, where the same phenomena take place; and so on till the last ring of openings is made near the base of the pile; when these are stopped the openings at the bottom of the pile are also stopped, and it is left closed for several days. The cover is then removed as rapidly as possible, the charcoal is drawn and quenched either with water or sand, this operation being always performed at night in order that sparks may be more readily seen and quenched.

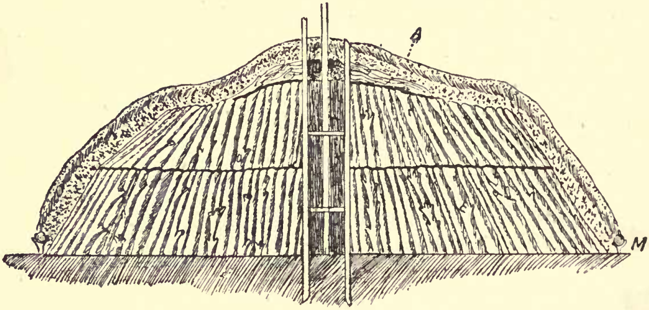


Fig. 2.—Charcoal Burning in Heaps.

The arrangement of the wood may be varied, the pieces being placed horizontally; or, as in Sweden, the heaps may be made rectangular instead of circular, but the principle is the same in all cases.

Theory of the Process.—This is very simple. During the first or sweating stage the fire is confined to the chimney and centre of the heap; the heat given off dries the wood and a large quantity of water is expelled. During the second stage the heat is distributing itself more uniformly through the heap and the drying is being completed. When at the commencement of the third stage openings are made at the bottom and top of the pile, a current of air is at once set up, air entering at the base and escaping through the hole in the cover. The air will take the path of least resistance, which will be along the base of the heap where the ends of the logs are not packed close together, and then upwards where the wood has been partially charred, and, therefore, has contracted, leaving larger spaces. As the air passes through, combustion begins with vigour, and the wood in the neighbourhood of the air current is charred, dense smoke being evolved; when the charring of that portion is completed the smoke ceases, and is replaced by the gases of burning charcoal. As soon as this happens the opening is closed and another is made lower down. The zone of combustion is thus lowered, charring goes on lower down, and the process is repeated. When the heap is burning there will thus always be three zones—1st, charcoal, where the charring is complete; 2nd, wood in process of charring, the zone of combustion being just at the junction of these two zones; and 3rd, dried wood very slightly charred.

If the process is properly conducted the heat should be derived largely from the combustion of the gases given off by the charring wood, which must find their way into the zone of combustion before they can escape. A considerable quantity of charcoal is, however, always burned.

Charcoal Burning in Kilns.—Charcoal is made in the United States in beehive-shaped kilns. These are about 27 feet in diameter and 21 feet high. At the bottom is a door and three series of air-holes and in the roof is a charging door. The wood (pine) cut into lengths of about 4 feet is put in through the door, and when above the level of this through the opening in the roof. A wood fire is kindled in the door, and as soon as it is well alight, the doors are closed and luted with clay. The supply of air is regulated by opening or closing the air-holes; the air enters by the lower row, and the smoke, &c., escapes by the upper ones. The process occupies about eight days.

Charcoal Making in Retorts.—Charcoal is now usually made as a bye-product in the manufacture of pyroligneous acid and wood spirit. The wood is distilled in closed iron retorts, the volatile products being condensed. The charcoal thus obtained is usually much inferior to that produced in heaps, principally because the wood used is not specially selected for yielding a good charcoal.

Yield of Charcoal.—The yield varies very much, depending on the nature of the wood and the way in which the process has been conducted; the slower the charring the larger will be the yield. The yield by weight, in charcoal burning in piles, is about 20 per cent., and by volume about 60 per cent. The yield in retorts is considerably larger.

The temperature at which the charring takes place has considerable influence on the quality of the charcoal, and especially on its ignition temperature; the higher the temperature of preparation, the higher will be the temperature required for ignition.

Peat Charcoal.—Peat may be charred, but it yields a charcoal with too little cohesion to be of much use. A fairly good charcoal can, however, be made by thoroughly pulping the peat, removing any hard fragments, compressing the mass into solid blocks by very great pressure, and then charring these at a moderate heat in superheated steam or otherwise.

COKE.

Coke bears the same relation to coal that charcoal does to wood, and is obtained by similar processes. The best coke is prepared specially, but coke is also obtained as a bye-product in the manufacture of coal gas. Coke for fuel was at one time made by coking in heaps, the coal being made up into round or rectangular piles and coked much in the same way as wood is converted into charcoal, but it is now almost invariably made in ovens.

The Beehive Coke Oven.—This is the form of oven most generally used. The dimensions and mode of working vary somewhat, but the principle is always the same. One form which gives very satisfactory results is shown in Fig. 3.

The oven is a circular chamber built of fire-brick and covered by a brick dome. At the crown of the dome is an opening which can be closed by a damper and can also be put in communication with the flue. Over the top of the ovens are arranged rails on which the trucks bringing up the coal can run. In front of the oven is an opening for discharging the coke. The charge having been withdrawn the oven will be at a dull red heat, an inner or false door, E, of fire-brick is built up, and the charge of ~~7 tons of~~ finely-divided coal is let in from the trucks through the opening in the roof, and the charge is levelled with a rake. The cover, C, is placed on and luted; the damper, D, is placed so as to put the oven in communication with the flue; and the front opening is built up with slabs or bricks set in fireclay, a hole, H, being made through the slabs near the top through which the process can be watched, and which can be closed by a plug of clay when not in use. In about four hours distillation commences, dense white smoke escapes from the flue and ignites. An opening, O, is now made in the clay at the bottom of the door, so that air can enter and pass up between the outer and inner door to the top of the coal. The gases given off take fire and for the next twelve hours, as distillation proceeds downwards, the gases burn with a dull red smoky flame; the temperature gradually rises till the interior of the oven is bright red hot; the flame now gradually gets redder, less gas is given off, and the quantity of air admitted is reduced; during the fourth twenty-four hours the flame becomes almost white and the quantity of air is gradually reduced; by the end of this period the flame disappears, distillation is over, and the charge has become a mass of white hot coke. A very small quantity of air is admitted for another six hours, the damper is then closed, all the openings are well luted with clay so as to exclude all air, and after another twelve hours the charge is ready for drawing. The outer door is taken down, and water is thrown in through a hose pipe for forty minutes, by which time the charge is cool enough for drawing;

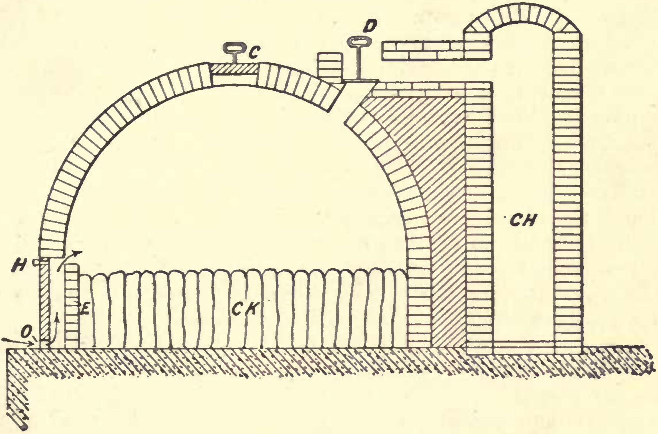


Fig. 3.—Beehive Coke Oven.

E, Inner or false door.
O, Air-hole in outer door.
H, Sight-hole.
C, Charging hole.
D, Damper.
C K, Coke.
C H, Chimney.

the inner door is then taken down and the coke is drawn out with long rakes. The coke is cooled with water when necessary, and shovelled into iron barrows with pronged shovels, which act as riddles and separate small coke and dust.

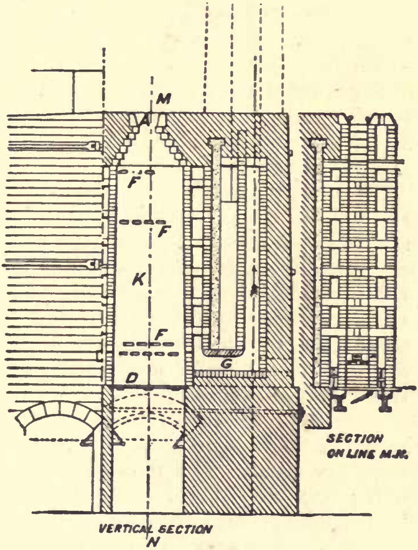
The coke comes out in irregular prismatic masses, the whole thickness of the charge; it is gray, with a metallic lustre; and is hard and dense. The yield is about 62 per cent., and the time of working a charge is five days. To prevent loss of heat by radiation the ovens are built in blocks of 50, two rows of 25 each, back to back. A soft black coke "smithy char" is made from the same coal in the same ovens in twelve hours, the yield being about 80 per cent.

The heating in this type of oven takes place almost entirely from above, the gases given off by the coal burn in the space above the charge; the gases given off from the lower layers have to pass upwards through the hot coke, and thus the heavier hydrocarbons are broken up into carbon which is deposited in the coke and gases which escape. A considerable quantity of fixed carbon at the surface of the charge is always burned, though the inner door diminishes this action very considerably.

The Appolt Oven.—Many ovens have been designed in which this combustion of fixed carbon is prevented, usually by excluding air altogether from the coking chamber, and burning the gases in an external combustion chamber.

One of the best known ovens of this type is that of the Bros. Appolt, which is now extensively used. This oven consists of a series of vertical ovens, or retorts, arranged in two rows in blocks of twenty-four or more. Each retort or chamber has its own walls, and is surrounded by a combustion space, with which it communicates by a series of openings usually arranged at three different levels. The retorts are about 16 feet high, 4 feet long, 13 inches wide at the top, and 1 foot 6 inches wide at the bottom; each is provided at the top with a charging hopper, and is closed at the bottom by folding iron doors; underneath is a vault with rails, along which trucks can be run to receive the coke when it is discharged. The ovens are built of fire brick; and, to strengthen them, cross bricks are built in passing across the combustion chamber, either to the other retort or to the external wall. The retort having just been discharged, and being red hot, a charge of 1 ton of powdered coal is dropped in and the charging door is at once closed, distillation commences, the evolved gases pass through the holes in the retort into the combustion spaces, where, meeting air, they burn, and thus heat up the retorts. When coking is complete, which is in about twenty-four hours, the doors at the bottom of the retort are opened and the charge is allowed to fall into an iron truck placed ready to receive it. The work is usually so arranged that one retort is discharged and charged again each hour, so that a continuous supply of gas may be kept up.

In this oven the temperature is kept up entirely by the combustion of the evolved gases, no air enters the retorts, and there is no combustion of the solid carbon; the yield, therefore, is high (from 10 to 15 per cent. higher than in the beehive oven), and the coke is hard, dense, and of uniform quality.



K, Retort chamber.
D, Iron doors.
F, Gas openings.
G, Combustion chamber.
A, Charging opening.
M, Charging floor.

Fig. 4.—The Appolt Coke Oven.

The Coppeé Oven.—This oven is very similar in principle to the Appolt, the gases being burned in a space outside the retort or coking chamber; but the retorts are of different form, and are horizontal instead of vertical. The retorts are about 29 feet long, 18 inches wide, and 4 feet high, tapering slightly from back to front; they are placed side by side in rows of twenty or thirty, a combustion space being left between each two. The retorts are closed at each end by two pairs of iron doors, a lower large pair about 3 feet high and a smaller pair closing the space above; and are provided at the top with two or three charging hoppers. The oven being red hot from the previous charge, the lower doors are closed, and the fresh charge is let in through the hoppers, and levelled as quickly as possible by rakes introduced through the upper doors, after which these also are closed. Distillation begins at once, the gases pass into the spaces between the retorts, where they meet air and are burned, and the products of combustion, after circulating under the ovens, pass away to the main flue and thence to the chimney, the waste heat often being utilised for steam raising. The ovens are worked in pairs, one being charged when its neighbour is about half done, so that a continuous supply of gas is kept up. When coking is complete the doors are opened, the coke is pushed out by a ram, and is quenched with water. The time taken is from twenty-four to forty-eight hours, and the coke yielded is hard and dense.

Recovery of Bye-Products.—When coal is distilled, in addition to the principal products, gas and coke, other substances, such as tar and ammonia, are evolved which are of value, and many attempts have been made to recover them.

By a slight modification, the beehive oven can be so arranged as to allow of the recovery of these bye-products, but the necessary changes are inimical to good coking. The door and charging openings must be made to fit air-tight, and the products of distillation, after being led through condensers for the separation of the condensible substances, are brought back and burnt under the bottom of the oven, which must be made double. In this form of oven the coking takes place entirely from below, and this tends to produce a soft coke.

The Simon-Carvés Oven.—Of the many ovens designed for the recovery of bye-products this has come more largely into use than any other. The oven is somewhat similar to the Coppeé oven, and consists of a series of chambers or retorts about 25 feet long, 6 or 7 feet high, and 2 feet wide, provided at the ends with doors which can be closed air-tight. The powdered coal is charged through hoppers in the roof; the products of distillation pass away by a pipe to the condensers, and, after the removal of the condensible products, the gases are brought back and burned under the oven, a small fire being kept burning so as to prevent the gas being accidentally extinguished. The products of combustion circulate under the floor of the oven, then pass through a series of flues in the wall between the ovens, and thence to the chimney flues. The coke is expelled from the ovens by means of a ram carried on a carriage running on rails at the back of the ovens, and is at once quenched with water. The yield in this oven is high, and the coke is of excellent quality.

Condensing Plant.—The gases from the ovens are led into an atmospheric condenser, a series of vertical pipes exposed to the air, connected at the top in pairs and standing in an iron box provided with diaphragms so arranged that the gases must pass up and down through the whole series of pipes. They then pass up scrubbers, tall towers filled with coke or flints, or crossed by perforated diaphragms, and down which a shower of water is kept constantly falling. The cooling and washing separate the tar, and the water dissolves out the ammonia. The tar and ammonia liquor are run into a tank, separated, and afterwards subjected to the necessary treatment for the recovery of their valuable constituents.

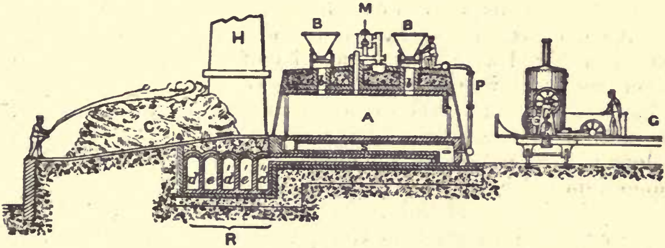


Fig. 5.—Simon-Carvés Coke Oven.

A, Oven.
B, Charging trucks.
b, Charging openings.
M, Gas main.
P, Return gas pipe.
H, Chimney.

C, Heap of coke.
G, Ram for forcing out coke.
R, Recuperator.
d, Air flues.
e, Smoke flues.

Coke.—Coke varies much in its properties. There are two varieties generally made—(1) porous or light coke, and (2) hard coke. The light coke is mostly used for smiths' forges and other similar purposes; it is black in colour, very porous, crushes easily, kindles readily, and burns away rapidly.

The hard coke is used in blast furnaces, cupolas, &c. It is hard and dense, very strong, and, therefore, offers great resistance to crushing; it rings when struck and is very difficult to ignite. The hard coke made in the beehive oven breaks into columnar fragments and often shows a tubular structure; it has a silvery-gray colour and a metallic lustre; that made in the Simon-Carvés oven is black and dull and breaks into rectangular fragments.

Coke is not pure carbon, but contains ash and small quantities of hydrogen and oxygen.

	Simon-Carvés Coke.	Beehive Coke.
Carbon, . . .	86·36	87·6
Hydrogen, . . .	·51	·25
Oxygen and Nitrogen, .	1·77	1·20
Sulphur, . . .	1·07	1·05
Ash, ...	7·94	8·52
Water, ...	2·35	1·38

Coke absorbs moisture very readily on exposure to moist air.

Gases.—The gases given off when coking is conducted in the beehive oven consist of carbon monoxide, carbon dioxide, and hydrogen mixed with a large quantity of nitrogen; the gas from the closed ovens very closely resembles coal gas.

Tar and Ammonia.—The amount of tar obtained varies with the nature of the coal, and usually amounts to from 4 to 6 gallons per ton of coal; while ammonia, equal to about 15 lbs. of ammonium sulphate per ton, may also be obtained.

Desulphurisation of Coke.—When coal is distilled, part of the sulphur present passes into the gas, as hydrogen sulphide and carbon disulphide, but a considerable quantity remains in the coke as iron sulphide. When coke is treated with steam the sulphide is partially decomposed, and the sulphur is evolved as hydrogen sulphide: $Fe_2S_3 + 3H_2O = Fe_2O_3 + 3H_2S$. It has been proposed to remove the sulphur from coke by treating it with superheated steam, but only with partial success, the difficulty probably being that the steam cannot readily permeate the coke, and that if the temperature be high, the coke itself may be attacked by the steam.

Coke from non-Caking Coal.—Coke may be prepared from non-caking coal by mixing it with tar, pitch, or other similar material, moulding into blocks and then coking.

BRIQUETTES.

Blocks of artificial fuel are now frequently made by mixing powdered coal with tar, pitch, starch, or other adhesive material, and moulding into blocks under pressure. Such fuels have the great advantages that the coal can be purified from pyrites and shaly impurities by washing; and, also, that they can be made in blocks of uniform and convenient size.

LIQUID FUEL.

Oils occur abundantly in nature and are also obtained by the distillation of shale, tar, and other materials. There are two general methods of burning liquid fuel—the one is to vaporise it in coils of hot pipe or otherwise, and burn the gas; the other is to project the oil into the furnace in the form of a very fine spray mixed with air, in which form it burns as readily as gas. Liquid fuels are at present of no metallurgical importance. (?)

GASEOUS FUEL.

Gas is now largely used as a fuel in metallurgical works and has some very marked advantages. It can be readily carried by pipes to any part of the works required, the producers being placed in such a position that the coal can be easily delivered; and an inferior fuel can often be used. It is much more easy to regulate a gas fire, and to maintain a uniform temperature than it is with solid fuel, while gas allows of the use of the regenerative system, and, therefore, of the production of higher temperatures than are obtainable with any other form of fuel, and complete combustion is readily obtained so that there is little or no smoke. Natural gas can only be used in the few districts where it occurs and hence is of little importance here. Coal gas, such as is used for lighting, made by distilling coal in closed retorts, is an excellent fuel; but is too expensive for all ordinary metallurgical operations.

PRODUCER GAS.

If air be passed over hot coke, kept in excess, the carbon is burned to carbon monoxide, $C + O = CO$, which is combustible. Air contains 21 per cent. by volume of oxygen, and as this yields twice its own volume of carbon monoxide, the gas thus produced would contain 34·7 per cent. by volume of carbon monoxide. This gas would have a low calorific power, though it could be well enough burnt in a regenerative furnace; but it may be enriched in several ways. (1) Coal may be used in place of coke; the coal will be distilled by the heat of the ascending gases, and coal gas will be mixed with the producer gas. (2) Steam or water spray may be blown into the producer; this will be decomposed, forming carbon monoxide and hydrogen, $C + H_2O = CO + 2H$. This action will increase the value of the gas by increasing the percentage of carbon monoxide and adding hydrogen. The quantity of steam which can be used is limited, because its decomposition absorbs a very large quantity of heat, and, if too much be used, the temperature will be reduced below the point at which the reaction takes place, and at which carbon will decompose carbon dioxide.

A good producer gas should contain as much carbon monoxide, hydrogen, and coal gas as possible; it should be nearly free from carbon dioxide, and steam should not be supplied in such quantity that any of it passes through the producer undecomposed.

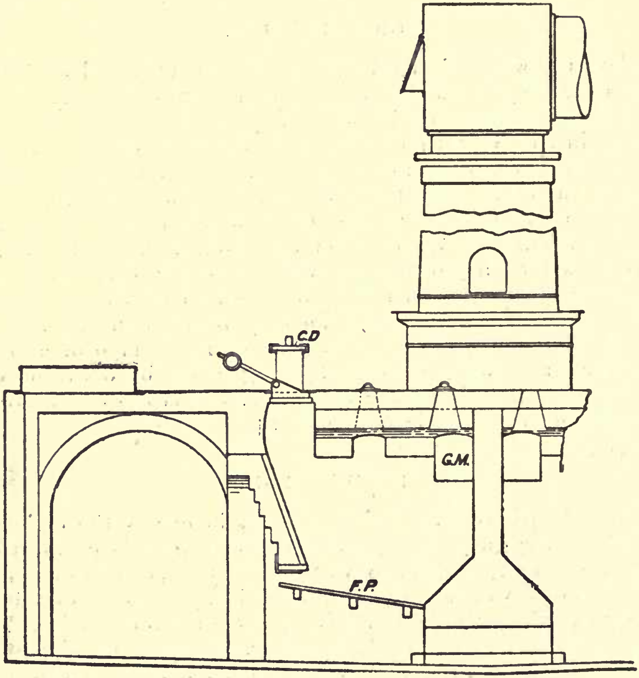


Fig. 6.—Siemens' Gas Producer.

C D, Charging door. F P, Fire-bars. G M, Gas main.

Siemens' Producer.—This is the form of producer first designed by Siemens, and is still in use. It consists of a rectangular chamber of fire brick provided with fire bars, through which air can enter, and by which ashes can be removed, and with a charging hopper for the supply of fuel. The producer is kept filled nearly to the top with coal, and water is kept in the ash-pit so as to supply a certain amount of water vapour, which is sometimes supplemented by a water spray. The air and steam entering produce the changes above described and the gas escapes at a temperature of from 300° C. to 500° C.; it is usually carried through a horizontal cooling tube, and then down a vertical tube to the furnace where it is to be burned, but often these tubes are dispensed with and the gas passed at once to the furnace. In this producer a considerable amount of tar escapes and condenses in the cooling tubes.

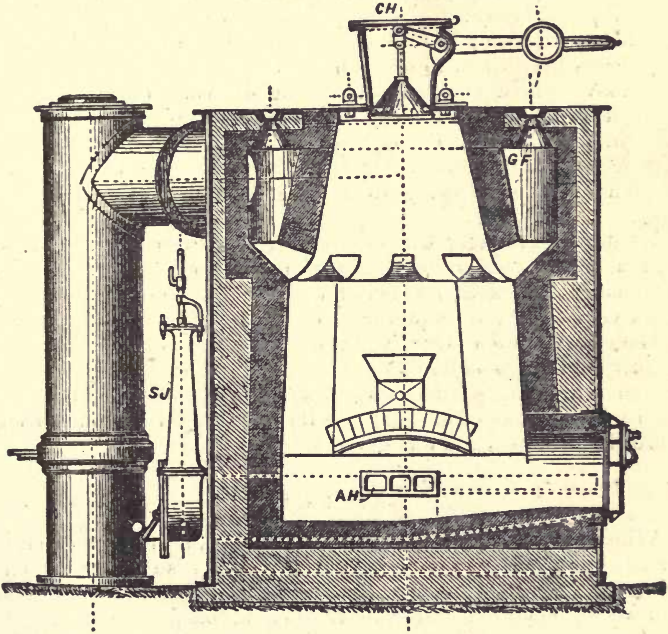


Fig. 7.—Wilson s Gas Producer.

C H, Charging hopper.
A H, Air-holes.
S J, Steam jet.
G F, Gas flue.

The Wilson Producer.—This form of producer consists of a circular shell of fire brick, cased with sheet iron; a charging hopper is provided in the centre of the top, and from this descends a conical tube of fire brick, carried from the sides on arches so as to form an annular gas space between it and the producer shell; across the bottom of the producer passes a hollow ridge of fire brick-work, the interior of which communicates with the interior of the producer by a series of openings. The air and steam are blown into this tube by means of a steam jet, and thence pass to the fuel. The producer is kept quite full of coal, and the gases produced by distillation can only escape by passing downwards through the hot coke to the arched openings leading to the gas space; in the passage the tarry matters are decomposed into permanent gases, which escape, and carbon, which remains with the coke. The producer being entirely closed, ashes collect at the bottom and are removed periodically; for this purpose two cleaning doors are provided, and also two side doors arranged so that bars can be put across from them to the central brick ridge to support the fuel above whilst the ashes are being drawn. The producer is cleaned about once in twenty-four hours, the blast being turned off while it is being done. The gas obtained in this form of producer is somewhat richer than that made in the ordinary Siemens' producer, owing to a larger quantity of steam being used.

Nature of Producer Gas.—Producer gas, even when enriched as much as practicable, contains only about 35 per cent. of combustible gases, and much of this is carbon monoxide, which has a very low calorific power; so that the actual heating power of the gas is always low. The production of gas necessitates the absorption of a considerable quantity of heat, so that the heat of combustion of the gas is always less than that of the solid fuel from which it was produced; but its advantages are such as to more than compensate for this loss.

WATER GAS.

When steam is blown over red-hot coke, a mixture of equal quantities of carbon monoxide and hydrogen is obtained. This mixture, which is called "water gas," has a high calorific power, and would be an excellent fuel if it could be produced cheaply. The reaction absorbs, as already pointed out, a very large amount of heat, which must be supplied in some way by combustion either inside or outside the producer.

An ordinary water gas producer is a tall fire-brick cylinder cased with iron. This is filled with hot coke. Air is blown in at the bottom, combustion takes place, the temperature rapidly rises, and producer gas is formed, which may either be used or allowed to escape into the air. When the temperature is sufficiently high the air supply is turned off and steam is blown in at the top of the producer; passing downwards it is decomposed, and water gas is produced. The production of the water gas rapidly lowers the temperature of the coke, till at last the steam is not completely decomposed, then the steam is turned off and air again passed through until the temperature is sufficiently high, when steam is again sent in. The production of water gas and the heating up of the producer is thus made to alternate. The amount of coke consumed in heating up the contents of the producer is about three times as much as is actually used in

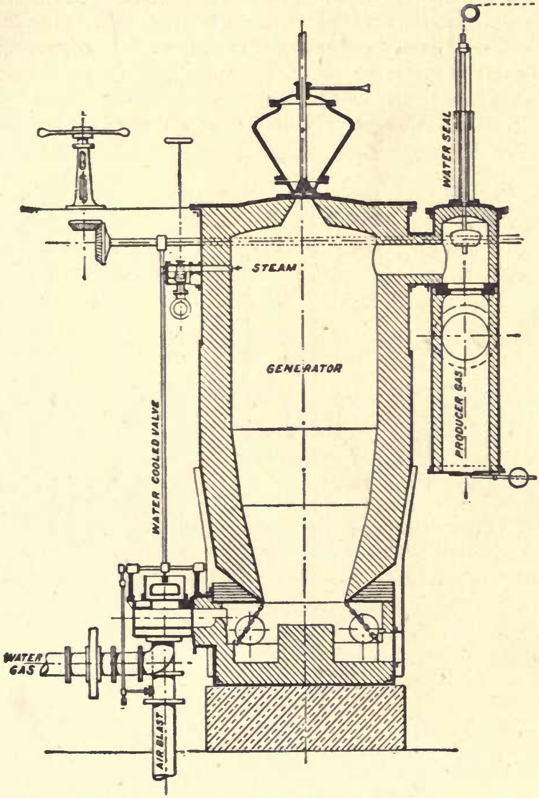


Fig. 8.—Water Gas Producer.

making the water gas; and the quantity of producer gas obtained is about four times as large as that of the water gas. It is not likely that water gas can be produced at a cheap rate, except under the very exceptional conditions where the producer gas can be economically used.

Water gas, owing to the large amount of carbon monoxide which it contains, is very poisonous, and, having no smell, its presence is not easily detected. It burns with an almost non-luminous flame.

Composition of Gaseous Fuels.

	Natural Gas.	Coal Gas.	Simple Producer Gas.	Enriched Producer Gas.	Water Gas.	Blast Furnace Gas, Coke Fed.	Blast Furnace Gas, Coal Fed.
Nitrogen,	...	...	67·4	56·11	3·7	57·06	55·35
Hydrogen,	20·02	48	...	11·55	48·6	2·74	6·73
Carbon dioxide,	·8	...	3·9	4·0	3·3	28·61	25·97
Carbon monoxide,	1·0	8	27·3	26·89	44	11·39	7·77
Marsh gas,	72·18	36	1·4	1·45	·4	·20	3·75
Other hydrocarbons,	4·3	3·8	...	...	...	...	·43
Oxygen,	1·1	...	...	...	...	...	...
Percentage of combustible,	98·1	95·8	28·7	39·89	93	31·35	36·88
Heating power per cub. ft. F. units,	728	694	93	140	293	...	...

CHAPTER IV.

REFRACTORY MATERIALS.

CLAY.

MANY very refractory substances occur in nature and have been used in furnace construction. There are many highly refractory rocks, but most of them are hard and difficult to work, or else break up on being heated. The only material now largely used is clay, which from the readiness with which it can be moulded into any form, and its hardness after heating, is well suited to furnace construction.

Clay is a hydrated silicate of alumina, containing in the purest form, kaolin or China clay, alumina 40 per cent., silica 47 per cent., water 13 per cent. ($Al_2O_32SiO_22H_2O$ or $Al_2Si_2O_72H_2O$). It is perfectly white, has a soapy feel, adheres slightly to the tongue, and is infusible at furnace temperatures. When clay is mixed with water it becomes plastic, and can be moulded into any required form. When the moulded mass is heated to redness the combined water is expelled and a hard residue or brick remains. This is porous, absorbs water readily but does not combine with it, and does not become plastic.

Clay is formed by the atmospheric disintegration of certain rocks containing felspar, the alkalies of the felspar being removed in combination with part of the silica and clay being left. If the rock be pure felspar the residue will be a pure clay, but if the rock contains any mineral which does not suffer disintegration, this will be mixed with the clay. The clays in nature are rarely found in the positions where they are formed, the light clay is readily carried away by running water, and being carried out to sea or into inland lakes ultimately settles, forming beds of clayey matter which become hardened into shales. Naturally the clays are often mixed with other matters, which may or may not affect their fire resisting properties. A clay which does not soften or fuse at very high temperatures is called a "fire clay."

Clays occur in all geological periods; those most generally used as fire clays belong to the carboniferous period, probably because the rocks of this period, being worked for coal and ironstone, occur in the neighbourhoods where the fire clay is required.

Clays vary much in appearance, those of the recent geological periods are soft and plastic; those of the carboniferous period are usually dark in colour, from the presence of bituminous matter have a talcy lustre and a soapy feel, and are apparently quite dry, but on ignition they lose water. When mixed with water they yield a plastic mass. The clays of still older formations have often lost their water of combination, and having passed into the condition of slates, no longer become plastic on the addition of water.

Impurities in Clays.—All clays are more or less impure. Most contain a considerable quantity of free silica. This is not a disadvantage, provided the quantity is not so large as to impair the plasticity, as it does not diminish the refractoriness of the clay. The alkalies, soda and potash, are always present, and if in considerable quantity, say over 1 per cent., they are very deleterious, as they form very fusible silicates. Lime and magnesia are objectionable for the same reason. Oxide of iron is also objectionable; if present in large quantity it imparts a reddish colour to the brick, owing to the formation of ferric oxide; if such a brick be exposed to a reducing atmosphere, this will be reduced to the ferrous condition and combine with the silica, forming a very fusible ferrous silicate. Pyrites (FeS_2) is often present, scattered through the clay in fine particles. When a brick or other article made from such a clay is fired the sulphur of the pyrites is burned off and oxide of iron is left, this combining with silica forms fusible ferrous silicate which at once sinks into the clay staining it black and forming a black ring round a small hole which was the space occupied by the speck of pyrites.

ANALYSES OF REFRACTORY MATERIALS.

	Dowlais Clay.	Stourbridge Clay.	Fire Clay Etherley.	Garnkirk Clay.	Glenboig Clay.	Ganister.	Dinas Rock.	Burnt Magnesian Limestone.
Silica and Titanic Acid,	67·12	63·3	55·61	44·37	57·57	89·37	98·31	8
Alumina, . . .	21·18	21·18	27 50	38·59	26·35	6·36	·72	4
Oxide of Iron, . .	1·85	1·80	1·91	1·82	1·33	1·72	·18	
Lime, ...	·32	·73	·32	·51	·60	·70	·22	52
Magnesia, . . .	...	...	·79	·31	·55	·36	...	36
Alkalies, . . .	...	...	·81	...	·48	...	·18	...
Water and Organic Matter, . . .	7·41	10·30	12·42	14·40	13·75	3·78	·35	...

Manufacture of Fire Bricks.—The clay is, if necessary, weathered by exposure to the air, and is then ground in a mill; the powder is passed through a riddle having about six holes to the inch, and the particles which do not pass through are returned to the mill. Water is added so as to make the mass plastic, and it is very thoroughly mixed in a mill. The plastic mass is cut up into pieces of convenient size, which are pressed into wooden moulds, the tops are smoothed off with a piece of wood, the bricks are turned out on to boards, dried at 60° to 70° so as to harden them somewhat, and are then fired in a heap or in a kiln, the temperature being raised very gradually for some days, till almost a white heat is attained. They are then allowed to cool slowly, and are ready for use.

GANISTER.

This is a close-grained dark-coloured rock, occurring in the carboniferous formations near Sheffield, though the name is now given to similar rocks occurring in other localities. It is an argillaceous sandstone, the clay being present just in sufficient quantity to allow of it being moulded. It may be taken as containing about—

Silica (in excess), ...	82
Clay (dry), ...	14
Impurities, ...	2
Moisture and organic matter, . . .	2

It is very refractory, and is used for making fire bricks, for furnace linings, and for linings for Bessemer converters.

SILICEOUS MATERIALS.

More siliceous materials are sometimes used for furnace linings and for the manufacture of silica bricks. Among these may be mentioned sand and calcined flints (which are nearly pure silica), and Dinas rock (p. 44) (often improperly called Dinas clay), a rock which occurs in the Vale of Neath, and which is used in making the famous Dinas bricks. These materials are not clays, and do not become plastic when mixed with water; so that it is necessary to mix them with some material to bind the particles together. The material usually used is lime, about 1 per cent. being required. Such bricks require very careful handling, as they are very tender until after firing, and they must be fired at a high temperature. The lime forms a small quantity of silicate of lime, which acts as a frit round the particles and binds them together.

PROPERTIES OF A GOOD FIRE BRICK.

A good fire brick must not fuse, soften, or become weak at high temperatures; it must not crack if exposed to sudden alternations of temperature; and it should resist the corrosive action of metallic oxides or other material which may come in contact with it. No brick has all these properties in the highest degree, and in the selection of a brick for any special purpose attention must be paid to the conditions to which it is to be exposed. Silica bricks are very infusible, but, as a rule, they are fragile, and will not stand sudden alternations of temperature; they are very rapidly attacked by fused metallic oxides or basic slags. Obviously, fire bricks containing silica must not be set in lime mortar, as at high temperatures the lime would flux the silica; but they must be set in good fire-clay.

BASIC MATERIALS.

All the refractory materials described above are "acid" in character—that is, they contain a large quantity of silica, which will combine readily with basic oxides, such as lime, magnesia, &c. For some furnace purposes basic materials are required, and for these the substances above described will not do. There are several very refractory basic materials known, but they will not bind, and, therefore, cannot be made into bricks without the use of some cementing material. Lime and magnesia have been used, being mixed with a little fire-clay or a soluble silicate as a binding agent. The material most generally used is burnt magnesian limestone. This rock is an impure dolomite, and consists of carbonate of lime, carbonate of magnesia, and a small quantity of silica; on burning, the carbonates are converted into lime and magnesia, and if this material be moulded into bricks and fired at a high temperature the silica combines with the lime and magnesia, forming a "frit" which binds the whole into a coherent mass. In order to bind the material and hold it together till the temperature is high enough to allow of the combination of the silica and lime, &c., the bricks are usually made by mixing the burnt magnesian limestone with hot anhydrous tar in a mill, and pressing the mixture into moulds by hydraulic pressure. Such bricks are very refractory, but are weak and friable.

NEUTRAL MATERIALS.

For some purposes a material which will neither combine with basic oxides nor with silica is required; such are called neutral materials. Those most used are graphite (C) and chrome iron ore (chromite, Cr_2O_3FeO).

OTHER REFRACTORY MATERIALS.

Some other materials are used for special purposes. Bull-dog, which is used for lining puddling furnaces, is a mixture of ferric oxide and silica, made by roasting tap cinder ($2FeO . SiO_2$). This is refractory in an oxidising atmosphere; but in a reducing atmosphere it is fusible, being reconverted into ferrous silicate. Ores of iron are sometimes used for furnace lining.

CRUCIBLES.

These are open-topped vessels in which substances are heated so as to be out of contact with the fuel or the products of combustion. Owing to its plasticity clay lends itself readily to the production of vessels of any form, and it, therefore, is the material always used. A good crucible must resist a high temperature without softening or becoming weak; it must not crack when exposed to sudden changes of temperature, as, for instance, when lifted out of a hot fire into the air; it must resist the corrosive action of fused substances; and it must not be porous. Most of these properties are of far more importance in the case of crucibles than in the case of fire bricks.

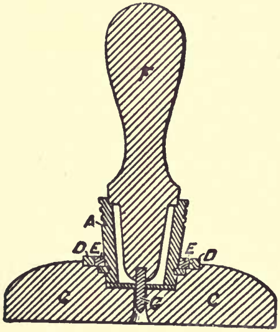
Manufacture of Crucibles.—The basis of the crucible is always plastic clay, but as clay contracts very much on heating it is necessary to mix it with some refractory substance which does not contract, and which will form a rigid skeleton to keep the plastic clay in form. The substances used are burnt clay, silica, and graphite.

As an example of the method of making crucibles one of the methods of making Stourbridge clay crucibles is described, but it must be remembered that it is given only as an example, and that other methods are also used.

The clay is ground under edge-runners in the usual way, sieved, mixed with powdered burnt clay, and the whole rendered homogeneous by mixing and treading. The crucible maker sits before a rotating table on which is a core having the form which the interior of the crucible is to have; on the top of this he places the lump of clay, and, as the block is rotated, he presses the clay down till the core is covered, and the exterior is roughly formed into the required shape; the exact form of the exterior and the thickness of the clay is then fixed by a gauge or template. The block is then removed from the table and inverted, the crucible slips off, the spout is made with the fingers, and the crucible is dried and fired.

The large crucibles used in steel foundries are made somewhat differently. The clay is ground and mixed with burnt clay as before. An iron vessel or flask, the interior of which has the shape which the exterior of the crucibles is to have, is placed on a circular iron bottom, in the centre of which is a hole. The requisite amount of clay is placed in this flask, and a core, the shape of the interior of the crucible, is forced down into it, a central spike in the bottom of the core fitting into the hole in the bottom of the flask, and thus keeping the core central. The core is pressed into position, the clay rises between it and the flask, and the crucible is made. The core is removed, leaving the crucible with a hole in the bottom; this may be stopped with a lump of clay, but in steel melting crucibles it is left, the crucible for use being placed on a brick, which is sprinkled with sand; as soon as the temperature rises the crucible becomes fused to the brick and thus the hole is stopped.

An exactly similar method is used in the laboratory for making small crucibles for the Swedish iron assay; but in this case the hole in the bottom of the crucible is stopped with a small plug of clay.

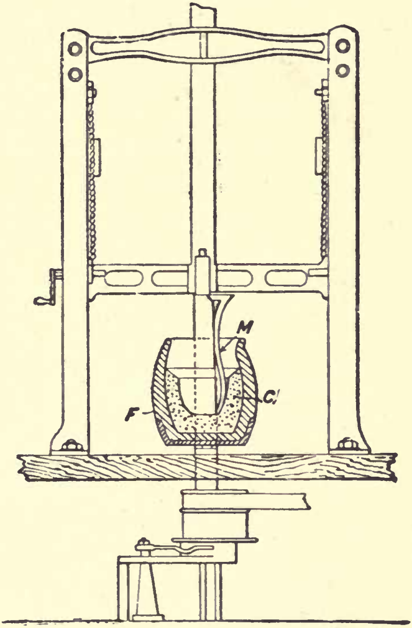


C, Base.
D, Brass ring.
A, Brass mould or flask.
F, Core.
G, Projecting point.

Fig. 9.—Laboratory Crucible Apparatus.

Forms of Crucibles.—Crucibles are made in various forms. At one time a particular form was always made of a particular clay mixture, but that is no longer the case; crucibles of any form can be had of exactly the same quality. Among the most famous crucibles for assaying purposes may be mentioned Juleff's Cornish crucibles, which are now made by the Morgan Crucible Company, London.

Plumbago Crucibles.—These are now very largely used; they are very refractory, will support sudden changes of temperature without cracking, do not become weak at high temperatures, are very smooth so that shots of metal do not adhere to them, and they can be used over and over again, whilst, as a rule, clay crucibles can only be used once. Plumbago crucibles have almost superseded clay crucibles for many purposes. They are made of good clay mixed with from 20 to 40 per cent. of its weight of graphite. The largest makers of these crucibles are the Morgan Crucible Company, Battersea. This firm makes its crucibles entirely by machinery, and thus secures very great uniformity. The apparatus consists essentially of a rotating mould in which a weighed portion of the mixture of clay and graphite is placed, and a template to give the internal form centred accurately over the mould, and arranged so that it can be gradually lowered as required on to the rotating mass in the mould, till the crucible is shaped. The crucibles are then dried and fired in a "saggar" in a kiln.



F, Flask.
M, Mandril.
C, Clay being formed into crucible.

Fig. 10.—Crucible Moulding Machine.

One objection to plumbago crucibles is that they must be carefully annealed before they are used for the first time or they are apt to fall to pieces in the fire. The Morgan Company now makes "Salamander" crucibles, which are plumbago crucibles covered with a glaze which is impervious to moisture. These can be placed at once in a hot fire without fear of breakage.

Lining Crucibles.—It is sometimes necessary to line crucibles with carbon. This is readily done by mixing powdered charcoal with treacle and water till the mass will just adhere on pressing between the fingers, pressing the mass tightly into the crucible, cutting or boring a hole in the centre, drying and then firing without access of air.

CHAPTER V.

FURNACES.

Classification of Furnaces.—Furnaces may be classified in several ways. The following arrangement, though not by any means perfect or complete, will be found to be convenient and to include most types of furnace in common use:—

1. Furnaces in which the substance being heated is in contact with the fuel.
    - (*a*) No artificial blast is used. Kilns.
    - (*b*) Artificial blast is used—
        - *α*. The height of the furnace is greater than its diameter. Shaft or blast furnaces.
        - *β*. The height of the furnace is less than its diameter. Hearths.
2. Furnaces in which the substance being heated is in contact with the products of combustion but not with the fuel. Reverberatory furnaces.
    - (*a*) Arranged for the use of solid fuel.
    - (*b*) Arranged for the use of gaseous fuel.
3. Furnaces in which the substance being heated is not in contact either with the fuel or with the products of combustion—*i.e.*, is contained in a separate vessel or chamber.
    - (*a*) The substance is not volatilised.
        - *α*. The vessel is fixed. Muffle furnaces.
        - *β*. The vessel is movable. Crucible furnaces.
    - (*b*) The substance is volatilised. Retort furnaces.

KILNS.

These are principally used for the calcination of ores, burning limestone, and other operations where a very high temperature and a large quantity of air are not required. As examples, two forms of kiln used for calcining iron ores may be taken.

The Dowlais kiln is rectangular in plan and widens towards the top; the length is somewhat greater than the height. It is lined with fire-brick and floored with iron plates, set to slope a little towards the front, in which there is a series of arched openings for withdrawing the charge, and a set of holes for the admission of air. The kiln being at work is kept filled with a mixture of ore and fuel, which is added at the top as the calcined charge is withdrawn at the bottom, so that the working is continuous. The quantity of fuel consumed is very small, in calcining ironstone, only about 1 cwt. per ton of ore.

The Gjers kiln, used largely in the Middlesbrough district, is probably the most perfect yet introduced. It consists of a fire-brick shell cased in iron plates, the upper part is cylindrical, the lower an inverted frustrum of a cone, resting on an iron ring carried on hollow iron columns. In the centre is an iron cone arranged so as to throw the descending charge outwards and allow of its easy withdrawal; this cone is connected with underground air-passages, so that a current of air passes into it, keeping it cool and also supplying air for the calcination. The quantity of fuel used in this kiln is probably smaller than in any other form.

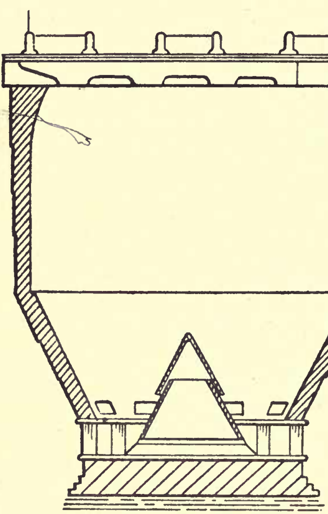


Fig. 11.—Gjers' Kiln.

Gas kilns have been suggested, but have not come largely into use. They are built much in the form of the Gjers kiln, except that the spaces between the supporting columns are closed by iron doors, which are only opened when the charge is to be drawn. The air and gas are supplied from mains passing round the kilns by openings just above the doors. The top of the kiln is usually closed. The only advantage of a gas kiln is that the ore does not become contaminated with the ash of the fuel.

SHAFT FURNACES.

These differ from kilns in two important particulars; (1) air is supplied by a blast under pressure, or, in rare instances, by suction produced by an exhaust, or by a tall chimney; (2) the charge is not drawn from the furnace in a solid condition, but is fused and tapped out.

As a type an iron smelting blast furnace may be taken. The sketch and index will sufficiently indicate the form and names of the parts.

Chemistry of the Blast Furnace.—In a furnace of this type air is supplied near the bottom and the gases escape at the top, after passing up the furnace through the mass of coke and ore. As the air comes in contact with the coke at a high temperature most of the oxygen forms carbon monoxide, $C + O = CO$;

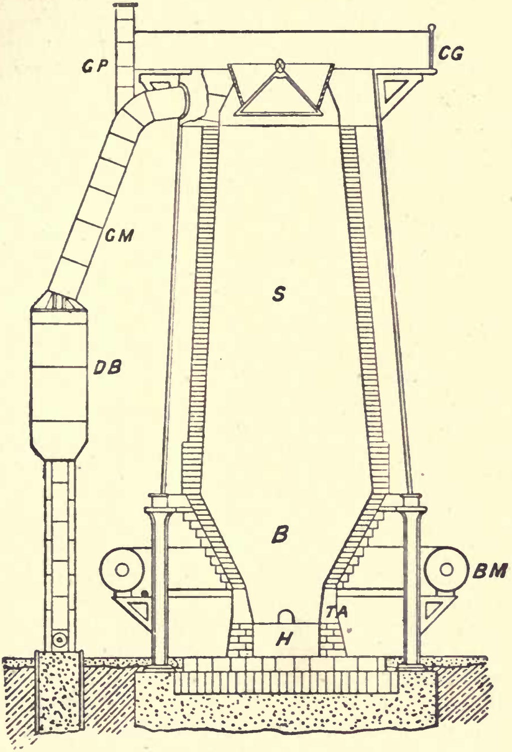


Fig. 12.—Blast Furnace.

S,	Shaft.	C G,	Charging gallery.	B M,	Blast main.
B,	Boshes.	D B,	Dust-box.	T A,	Tuyere arches.
H,	Hearth.	G M,	Gas main.	G P,	Stand pipe.

though some carbon dioxide may be formed at the same time, $C + 2O = CO_2$, but this will probably be speedily reduced as it passes upwards by the excess of carbon, $CO_2 + C = 2CO$. The air contains about 21 per cent. of oxygen and 79 per cent. of nitrogen by volume, and as carbon monoxide occupies twice the bulk of the oxygen from which it was formed, the gas as it passes upward will contain about 34·7 per cent. carbon monoxide and 65·3 per cent. nitrogen, and will be quite similar to producer gas; it will also be mixed with traces of hydrogen from moisture carried in in the air. The combustion of the fuel will produce a very high temperature where the air enters, and as the hot gases ascend they will gradually give up their heat to the descending charge; and should escape at a temperature not above 316° C. (600° F.); which, therefore, will be about the temperature at the top of the charge. The atmosphere of the furnace will be powerfully reducing from the presence of the carbon monoxide, and any reducible metallic oxides will be reduced near the top of the furnace. The escaping gases will be combustible and may be used for firing boilers or other purposes. When raw coal is used as fuel, as is the case in most Scotch iron furnaces, this will be coked very near the top, and the furnace gases will be mixed with coal gas and other products of the distillation of the coal. At one time all the gas was allowed to burn at the throat of the furnace, the furnaces then being "open-topped," now it is almost always drawn off and used for heating: the furnaces having "closed tops."

Charging.—The charge is introduced through the throat of the furnace. In the open topped furnaces it was put in through openings in the tunnel head, in closed top furnaces some device must be used, so that while the top is usually closed it can be quickly opened to let in the charge, and quickly closed again. Various arrangements are in use, the most general being the bell and cone. When the top of the furnace is closed the gases are drawn off by a pipe passing through the furnace wall, and communicating with a gas main running round the exterior of the furnace. The throat of the furnace is fitted with an iron inverted cone, made of several castings bolted together descending about 6 feet into the furnace, and firmly bolted at its upper edge to the furnace top, this is the "cone." Inside this is suspended another cone of cast iron (the "bell"), of such size that when it is drawn up it completely closes the opening of the cone. This bell is suspended by a chain from a lever, at the other end of which is a balance weight, and to which is attached some form of mechanism by which it can be raised or lowered. When the bell is up the top of the furnace is quite closed, and the charge (ore, fuel, and flux) is thrown in and rests in the space between the bell and cone. When the charge is to be let down the gas is cut off from the gas main and turned into a stand pipe, by which it escapes into the air; if this were not done air might find its way into the gas mains and cause an explosion.

The bell is then lowered, the charge slips into the furnace and a mass of flame rushes out. The bell is instantly raised and the gas is turned at once into the gas mains, the whole operation occupying only a very short time. A brazier of burning coke is usually hung over the cone so that the escaping gas may be at once ignited. If the bell does not fit sufficiently tightly against the cone to prevent the escape of gas, the junction is damped by throwing in a small quantity of powdered purple ore or other material.

In some furnaces, usually those of small size, such as the Freiberg Pilz furnaces, an arrangement called the "pipe" charging apparatus is used, which consists of a large pipe passing down into the furnace to some distance below the gas tubes. This is kept filled up with ore and fuel, the mass of which offers sufficient resistance to the passage of the gas to prevent any escaping. When this arrangement is used on wide furnaces the pipe is attached to the bottom of an inverted cone and then forms the "cup and pipe." Many other forms of charging arrangement are used under special circumstances.

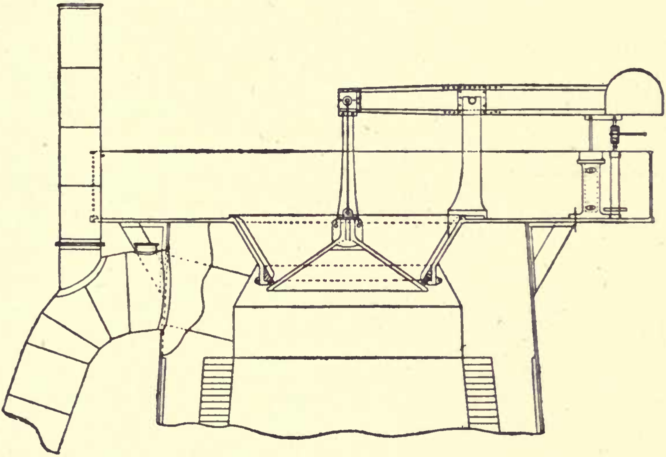
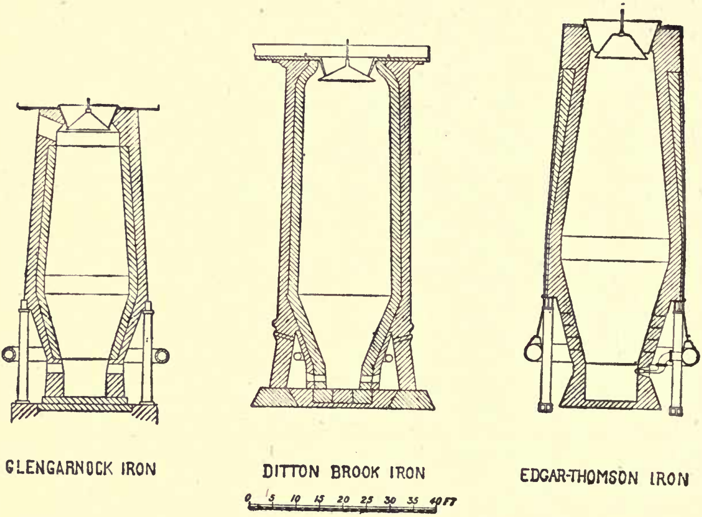


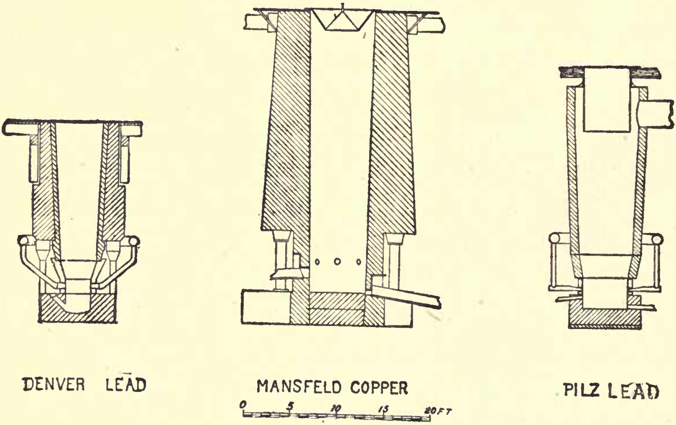
Fig. 13.—Bell and Cone Charging Apparatus.

Form and Materials.—The form of the blast furnace varies very much, according to its size and the purpose for which it is to be used. The older type of blast furnace for iron smelting consisted of a heavy mass of masonry, about 40 to 45 feet in


GLENGARNOCK IRON

DITTON BROOK IRON

EDGAR-THOMSON IRON


DENVER LEAD

MANSFELD COPPER

PILZ LEAD

Fig. 14.—Typical Blast Furnaces.

height, the upper part being carried on massive stone arches. The interior consisted of three portions, the shaft, the bosh, and the hearth, with well marked lines of demarcation between them. Modern furnaces are much taller, being from 50 to 100 feet in height; they are cylindrical, and are strengthened by bands or by an entire casing of iron. Owing to their decreased external diameter the charging gallery has to be carried on brackets and projects considerably. The upper part of the furnace is carried on a series of iron columns, so that access can be had to all parts of the hearth and boshes. The internal form has also been changed; all sharp angles have been removed, the interior contour being a continuous curve; and though the names of the various parts are retained, the sharp lines of demarcation between them have disappeared. The shell of a modern furnace is built of ordinary brick, inside this is the lining of refractory fire brick; the hearth and boshes being subject to the most intense heat are built entirely of very refractory fire bricks poor in silica. The upper part of the furnace is built quite independently of the boshes and hearth, so that, should these become worn out, they can be replaced without interfering with the shaft. Smaller blast furnaces often taper gradually from the top to the bottom. Blast furnaces are usually circular in cross section, but Rachette furnaces which are much longer in one direction than in the other are used to a considerable extent for lead smelting.

The Hearth.—The lower part of the furnace where the fused material collects is called the hearth; it may be entirely closed except for the tap holes, or it may be partially open. In the older iron furnaces, and in most lead and copper furnaces the open hearth is used. The front of the hearth is carried a little forward beyond the furnace, which is supported by an arch called the tymp arch. The front of the hearth is closed by a wall of fire brick, the "dam" which is strengthened by an iron plate, the "dam plate;" at the top of the dam is a groove, the "cinder notch," through which the slag flows, and runs down a slope, the "cinder fall," to the moulds placed to receive it, and through the bottom of the dam is made the tap hole, by which the metal is tapped out. When the furnace is at work the space between the top of the dam and the tymp arch is closed with clay. The reason for the use of the open hearth is that in some blast furnace operations where the temperature is not very high masses of unfused slag may accumulate, and the tymp opening allows of these being drawn out. In modern iron furnaces and many others, the hearth is circular and has no opening except two tap holes at different levels, one for the metal, the other for the slag.

The Bosh, Water Blocks, and Water Jacket.—The air from the tuyeres passes in at the bottom of the bosh or at the top of the hearth, just above the level of the liquid material when the hearth is full; this, therefore, will be the zone of highest temperature, and, sometimes, the temperature is so high that the refractory lining is rapidly cut away. To keep the bosh cool water blocks are often built into the masonry. These are hollow blocks of iron through which water is kept circulating; without sensibly reducing the temperature of the furnace, they keep the brickwork cool, and so protect it from injury.

In furnaces used for smelting lead or copper ores, the oxides of which rapidly attack fire bricks, and thus destroy the furnace, more complete cooling of the boshes is necessary. A water jacket is then used. This consists of a hollow casing of wrought or cast iron, through which water is kept circulating, and which takes the place of the lower part or the whole of the masonry of the bosh. The flowing water keeps this so cool that the surface becomes covered with a layer of solidified slag, which is constantly being melted away and as constantly replaced, and which, therefore, protects the surface of the metal from injury. The water jacket is the most important recent improvement in the blast furnace.

Tuyeres.—The air supplied by the blowing engine passes into an air main which usually runs round the furnace about the level of the top of the bosh; from this a pipe descends to the tuyere by which the air passes into the furnace. The tuyere itself is simply a metal tube, and when the blast is used cold it is often connected to the pipe from the main by a leather hose, forming what is called a "goose-neck." When the blast is used hot this is not practicable; an iron pipe is then attached to the tuyere, which is connected to the vertical pipe or down-comer from the main by a ball and socket or other joint. At the angle there is usually a hole covered with a plate of mica or blue glass, through which the workman can observe the temperature of the interior of the furnace. When the blast is used hot it is necessary to cool the tuyere nozzle, or the hot air passing through it and the heat of the furnace would soon destroy it. The nozzle is, therefore, surrounded by a water tuyere, through which cold water is kept circulating, and which thus prevents the tuyere itself from being injured, though it does not sensibly cool the air passing through.

The Staffordshire tuyere consists of a hollow, slightly conical box with a passage through the centre, in which the tuyere nozzle rests, the box being kept full of cold water.

The Scotch tuyere consists of a coil of wrought-iron pipe embedded in a mass of cast iron, with a passage through the centre for the tuyere; water circulates through the coil, and thus keeps the whole cool.

The spray tuyere consists of a double box like the Staffordshire tuyere, but the water is supplied in a fine spray into the upper part, and the water is not allowed to collect to above the level of the bottom of the front.

Pig and Slag Moulds.—The metal tapped from the furnace is received into moulds; these moulds are usually simply depressions made in a bed of sand by means of a wooden pattern. The bed of sand is arranged in front of the furnace, and is called the "pig bed." In the case of iron-smelting furnaces the metal flows along a runner, at right angles to which are other passages to which the separate pig-moulds lead. The metal which solidifies in the pig-moulds forms the "pig," that which solidifies in the channel uniting the pigs is called the "sow."

The slag is in some cases run into sand moulds, but more often into an iron vessel resting on a truck, which can be drawn away, and the slag taken to the slag heap.

Hoists.—As a blast furnace is of considerable height a hoist will be required to raise the materials to be charged to the top. The hoist most used is the direct-acting steam hoist. This consists simply of a cage running on vertical guides, lifted by a rope which passes over a pulley at the top, and thence round a drum, which is driven by a steam engine.

Many forms of hoist, inclined plane, hydraulic, and pneumatic, have been suggested; but only one need be described here, the Gjers pneumatic hoist, which is probably the best of its kind.

This consists of a vertical iron tube about 3 feet in diameter, and a little taller than the furnace; at the top of this are fixed four large pulley wheels, over which pass wire ropes. One end of each rope is attached to a corner of a square platform which surrounds the vertical pipe, the other end is attached to a piston fitting air-tight within the tube. The piston is weighted so as to balance the weight of the platform. The bottom of the tube is connected with an air-pump, which can be worked either for exhaust or compression. Suppose the table to be at the furnace top, the piston will be at the bottom; the barrows, or whatever is to come down, are run on to the platform, and air is forced under the piston at a pressure of about 2 lbs.; the piston slowly rises, and the platform comes down. When the platform is at the bottom the empty barrows are run off, and the load which is to go up is put on; the air-pump is worked to exhaust; the atmospheric pressure forces down the piston, and the platform is raised. It will be seen that with this apparatus overwinding is impossible, and should a rope break, one corner of the platform only would drop, and it would wedge against the central support, so that it could not run down.

SHALLOW HEARTHS.

The hearth differs from the blast furnace in form, but resembles it in needing an artificial blast and the material being mixed with the fuel. It differs from it in that it is usually of small size and low, the blast being supplied at or near the top. As an example, the Catalan forge, used for the direct production of malleable iron in the Pyrenees, may be taken.

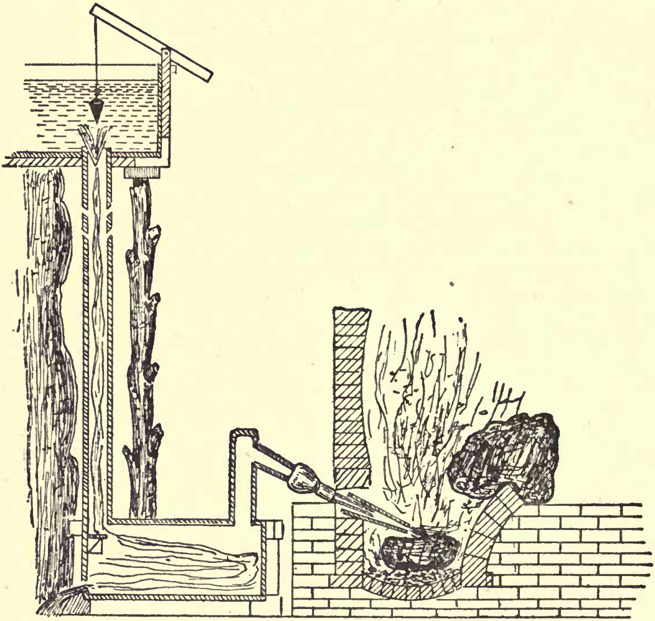


Fig. 15.—Catalan Forge and Trompe.

The furnace consists of a small cavity lined with refractory stone, and supplied with air from a single tuyere. The mixture of ore and fuel, in this case charcoal, is introduced, and the blast is turned on; combustion takes place; the ore is reduced by contact with the carbon and the metal and slag sink to the bottom, the latter being tapped out, and the former being left as a spongy mass. In such a furnace the temperature will always be comparatively low; and, whilst the atmosphere at the bottom will be reducing, at the top it may be oxidising; so that, in some cases, a hearth may be used where oxidation alone or in conjunction with reduction is required.

Hearths are not much used in metallurgy. The most common example is the smith's forge, in which the blast is supplied from below.

AIR SUPPLY.

Most forms of blast furnace or hearth require air to be supplied under pressure. The simplest form of blowing apparatus, a pair of bellows, gives only an intermittent blast; a double bellows, such as is used by the smith, gives a much steadier, though not by any means a perfectly steady, blast. The quantity of air which can be supplied by apparatus of this

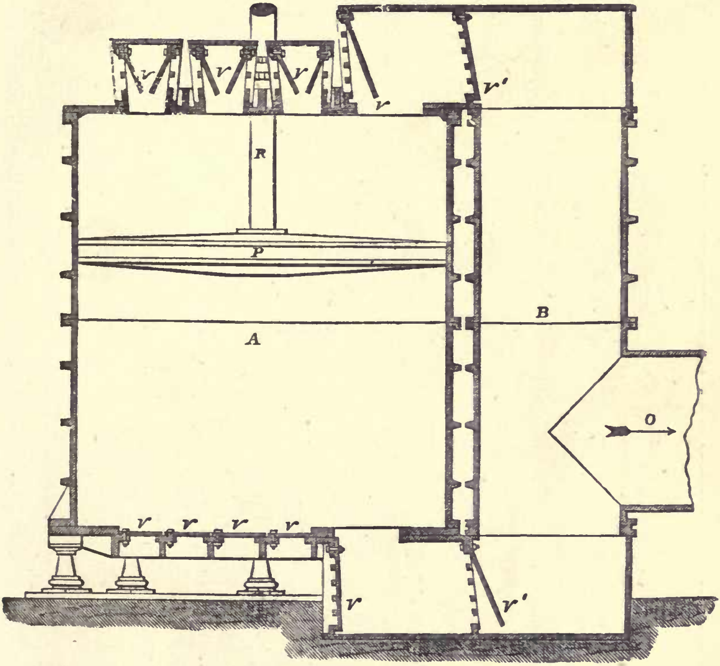


Fig. 16.—Blowing Engine Cylinder.

V, Intake valves.
V^1, Outlet valves.
P, Piston.
R, Piston-rod.
A, Cylinder.
B, Blast main.

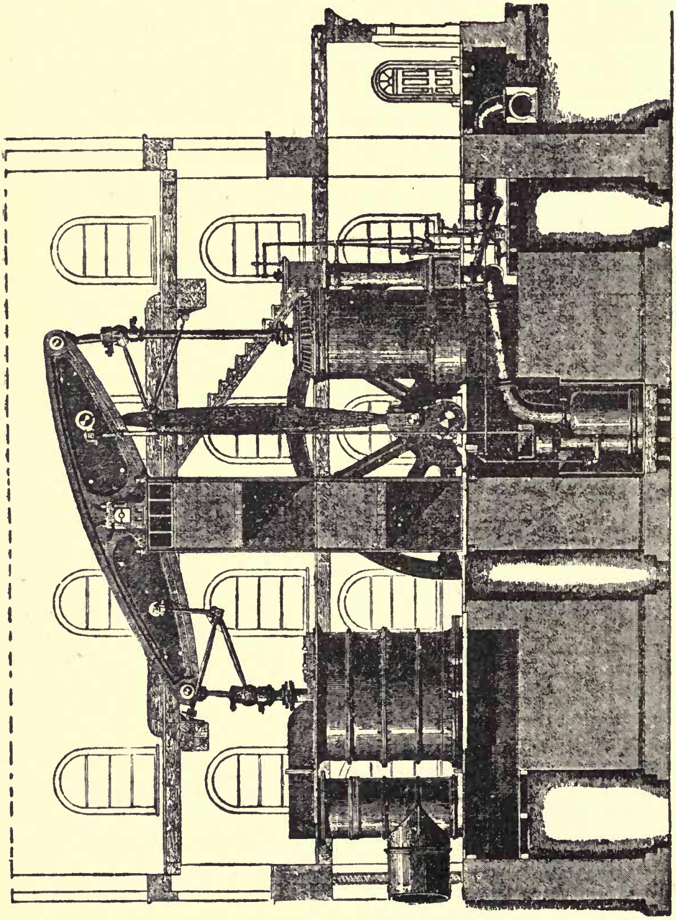


Fig. 17.—Blowing Engine, Ebbw Vale Iron Works.

kind is insufficient, except for very small hearths. For larger furnaces blowing engines or blowers must be used.

Blowing Engine.—The essential part of a blowing engine is the air cylinder. This is often of large size, up to 12-feet stroke and 12 feet in diameter. It is provided with a piston working air-tight; and if it is—as is almost always the case—double-acting, it must have two sets of valves, intake and outlet, at each side of the piston. To prevent undue resistance to the inrush of air the area of the intake valves must be large, about half the area of the piston; one large valve would be impracticable, so a series of small valves are used, those at the top being fitted in valve boxes on the cover of the cylinder, those at the bottom being in the bottom of the cylinder. The valves are usually rectangular plates of metal hung at the one side and resting on india-rubber or leather pads. The outlet valves are similar, but are much smaller in total area.

The blowing cylinder is almost always worked by steam. The steam and blowing cylinders may be connected in various ways. The usual arrangement being to attach each piston-rod, by means of a parallel motion, to one end of a massive beam. If the beam is hung in the centre, the steam cylinder must have the same stroke as the blowing cylinder, but it will always be of much smaller diameter. A flywheel is usually, but not invariably, attached, and is often worked by a connecting-rod from a projecting nose at the steam cylinder end of the beam.

Direct-acting vertical engines are sometimes used. In these the cylinders are placed one above the other, the two pistons being attached to the same piston-rod. Either cylinder may be placed uppermost, and a flywheel is usually attached.

Where a smaller quantity of air at a higher pressure is required, smaller direct-acting engines with the cylinders placed horizontal, and driven at a higher speed, are often used.

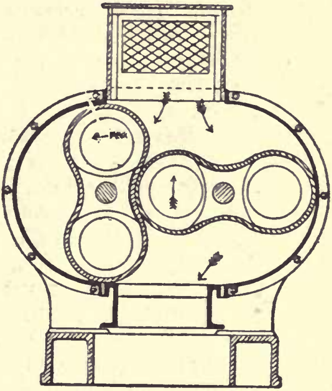


Fig. 18.—Roots Blower.

Blowers.—Two forms of blower are in use—The "Roots" and the fan. They are generally used where the quantity of air required is small or for suction.

The Roots blower consists of a casing in which two arms revolve at very high speed. These arms come at one period of their revolution into very close contact, and fit closely into the casing, and drive the air before them.

The fan consists of a set of curved arms revolving very rapidly in a casing. The air entering by an opening near the centre is swept round by the arms, and has a rapid motion imparted to it; coming opposite the opening in the casing it flies off at a tangent, and thus produces a strong current.

Equalising the Pressure.—The blast will leave the cylinder of the blowing engine irregularly, as the speed of the piston varies. To regulate this, large iron vessels, having a capacity several times that of the blowing cylinder, are attached to the main. The elasticity of the mass of air takes up the variations in pressure, and thus air is delivered to the furnaces in a steady stream.

Measuring the Pressure.—It is necessary to measure the pressure of the blast. This is usually done by a mercury manometer, a U-tube containing mercury, one end of which is open to the air, the other being attached to the blast main. If there be no pressure in the blast main the mercury will stand at the same level in both legs; if there be a pressure the mercury will fall in the leg attached to the blast main, and rise by the same amount in the other. The pressure of the atmosphere when the barometer stands at 30 inches is approximately 15 lbs. on the square inch, so that a pressure of 2 inches of mercury—*i.e.*, a difference between the levels of the mercury of 2 inches may be taken as being equal to a pressure of 1 lb. on the square inch.

Heating the Blast.—The blast is now usually supplied to the furnace hot, and is heated by being passed through heating stoves after leaving the blowing cylinder.

Where only a moderate temperature is required iron pipe heaters are frequently used.

The U-pipe stove consists simply of a set of ∩-shaped iron pipes usually elliptical in section, set over a fireplace, and connecting the pipe leading from the engines with that leading to the furnace. The air traversing these pipes becomes heated. These pipes are very apt to crack at the crown, and to remedy this the "pistol pipe" stove is often used. In this the heating pipe is half ∩, and is closed at the upper end; it is supported at the lower end, the upper being free, and the tube is divided by a wing, which passes from the bottom nearly to the top; so that the air passes up one side over the top of the wing, and down the other side. Pipe stoves may be heated either by coal fires or by furnace gas, but only a very moderate temperature can be attained.

Regenerative Stoves.—In most modern iron works the blast is used at a high temperature, and stoves, on the regenerative principle (see p. 69), are employed. As a type of this form of heating stove that known as the Whitwell may be taken.

The stove consists of a circular shell of fire brick cased with iron; inside, on one side of the centre, is a circular tube of fire brick—the combustion chamber—reaching nearly to the top of the stove; the space between this and the shell is filled up with rectangular tubes of fire brick, which communicate freely at the bottom. Communicating with the combustion chamber are three tubes, G, A, and H B, each provided with suitable valves, H B being lined with fire brick. Communicating with the bottom of the outer space, and, therefore, with the bottom of all the vertical brick tubes, are two other tubes, C and CB. Two stoves work together and supply one or more furnaces. The valves G, A, and C being opened, gas from the furnace enters through G, and air through A; combustion takes place in the combustion chamber, and the products of combustion pass upwards, then downwards through the rectangular tubes, and away to the chimney by C. When the brickwork is sufficiently hot the valves are all closed, and the gas and air are turned into the second stove. The valves C B and H B are now opened, the cold air enters by C B, passes upward through the brick tubes, down through the combustion chamber, and away by H B to the furnace, becoming

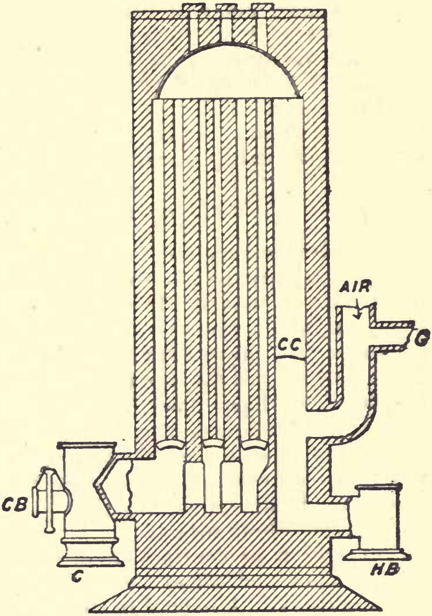


Fig. 19.—Whitwell Hot-Blast Stove.

C C, Combustion chamber.
C B, Cold blast pipe.
C, Chimney pipe.
G, Gas pipe.
H B, Hot blast pipe.

intensely heated in its passage. After about an hour the gases are again reversed, and so on, one stove being always in process of being heated while the other is heating the blast. At the top of the stove are arranged a series of little doors for cleaning the passages, and at the bottom are doors for removing the dust.

The Cowper stove is identical in principle, but the heating tubes are made of a chequer-work of either rectangular or hexagonal bricks. In the Ford and Moncur stove there are four separate chambers, each divided into passages by vertical walls, these passages being crossed by diamond-shaped bricks with the long axes vertical, which give a very large heating surface.

Temperature of the Blast.—With the pipe stoves a temperature up to about 600° F. can be attained; with the regenerative stoves the air can be heated up to 1,400° F. or thereabouts and can be kept sensibly constant by reversing the current hourly.

REVERBERATORY FURNACE.

The reverberatory differs in appearance and in principle from the blast furnace, as the fire and the material being heated are in separate chambers. The furnace consists essentially of a fireplace, on which the fire is made, and a hearth, on which the substance to be heated is placed, these being separated by a ridge of fire brick called the fire-bridge. The roof is highest over the fireplace and slopes down towards the flue end, where another bridge, the flue-bridge, may be placed. The roof is thrown from side to side, and to prevent the pressure of the arch forcing the walls outwards the furnace is strengthened by vertical stays, which are held together by tie rods passing across the roof of the furnace; these rods should be capable of adjustment, so that the expansion or contraction of the arch may be allowed for. At the sides, and sometimes also at the end of the furnace, are working doors. A hopper is often placed above for the delivery of the charge through a hole in the roof, and there is often an arched space under the hearth. The draught is usually produced by a tall chimney.

Reverberatory furnaces differ somewhat in arrangement, according to whether the charge is to be melted or drawn in a solid condition.

Roasting Furnaces.—In furnaces of this type the fireplace is small, as a very high temperature is not required. The hearth is flat, and is usually made of fire brick very carefully set; there are several working doors, the lower edges of which will be little, if any, above the hearth. The hearth must be so shaped that every portion of it can be reached with a rake or rabble from the doors, and, therefore, the space between the doors is usually made to project somewhat into the hearth. Very frequently close to each door there is an opening communicating with the arch under the furnace; this is closed by an iron plate whilst roasting is going on, and when the operation is over the plates are removed, and the charge is raked through the openings into the space below; or, if these are not provided, the charge is raked through the doors on to the floor. As a large quantity of air is required, openings are frequently made on or near the fire-bridge for its admission.

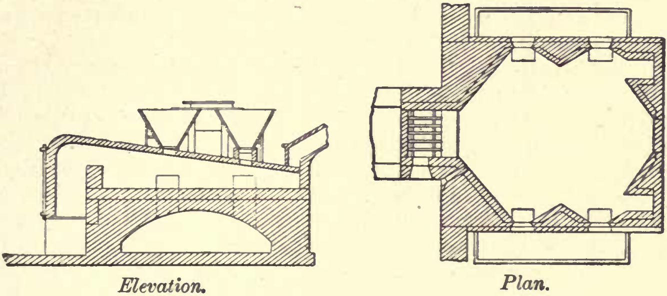

Elevation. *Plan.*

Fig. 20.—Roasting Furnace.

Double furnaces are often used; in these there are two hearths, one raised a few inches above the other. The charge is first let down on to the upper hearth; and when partially roasted it is moved down to the lower one, so that two charges are always being roasted together. Furnaces with three, four, or even more hearths are occasionally used.

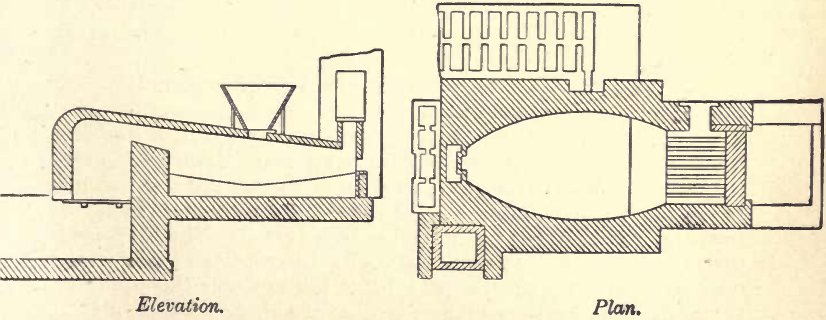

Elevation. *Plan.*

Fig. 21.—Melting Furnace.

Melting Furnaces.—In this type of furnace the fireplace is larger than in those used for roasting; the hearth is not flat, but slopes from all parts towards a point called the well, which is usually, but not always, under the middle door; from this well a tap hole communicates with the outside of the furnace, where a pot or moulds are placed to receive the molten material tapped out. The hearth is usually covered with refractory slag or some other substance which is not likely to be attacked by the molten charge. The doors are fewer in number than in roasting furnaces, and are so arranged that they can be luted so as to prevent the admission of air.

Atmosphere of a Reverberatory Furnace.—The atmosphere of a reverberatory furnace may be either oxidising or reducing. If fired with a thin layer of fuel unconsumed air will pass through and thus produce an oxidising atmosphere; and if the doors be opened so that air has free access the atmosphere will be very oxidising. When the furnace is worked with a thick layer of fuel no free oxygen can get through, and the flame will be reducing; if the doors be closed so that a limited supply of air only can get in, there will be an excess of combustible gas and the atmosphere of the furnace will be very strongly reducing. One great advantage of the reverberatory furnace is this power of altering the nature of the flame, but owing to the great loss of heat it is not an economical form.

Modifications of Reverberatory Furnaces.—The reverberatory furnace usually depends for its air supply on a tall chimney, but if the ashpit and fireplace be well closed a blast may be used; or, for roasting, a blast of air may be blown over the fire to ensure an oxidising atmosphere.

The hearth may be made to rotate horizontally, or the whole body of the furnace may be made in the form of a cylinder, either horizontal or inclined, which may be kept in rotation by machinery. Some furnaces of this type are very efficient for roasting.

Gas Furnaces.—Gas is readily applied to reverberatory furnaces; indeed, most gas furnaces are of this type. The gas producer may be attached directly to the furnace in place of a fireplace, for the gas producer is essentially little more than an enlarged fireplace. Usually, however, the gas producers are at a distance from the furnace, and the gas and air are heated by being passed through regenerators before being used. The fireplace is dispensed with, and at each end of the furnace are two rows of openings or ports, the lower set for the gas and the upper for the air. Combustion takes place as usual, and by regulating the relative quantities of air and gas the flame can be made oxidising or reducing as required, and can be very perfectly controlled.

Regenerative System.—The success of gas furnaces dates from the introduction of the regenerative system. The gas and air burn, and the products of combustion are not allowed to escape into the air, but pass away by the ports at the other end of the furnace to the regenerators, which are fire-brick chambers, filled up with a chequer-work of very refractory bricks, and are usually placed four under each furnace. The hot products of combustion are passed through two of these and soon heat the brickwork to a very high temperature. When the bricks are hot the current is reversed, air and gas are sent through the two hot regenerators, and the products of combustion are sent through the other pair on their way to the chimney. The current is reversed every hour or so, the brickwork in the regenerators is maintained at a bright-red heat, and as the air and gas acquire this temperature before reaching the furnace they produce a far higher temperature than it is possible to obtain by combustion in any other way.

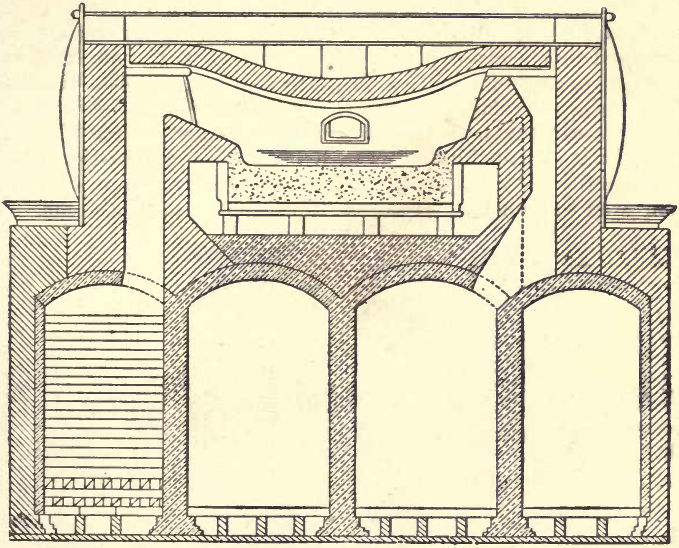


Fig. 22.—Siemens' Regenerative Furnace.

Siemens' New Form of Furnace.—In a modified form of furnace recently introduced, and described by Mr. Head, two regenerators (those for air) only are used. The gas producers are attached to the furnace, and whilst half of the products of combustion is passed through the regenerator, the other half is passed through the producer, any carbon dioxide it contains is at once decomposed ($CO_2 + C = CO$), and the gas, still very hot, passes to the furnace for combustion.

Heating in a Reverberatory Furnace.—The usual idea has been that to heat any substance on the hearth of a reverberatory furnace the flame should be made to play down upon it, and, therefore, the roofs of such furnaces have been designed to reflect the flame downwards on to the hearth. Mr. F. Siemens has pointed out that in furnaces where very high temperatures are obtained this leads to a very rapid destruction of the arch, and further that contact with solid materials by promoting dissociation limits the temperature of the flame; hence he holds that in gas furnaces the best results will be obtained by building the roof of the furnace high so as to leave a large combustion space, and heating the material on the hearth almost entirely by radiation from the hot masonry of the furnace and the hot flame.

A, Furnace.
g, Fire bars.
B, Ashpit.
c, Door with draught regulator.
D, Flue.
e f, Covering bricks.

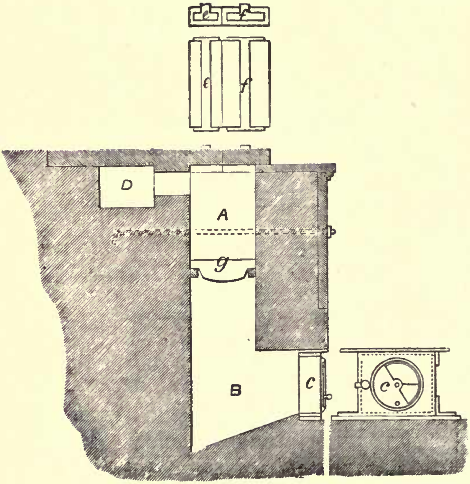


Fig. 23.—Assay Furnace.

CRUCIBLE FURNACES.

These are mainly used for the fusion of metals or other minor operations. As a type the laboratory assay furnace may be taken. This consists of a closed fire-brick chamber with fire bars at the bottom; beneath these is an ashpit, which should be fitted with a door for regulating the draught. A furnace of this type must be fed with coke or anthracite, and as the fuel is mostly burned to carbon monoxide it is not very efficient, though a very high temperature can be attained. The draught is almost always obtained by means of a chimney. A furnace of this type used for steel melting is described in Chap. XI.

MUFFLE FURNACES.

These are made in very many forms, according to the purposes for which they are to be used. The essential characteristic is that the material being heated does not come in contact either with the fuel or the products of combustion. As an example, the Tharsis close roaster may be taken. In appearance this is very like a reverberatory furnace, but the hearth is covered with an arch, above which the flame and products of combustion pass; they then descend by short vertical flues, travel along under the hearth, back again, and thence to the chimney. Any gases evolved during the roasting pass away by a separate flue.

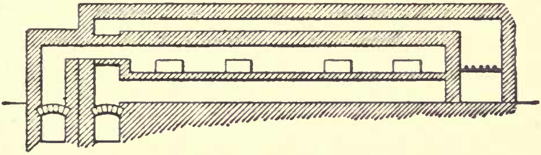


Fig. 24.—Muffle Furnace.

DISTILLING OR RETORT FURNACES.

These are made in very various forms, depending on the nature of the material being heated. They may be either of the muffle or crucible type, and will differ from others only by the products going off in the form of gas and being condensed in suitable condensers. Forms of distilling furnaces used for various purposes will be described later.

CHAPTER VI.

OCCURRENCE OF THE METALS IN NATURE.

ORES.

The metals do not occur in nature in a form ready for use, but always require treatment; it is the business of the metallurgist to take them as they occur in nature, separate them from other substances with which they are mixed or combined, and fit them for the purposes for which they are to be used. Any mineral substance containing enough of a metal to pay for extraction is called an "ore." The amount of metal necessary to constitute a mineral an ore will vary with the value of the metal and the nature of the mineral. A mineral containing 10 per cent. of iron would be valueless, with 10 per cent. of copper it would be worth treating, whilst with 1 oz. of gold to the ton, or ·0028 per cent., or even less, it would be regarded as an ore of gold.

The mineral which contains the metal is usually mixed with more or less mineral matter, which is of no value and which is called "gangue" or veinstuff.

OCCURRENCE OF ORES.

Gravels, &c.—Many metallic minerals are heavy, and when the rocks containing them are broken up or disintegrated by the agencies of nature, and the fragments are washed away by running streams, the heavy minerals are deposited and accumulate in the gravels. This is the nature of the alluvial gold deposits of Australia and California, of the stream tin works of Cornwall, Borneo, and other places.

Beds.—Sometimes ores occur in beds lying parallel to the strata of which the mass of the rocks is composed. In this case they are usually beds of shale or other material into which mineral matter has in some way penetrated. This is the nature of the clay-band and black-band ironstones of the coal districts, of the Northamptonshire, Cleveland, Lincolnshire, and other iron ores, and of the great pyrites deposits of Huelva. Allied to beds are various irregular deposits, such as many of the hæmatite deposits.

Mineral Veins or Lodes.—Most ores occur in mineral veins. These are usually cracks in the crust of the earth which have been filled up by the deposition of mineral matter; they often slope at a high angle, and may be nearly or quite vertical; at other times they are nearly horizontal; but even on the same lode the dip may vary very much. The veins cut across all the rocks of the district, and very often their richness in a particular metal seems to depend on the rock in which they are situated. Most veins are in rocks of considerable age, which have been subjected to great upheavals and other disturbances, so that the rocks have become broken and cracked. In the vein the ore may occur in layers, or may be irregularly mixed with the gangue or veinstuff, and the richness of the vein may vary very much in different parts.

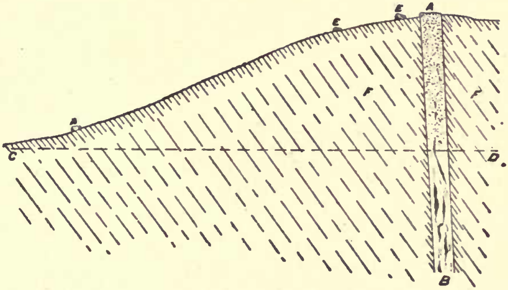


Fig. 25.—Mineral Vein.

AB, Lode; A, Craggy outcrop; E E E, Fragments of outcrop or shodestone; F, Enclosing rock or country; C D, Drainage or water level. The black strings represent ore, the unshaded portion being veinstones.

Pockets.—These are large cavities which may, or may not, be connected with veins, and which have become filled with mineral matter. These are often of large size, and are most common in limestone rocks.

FORMS OF ORE.

Native Metals.—When metals occur uncombined, or only combined with other metals, they are said to be native; the native metals may be, and usually are, mixed with large quantities of gangue or earthy matter.

Sulphides.—The most stable ores of the metals and those which occur at considerable depths are usually sulphides or arsenides, and are often very complex.

Oxide Ores.—Where mineral veins come to the surface the mineral is exposed to the action of moisture and air and undergoes gradual changes, the sulphides being converted into sulphates, carbonates, oxides, native metals, or, occasionally, chlorides, and, frequently, some of the constituents are removed in solution. It is a general rule, to which there are few exceptions, that when at the outcrop of veins, oxides, carbonates, or similar ores are found, they will pass into sulphides at greater depths.

IRON.

Iron does not occur native, except in meteorites which have fallen on the earth from space outside. In these it is always alloyed with nickel. The oxidised minerals are the only ores of importance.

Hæmatite, Fe_2O_3, sometimes called red hæmatite, is a very valuable ore. Externally it is black or red, but is always red when powdered, and makes a red streak on a hard substance. It is in its purest forms so hard that a knife will not scratch it, and sometimes occurs in rounded kidney-shaped lumps (hence the name kidney ore), which have a distinctly radial structure. When pure it contains 70 per cent. of iron, and is usually very free from sulphur, phosphorus, and other deleterious substances; hence it is used for making hæmatite pig for steel making.

A dark shining variety of the same mineral called specular iron ore, or mirror iron ore, occurs in Elba, from whence it is imported. Softer earthy varieties of a red colour occur in some places, and are known as red ochre, red raddle, puddler's ore, &c. These are usually mixed with earthy matters.

In Great Britain these ores only occur in any quantity in Cumberland. Very large quantities are imported from Bilbao, Algeria, and other places. Hæmatite is very abundant in Canada and the United States, where it is the principal ore worked.

Brown Hæmatite.—Ferric oxide forms several hydrates—limonite, $2Fe_2O_33H_2O$; göthite, $Fe_2O_3H_2O$; and turgite, $2Fe_2O_3H_2O$. These are generally grouped together as brown hæmatites, or brown iron ores. Externally they often resemble hæmatite, though they are more frequently brown than black, and they give a brown streak. The hard varieties are as hard as hæmatite, but soft brown earthy varieties are far the most common.

Brown ores occur in Northamptonshire, Lincolnshire, the Forest of Dean, and other localities. They often contain considerable quantities of sulphur and phosphorus, and cannot be used for making high-class iron. Brown ores are imported from Bilbao and Algeria; these are as pure as the red ores from the same localities.

Magnetite, Black or Magnetic Iron Ore, Fe_3O_4.—This is black in colour, so hard that a knife will not scratch it, and yields a black powder; it has a metallic lustre and is always magnetic, some pieces, but not all, showing polarity. It contains, when pure, 72 per cent. of iron, and is usually free from sulphur and phosphorus. It occurs only in small quantities in Great Britain, the principal European localities being Norway and Sweden; and it occurs in considerable quantities in America. Very similar to magnetite are titaniferous iron ore (ilmenite), $Fe_2O_3Ti_2O_3$; franklinite, Fe_2O_3ZnO, worked for iron and zinc in New Jersey; and chrome iron ore (chromite), Cr_2O_3FeO, the principal source of chromium.

Siderite, Chalybite, or Spathic Iron Ore, $FeCO_3$, is a buff or brown substance with a pearly lustre, and so soft that it can be readily scratched with a knife. In its pure form it is of little importance as an ore of iron since it occurs in very few localities. The only British locality of importance is the Brendon Hills in Somersetshire. It usually contains manganese, and is, therefore, sometimes used for making manganiferous irons. Mixed with various impurities it is, however, a very important iron ore. With clay it constitutes the clay band ironstone of the coal fields of the Midlands and South Wales. This ore is clay or shale mingled with a considerable quantity of carbonate of iron; sometimes it occurs in beds, and very often in nodular concretions disseminated through beds of shale.

Black-band ironstone, which occurs in the Lanarkshire and Staffordshire coal fields, and to a smaller extent in other coal districts, is carbonate of iron mixed with bituminous or coaly matter, and is, therefore, dark coloured; it is sometimes coaly in appearance, though more often it is hard and shaly, and frequently contains enough carbonaceous matter to burn.

Cleveland ironstone, which occurs in rocks of Liassic age in the North of Yorkshire, is carbonate of iron mixed with a small quantity of silicate and with carbonate of lime. It is usually of a pale-green colour, and is free from carbonaceous matter. These three forms of ironstone are impure; they contain phosphorus in considerable quantity and therefore yield a phosphoric pig.

Sulphides and Arsenides.—Several of these are known, but they are of no value as ores of iron. Iron pyrites, FeS_2, has a brass-yellow colour, and is so hard that a knife will not scratch it; it is very widely distributed and is used as a source of sulphur for the manufacture of sulphuric acid. The residue, after the sulphur is burnt out, consists mainly of ferric oxide, and is called blue billy or burnt ore. Some varieties contain a few per cent. of copper; these are treated for the extraction of the copper, and the residue is called purple ore.

Marcasite or white iron pyrites has the same composition as pyrites. Mispickel or arsenical pyrites, FeAsS, is very similar, except that it is white in colour. These minerals often carry gold.

COPPER.

Native Copper.—This occurs in several localities, but in few in large enough quantity to work. It is copper coloured, but tarnishes or becomes much darker on exposure to the air; it occurs in flat plates or crystalline masses, or in shots disseminated through rocks. The principal locality is the copper district to the south of Lake Superior. There it occurs in some places in vast masses of nearly pure copper; in others it cements together the grains of sandstone rock. The native copper of this district often contains crystals or strings of native silver and is very free from deleterious impurities.

Both oxides occur in nature.

Cuprite (Ruby Copper Ore), Cu_2O, has a brown or red colour and yields a red powder. It contains 88·8 per cent. of copper and 11·2 per cent. of oxygen, and is, therefore, a very valuable ore of copper, especially as it is usually free from arsenic and other objectionable impurities. The principal locality in which it has been worked is Australia, though it occurs in small quantities in other places—*e.g.*, Cornwall, Siberia, &c. Tenorite and melaconite are forms of the black oxide, CuO; they are rare and of no practical importance.

Two carbonates are known.

Malachite (Green Carbonate), $CuCO_3, CuO, H_2O$, is a basic carbonate; it has a green colour and very often a banded structure, the bands showing different shades of green. It is soft, and gives a pale-green streak. It contains 57 per cent. of copper, and is a valuable ore; and is also used for ornamental purposes.

Azurite (Blue Carbonate), $2CuCO_3, CuO, H_2O$, is also a basic carbonate; it has a fine blue colour, and is similar in its other properties to malachite.

These two ores often occur together, and have been worked on a large scale in Australia. Malachite occurs in Siberia in considerable quantities, and azurite occurs at Chessy in France, whence it is sometimes called chessylite. Bisbee, Arizona.

These ores are usually free from deleterious impurities, and, therefore, yield a very pure copper.

Among the other coloured ores may be mentioned chrysocolla, $CuSiO_3, 2H_2O$, which is very like malachite; it occurs in Chili and other places; and atacamite, $CuCl_2, 3CuO, H_2O$, occurring in Chili and Peru. These, however, are not of much importance as ores.

Several sulphides are known, and they are the most important of the ores of copper.

Copper-glance, Cu_2S.—This mineral has a black colour, a metallic lustre, and is so soft that it can be cut with a knife. It contains 79·8 per cent. of copper. It occurs in Cornwall, Chili, and other districts, and is a valuable ore.

Copper Pyrites, $CuFeS_2$.—This is the most common ore of copper. It has a fine yellow colour, often with an iridescent tarnish; it is softer than iron pyrites, can be easily scratched with a knife, gives a blackish streak, and contains 34·6 per cent. of copper. It occurs abundantly in Cornwall and in most other copper districts.

Erubescite, Cu_3FeS_3.—This ore has a somewhat variable composition. It is brittle, and can be scratched with a knife, but not cut. It has a reddish-bronze colour, on a freshly fractured surface, but rapidly becomes covered with an iridescent tarnish showing principally shades of blue, and, therefore, it is often called "peacock" copper ore, and by the miners horse-flesh ore. It occurs principally in Cornwall.

Fahl Ore, $4Cu_2S, Sb_2S_3$.—This ore has a steel-gray colour, is soft and brittle. The composition is variable; it often contains zinc, iron, silver, and arsenic. Owing to the presence of the antimony and arsenic it is difficult to work as copper ore, but it often carries silver enough to pay for treatment. It occurs in Cornwall, The Harz, Chili, Mexico, and other copper and silver districts.

LEAD.

Native lead and lead oxide occur, but are so rare as to be of no importance.

Cerussite, White Lead Ore, $PbCO_3$, occurs abundantly at the surface in many lead districts, notably at the Lead Hills, Lanarkshire. It is a white earthy mineral, often well crystallised, so soft that it can be easily scratched with a knife, and very heavy (specific gravity 6·45). It contains, when pure, about 75 per cent. of lead, but owing to its earthy appearance was, in many lead districts, formerly thrown away. It is the most easily smelted of all lead ores.

Anglesite, $PbSO_4$, also a heavy white mineral, occurs in many lead districts, but never in sufficient quantity to be worked by itself.

Many double salts also occur; these are mostly white, earthy, and heavy. Lead chromate, crocoisite, $PbCrO_4$, is light red, so is the molybdate, wulfenite, $PbMoO_4$; whilst the phosphate, pyromorphite, $3Pb_3(PO_4)_2 . PbCl_2$, is yellowish-green or brown, and is commonly called by the miners "linnets," green or brown, as the case may be.

Galena, PbS.—Has a steel-gray colour and brilliant metallic lustre, which, however, tarnishes, and sometimes becomes iridescent. It is very frequently crystallised in cubes, and the cleavage is cubic, very perfect, so that it breaks up into more or less rectangular fragments. It contains 86·6 per cent. of lead, can be scratched with the knife, and has a specific gravity of about 7·5.

It occurs widely distributed all over the world; Cornwall, North Wales, Derbyshire, Northumberland, Lanarkshire, and the Isle of Man being the principal British localities; it also occurs in many parts of Europe, the United States, South America, &c. It is the principal ore of lead, and almost all lead smelting processes are designed for its reduction. It almost always contains silver, sometimes to such an extent that it may be regarded rather as a silver than as a lead ore.

Many double sulphides of lead with other metals are known, but they are of little practical importance.

ZINC.

Native zinc does not occur. The oxide **Zincite, red zinc ore,** ZnO, occurs in red grains disseminated through franklinite at Franklin, New Jersey.

Franklinite, Fe_2O_3ZnO, occurs in the same locality; it is black with a metallic lustre, and closely resembles magnetite. Both these ores usually contain manganese.

onite **~~Calamine~~**, $ZnCO_3$, is a white mineral, but is often coloured brown by the presence of iron. When in crystals it has a vitreous lustre, and can hardly be scratched with a knife, but often occurs in much softer earthy masses. It occurs with other zinc minerals in Belgium, Silesia, and other European localities, in the United States, and, to a small extent, in Derbyshire. It is a valuable zinc ore.

mine **~~Smithsonite~~**, $ZnSiO_4,H_2O$, is very similar to calamine, and occurs with it in many localities. It is much more difficult to reduce, and is, therefore, of less value as an ore.

Willemite, $ZnSiO_3$, is very similar, but is much less abundant.

Blende, ZnS.—This is the most abundant and important ore of zinc. Its colour is variable, from colourless through shades of yellow and brown to black; the Cornish blendes are usually black, and are called by the miners black jack; the dark colour is due to the presence of iron, and the yellow varieties often contain cadmium; the lustre is resinous, the mineral is often translucent, can be scratched with a knife, is very brittle, and has a specific gravity of about 4.

It occurs usually with galena in Cornwall, Derbyshire, the Isle of Man, and other British localities; in Belgium and other parts of Europe; and many parts of the United States. In Australia and some other districts the blende-galena ores carry a considerable quantity of silver.

TIN.

The only ore of tin is the oxide, **Cassiterite or tinstone,** SnO_2. This is black or brown in colour, with a bright lustre when in crystals, but dull in the massive forms. It is so hard that a knife will not scratch it, and it has a specific gravity of between 6 and 7. Owing to its hardness, durability, and great density it has accumulated in gravels formed by the washing down of the rocks in which it occurred. These gravels constitute the stream tin deposits from which enormous quantities of tin have been obtained in Cornwall, the tin stone occurring in rounded pebbles. The deposits of the Straits of Malacca, Banca, Borneo, and Australia are of this character. In Cornwall the tin is now obtained from the veins. The Cornish tin deposits are known to have been worked for thousands of years. Tin ore alone is not widely distributed; there are no important deposits either on the Continent of Europe or in the United States.

Stannine, tin pyrites or bell-metal ore, a sulphide of copper, iron, and tin, occurs in Cornwall, but is not worked as an ore.

GOLD.

Gold almost always occurs **native,** either in small flakes or grains, or in large pieces called nuggets, scattered through river gravels, usually of recent age, from which it can be extracted by washing; or disseminated through rock or quartz veins or reefs, in which case the rock must be crushed before the gold can be obtained. When the gold is not associated with pyrites it is easily attacked by mercury or chlorine, and the ores are said to be free milling; when it is associated with pyrites, although it is probably still in the metallic form, the pyrites protects it from the action of reagents, so that it cannot be so easily obtained, and the ores are said to be refractory or rebellious.

Alluvial gold occurs in river gravels all over the world. Considerable quantities have been obtained from Wales, Wicklow, the Lead Hills, and many other places in Europe. The great alluvial deposits, however, are those of Australia and California, and washings on a smaller scale are carried on in many other parts of the world. Large quantities of gold are now obtained from Australia, South Africa, California, and parts of America and Russia.

Native gold is yellow in colour, soft, and has the general characters of the metal; it is never pure, but always contains considerable quantities of silver, and often copper and other metals.

The only compounds of gold occurring in nature are some tellurides, which occur in Transylvania and in Colorado, ~~but which are very little worked~~.

Many samples of galena, and various copper ores carry small quantities of gold.

SILVER.

Native silver occurs in most silver ore districts in crystals or arborescent filaments running through masses of silver ore. It is silver white, but tarnishes black on exposure to air.

Horn silver, $AgCl$, is a valuable ore; it occurs in masses, rarely in crystals; it is very soft and can be cut; it is usually white, but on exposure to the light it darkens till it becomes black. The bromide and iodide of silver are very similar.

Silver glance, Ag_2S, is very like copper glance in appearance; its colour is lead gray; it has a metallic lustre, is soft and easily cut, and has a specific gravity of about 7·2.

There are a large number of double sulphides of silver with arsenic and antimony, among which may be mentioned **Pyrargyrite, Dark red silver ore,** $3Ag_2S, Sb_2S_3$; **Stephanite, brittle silver ore,** $5Ag_2S, Sb_2S_3$; **Proustite, light red silver ore,** $3Ag_2SAs_2S_3$.

Silver ores are widely distributed, and silver is of such value that a small quantity of any ore scattered through a considerable mass of gangue is worth working. There are no silver ores proper in Great Britain, though small quantities have been found in Cornwall. They occur in considerable quantity in Hungary and Saxony, in Mexico, Chili, Peru, and other parts of South America, and in Nevada, and other parts of the United States.

Galena, blende, and other minerals, as already mentioned, often carry enough silver to render them valuable as ores.

MERCURY.

Native mercury occurs in liquid globules in other mercury ores.

Cinnabar, HgS, is the only ore of mercury. It has a fine red colour, and yields a brilliant red powder (vermilion). It can be easily scratched with a knife, and has a specific gravity of about 8·5. It occurs in Spain, in Austria, and other European localities, Chili, Peru, Mexico, and California. In Borneo it occurs in rounded pebbles in river gravels.

hat a "break"!

CHAPTER VII.

PREPARATION OF THE ORE FOR THE SMELTER.

The ores as raised to the surface are rarely in a fit condition for smelting; they are in large and irregular lumps, and are usually mixed with gangue and earthy materials, which should be separated as far as possible.

Hand Picking.—Many ores are simply broken up by a hand hammer, carefully examined, and any pieces which do not seem to contain metal picked out and thrown on one side. For many processes this is all that is required, but in others it is not sufficient, and fine crushing is necessary.

Stone-Breaker.—For coarse crushing a stone-breaker is generally used; this consists of a heavy jaw which is moved backwards and forwards by a strong arm, driven by an engine; as the arm advances it crushes the ore, and as it retreats the crushed ore falls through into a receptacle, whilst a fresh portion descends to be broken by the next stroke.

Stamp Mill.—This is one of the most common forms of apparatus for fine crushing. It consists essentially of two parts, the stamp, and the mortar in which it works. The stamp consists of an iron stem with a head, to the bottom of which is attached a shoe of hard iron or steel. The stamp rod works in guides, and carries a projecting tappet which is acted on by a cam attached to a revolving axis. As the cam tooth catches the tappet it raises the stamp; and as it clears it, the stamp falls, and, by its impact, crushes the ore beneath it. The stamps work in a mortar. This is an iron box open at the top, and closed in the front by a perforated screen; at the bottom it has a die to receive the force of the blow of the stamp. Five stamps usually work side by side in the same mortar, and form a battery. The ore is fed into the mortar into which a stream of water is also supplied; as the material is crushed sufficiently finely, it is carried through the screen by the running water (see p. 208).

Rolls are sometimes used for fine crushing. These consist of a pair of chilled iron rolls mounted in very strong bearings and forced together by weights, or by a powerful spring so arranged that should anything too hard for the rolls to break come between them they will yield and let it through.

Dressing.—The object of dressing is to separate the valuable mineral from the gangue with which it is mixed. The methods used always depend on the separation due to differences in specific gravity, and to the fact that if particles of various weight be mixed up with running water the heavy particles will settle first whilst the lighter will be carried further. The simplest arrangement is a series of boxes, each a little larger than the one before, so that the stream of water carrying the ore is gradually reduced in speed; as the heavier materials will be deposited first, a separation may be thus effected. It is obvious that for the separation to be anything like complete the particles must be approximately of the same size; where this is not the case a preliminary "sizing" is often resorted to.

Another form of apparatus very commonly used is the jig. This consists of a vessel divided into two by a vertical division, on one side of which is fitted a piston and on the other a sieve or screen, the holes in which are a little larger than the grains of the material to be jigged; on this screen is placed a layer of pebbles just large enough not to pass through the holes; the ore is charged on to this; the piston on the other side of the partition is depressed; this forces the water up through the material being treated, and the ore, being heavier than the gangue, partially settles with the stones, whilst the lighter gangue remains at the top; as this action is continued the ore finds its way down through the screen, whilst the gangue is carried away by the overflow water.

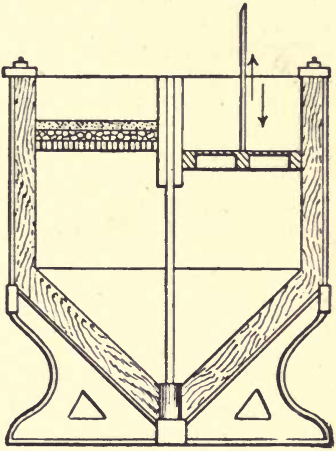


Fig. 26 —Jig.

Ores which will Concentrate.—In order that an ore may concentrate it must be finely powdered; hence iron and other ores which are required in lumps for blast furnace use are never concentrated, but only hand-picked. There must also be considerable difference in specific gravity between the ore and the gangue, and the ore must not be so brittle that it will break down into too fine a powder, and so be lost in the slimes. Ores of tin and lead are usually concentrated; copper sulphides, owing to their brittleness, cannot conveniently be so treated.

Calcination.—Very frequently the ores require calcination, and this is usually done at the mine before the ore passes to the smelter. If the ore has been dressed and is, therefore, in a fine state of division, it must be calcined on the hearth of a reverberatory furnace; if it be in lumps, it may be calcined in heaps or in kilns. It is well to distinguish between calcination and roasting, though the terms are very frequently used as synonymous. In calcination heat only is necessary to produce the required changes; in roasting the presence of air is essential, as impurities have usually to be oxidised away. For roasting, the charge is best powdered and treated in a reverberatory furnace.

Calcination in Heaps.—This method is very largely used for the calcination of iron and other ores which are in lumps. A layer of wood or coal is spread on the ground, and on this is placed a layer of ore, then another layer of coal, then more ore, and so on, till the heap is 5 or 6 feet in height. The heap is lighted and combustion goes on till all the combustible matter is burned, and the heat has expelled the volatile constituents from the ore. Great care is required in sheltering the heap from the wind or the burning may be very irregular. Many blackband ironstones and pyritous ores can be burned in heaps without the addition of any fuel, except a little to start combustion. Burning in heaps is cheap, but is apt to be very irregular; one part of the heap often being almost unchanged, whilst another part may have been heated to a temperature high enough to fuse it.

Calcination in Stalls.—This is the simplest advance on heap calcination. Three walls are erected at right angles so as to enclose a space; in this the floor of fuel is laid down, then the ore, then more fuel, more ore, and so on. The front is then loosely built up, openings being left for the admission of air.

Calcination in Kilns.—This is by far the most satisfactory and economical method of calcining ores in lump. The calcination is much more uniform and the consumption of fuel is much less. Calcination of Cleveland iron ores in the Gjers kiln (see p. 52) only consumes about ¾ cwt. of small coal per ton of ore.

Effects of Calcination and Roasting.—These are usually simple. Water and organic matter, if present, are expelled. Carbonates are decomposed, oxides being left. In most cases the reaction which takes place is $RCO_3 = RO + CO_2$—*e.g.*, $ZnCO_3 = ZnO + CO_2$; but in the case of iron carbonate the reaction is different, ferrous oxide being so unstable that it cannot exist under ordinary conditions. In this case ferric oxide is left, the reaction being $2FeCO_3 = Fe_2O_3 + CO + CO_2$.

Sulphides behave differently under different conditions. If air be excluded they undergo little change, except in the case of pyrites which loses sulphur, $2FeS_2 = Fe_2S_3 + S$. In presence of air they are all decomposed with the formation of oxides and sulphates, the latter being decomposed into oxides and sulphuric anhydride, if the temperature be sufficiently high.

Weathering.—Weathering, or exposure to the weather, is occasionally resorted to before or after roasting. Many argillaceous rocks which are hard at first, soften and crumble on exposure to the air. Sulphides are oxidised to sulphates; in the case of pyrites with separation of sulphur. Pyrites oxidises very rapidly and helps the disintegration of rocks which contain it. On exposure to rain the sulphates are partially dissolved.

CHAPTER VIII.

METALLURGICAL PROCESSES.

Smelting.—The term smelting is one which is generally and loosely used to include all the operations whereby the metal is extracted from its ores and purified, though it may conveniently be restricted to those processes in which fusion by heat is resorted to.

Dry and Wet.—Metallurgical process may in general be divided into dry and wet. In dry processes the operations are carried on by the aid of heat, and if fluidity is necessary fusion is resorted to. In wet processes the reactions are effected by means of reagents in solution. Which class of process will be best will depend on circumstances, but, as a rule, dry processes are to be preferred, and they are used in most cases when fuel is to be had at a moderate price. The term **leaching** means the dissolving out of a soluble constituent by means of solvents. Wet processes are, therefore, often called leaching processes.

Reduction.—In metallurgy this term strictly means the separation of a metal from an oxidised ore, and hence is used in a more restricted sense than in chemistry. It is often used to mean the separation of a metal from any ore whether oxide or not.

Reducing Agents.—Of these there are but few in practical use. The most important is carbon, which is used either as coke, charcoal, or coal, as may be most convenient or economical. In the laboratory argol (crude potassium tartarate) is often used, as on heating it is converted into a mixture of potassium carbonate and carbon. In almost all cases of reduction by carbon, the carbon monoxide is formed—*e.g.*, $SnO_2 + 2C = Sn + 2CO$, $Fe_2O_3 + 3C = 2Fe + 3CO$; but in the case of lead oxide the temperature of reduction is so low that carbon dioxide is produced, $2PbO + C = 2Pb + CO_2$. Carbon monoxide (CO) is largely used as a reducing agent, and in the blast furnace it is almost entirely by its agency that reduction is brought about, thus, $Fe_2O_3 + 3CO = 2Fe + 3CO_2$.

In the laboratory potassium cyanide is sometimes used, it being readily oxidised to cyanate, $SnO_2 + 2KCN = Sn + 2KCNO$, but it is far too expensive for use on the large scale.

Calcination.—This term has already been defined as the expulsion of volatile matters by heating with or without access of air.

Roasting.—This is heating with free access of air for the purpose of oxidising the substance or some constituent of it. If the material be in lumps it may be roasted in heaps or kilns, a method which, however, is never very efficient. Roasting is best carried out either on the powdered or sometimes the fused material on the hearth of a reverberatory furnace. In roasting a powdered substance it must be turned over very frequently so as to expose fresh surfaces to the air, and great care must be taken that the temperature is not allowed to rise high enough to cause softening or clotting, or good roasting will be impossible.

Roasting operations may be classified into—

(1) *Oxidising Roasting*, the object of which is to convert the material into oxide, as when tin is roasted to oxide (putty powder), $Sn + 2O = SnO_2$.

(2) *Desulphurising Roasting*, the object of which is to remove sulphur in the form of sulphur dioxide. The action of air on sulphides is not the same for all sulphides.

Most sulphides are converted partially into oxides and partially into sulphates—*e.g.*, copper sulphide, $CuS + 3O = CuO + SO_2$, $CuS + 4O = CuSO_4$, and on stronger heating the sulphates are decomposed into oxides and sulphuric anhydride—*e.g.*, $CuSO_4 = CuO + SO_3$. Iron sulphide behaves similarly, but ferric oxide is always formed, $2FeS_2 + 11O = Fe_2O_3 + 4SO_2$, $FeS_2 + 6O = FeSO_4 + SO_2$, $2FeSO_4 = Fe_2O_3 + SO_2 + SO_3$.

Antimony and tin sulphides are converted into oxides,

$$Sb_2S_3 + 9O = Sb_2O_3 + 3SO_2 . SnS_2 + 4O = SnO_2 + 2SO_2.$$

Sulphate of lead is among the most difficult of the sulphates to decompose. Silver forms no oxide; it roasts to metal, unless mixed with large quantities of other sulphides when sulphate is formed; and if this be very strongly heated metallic silver is left. Mercury sulphide forms neither oxide nor sulphate, but liberates the metal, $HgS + 2O = Hg + SO_2$.

When a sulphide has been roasted till there is no sulphur remaining, which is usually judged by its no longer smelling of sulphur dioxide, it is said to be roasted, "sweet" or "dead."

(3) *Purifying Roasting*, the object of which is to remove oxidisable impurities, as for instance to oxidise antimony out of lead, or silicon out of iron.

Bessemerising.—This term is generally used to describe any process in which oxidation is brought about by blowing air through molten material contained in a closed vessel or converter.

Scorification is the roasting an alloy with the object of removing the more oxidisable metal, though the term is almost always restricted to the oxidation of lead. The alloy, say of lead and silver, is melted and exposed to an oxidising atmosphere; the lead oxidises; the litharge formed fuses and forms a slag on the surface, which can be removed, whilst silver or gold will remain in the residual unoxidised lead. In scorification assays the ores of the precious metals are melted with lead in an oxidising atmosphere.

Cupellation differs very little from scorification. The essential difference is that the oxidation is carried out on a hearth of some absorbent material which will absorb the fused oxide, but not the metal. The material always used for the cupel is bone ash, ground burnt bones.

Reduction of Sulphides.—The separation of the metal from sulphides may be brought about in several ways:—

(1) The sulphide may be roasted "sweet," and the oxide thus formed reduced by carbon. The complete removal of sulphur by roasting is, however, often so difficult as to be practically impossible.

(2) In some cases the sulphide may be partially roasted, and the metal liberated by melting down the residual sulphide with the oxide and sulphate thus formed—*e.g.*,

$$2PbO + PbS = 3Pb + SO_2 \quad PbSO_4 + PbS = 2Pb + 2SO_2.$$

This reaction is made use of in the smelting of lead and copper ores. Such processes are frequently spoken of as "air reduction processes."

(3) Some sulphides are reduced by being heated in air, the oxides of the metals being decomposed at a red heat—*e.g.*, $HgS + 2O = Hg + SO_2$. $Ag_2S + 2O = 2Ag + SO_2$.

(4) Some sulphides are reduced by fusion and with a metal—*e.g.*, lead and antimony sulphides are readily reduced by fusion with iron, $PbS + Fe = FeS + Pb$.

Chloridising (Chloridising Roasting). — In some "wet processes," chiefly in the treatment of silver and copper ores, it is necessary to convert the metals into chlorides. This is almost always done by roasting with salt or with a mixture of salt and some other substance.

Chlorination.—This is the conversion of metals into chlorides by the action of free chlorine.

Distillation.—When a metal melts and then passes into the gaseous condition, it is said to be distilled, and the operation is called distillation; when the metal passes at once into the gaseous condition, and, therefore, condenses as a solid, it is said to be sublimed, and the operation is called sublimation.

Liquation.—When two metals which do not alloy are melted together and rapidly cooled, they appear to form a homogeneous mass; but if this be heated up to the fusion point of the more fusible metal, it will melt and run or "liquate" out. Many alloys are not homogeneous, and the metals tend to "liquate" on slow cooling.

Amalgamation.—Certain metals dissolve readily in mercury, forming amalgams. This operation is called amalgamation, and is largely used in the treatment of gold and silver ores.

Regulus.—In some operations, especially in copper and nickel, smelting the metal (usually copper or copper and iron) is obtained in combination with sulphur. Such a compound is called a regulus or matt (matte).

Speise is a compound of nickel (with or without other metals) with arsenic, obtained in some processes of nickel smelting.

Fluxes.—In many smelting operations the addition of a flux is necessary. The main function of the flux is usually to form with the gangue of the ore, or other earthy materials that may be present, a slag sufficiently liquid to allow the metal to settle, and which can be tapped off. Sometimes the function of the flux is merely to form a larger quantity of slag so as to keep the fused metal covered.

Very few fluxes are used on the large scale. In iron smelting lime is almost invariably used, and sometimes clay and silica. Fluorspar is occasionally used in some operations to increase the quantity and fusibility of the slag. Many fluxes are used in the laboratory which are too expensive for use on the large scale.

Neutral Fluxes—*i.e.*, those which are neither oxidising nor reducing.

Sodium carbonate is usually used in conjunction with potassium carbonate, which increases its fusibility. It forms a good covering slag, though it is always somewhat pasty. It decomposes silicates with the evolution of carbon dioxide and the formation of fusible sodium silicate.

Common Salt fuses very readily to a very fluid liquid, which usually rises to the top of the mass, and thus forms a covering flux.

Borax melts readily and forms a fluid liquid; it dissolves most metallic oxides, forming even with those which are difficultly fusible readily fusible compounds.

Fluorspar is rarely used.

Litharge is very frequently used in gold assaying; it combines with silica, forming very fusible silicates of lead; it combines also with metallic oxides, and acts as an oxidising flux giving up its oxygen to any easily oxidisable substances with which it is in contact, lead being separated.

Reducing Fluxes.—These are substances which take up oxygen on heating; they are carbon, argol, starch, flour, &c.

Oxidising Fluxes.—Litharge, already mentioned, is the most important of these; and nitre is sometimes used.

Slags. — The earthy materials present in a charge being smelted, and the fluxes added, are usually tapped from the furnace as a liquid slag.

A good slag should be sufficiently liquid to be removed from the furnace easily; if it is to be tapped out it must be quite liquid; if it is to be raked out it must be more or less viscous; but sufficiently liquid to allow the metal to readily settle. In a few operations slags are produced which are so refractory that they have to be drawn from the furnace in a solid or pasty condition; such slags always retain a considerable quantity of metal or unreduced ore scattered through them. On the large scale slags are almost always silicates. Silicates may be regarded, according to the dualistic system, as being built up of a basic oxide, say lime or alumina, combined with the acid oxide silica; and they are usually classified according to the ratio which the oxygen combined with the metal (in the base) bears to that combined with the silicon (in the acid).

General formula.	Oxygen ratio.	Name.	Examples.	
			Calcium silicate.	Aluminium silicate.
$4RO, SiO_2$	2 : 1	Subsilicate	$4CaO, SiO_2$	$4Al_2O_3, 3SiO_2$
$2RO, SiO_2$	1 : 1	Monosilicate	$2CaO, SiO_2$	$2Al_2O_3, 3SiO_2$
RO, SiO_2	1 : 2	Bisilicate	CaO, SiO_2	$Al_2O_3, 3SiO_2$
$2RO, 3SiO_2$	1 : 3	Trisilicate	$2CaO, 3SiO_2$	$2Al_2O_3, 9SiO_2$
$4RO, 3SiO_2$	2 : 3	Sesquisilicate	$4CaO, 3SiO_2$	$4Al_2O_3, 9SiO_2$

If the monosilicate $2RO, SiO_2$ (R_2SiO_4) be taken as the normal silicate, then any others in which the quantity of base is larger will be basic silicates, and any in which it is lower will be acid silicates.

The fusibility of a slag depends on the nature of the metals which it contains, and on the type of silicate to which it belongs. The silicates of lead, potassium, and sodium are the most fusible, those of iron and copper standing next; whilst those of lime and alumina are among the least fusible. A silicate which contains only one base is usually less fusible than one which contains two bases (a double silicate), the silicate of lime and alumina for instance being more fusible than either the silicate of lime or of alumina, and, therefore, when silicate slags are produced they are generally made to contain at least two bases.

The subsilicates are the most fusible, and the fusibility decreases as the quantity of silica increases. The more basic silicates melt to a limpid liquid and are glassy after solidification, whilst the more siliceous silicates often form a viscous mass, and on solidification are stony. The basic silicates are usually heavy and are sometimes dark coloured.

Most silicates are white; silicate of iron is black; silicate of copper is dark brown. Many slags contain other substances besides silicates, which may alter the colour and otherwise modify their properties.

CHAPTER IX.

IRON—PREPARATION OF PIG IRON.

Properties of Pure Iron.—Pure iron is very difficult to obtain even on a small scale. It is grayish-white in colour, grayer than platinum; it is soft and malleable; very readily magnetised, but does not retain its magnetism, and is very difficultly fusible. Exposed to moist air in presence of carbon dioxide it rusts rapidly. It dissolves freely in most acids; in the absence of oxidising agents forming pale green solutions of ferrous salts; and in the presence of such agents forming yellow solutions of ferric salts. Iron combines readily with sulphur, phosphorus, and arsenic, forming fusible compounds. At very high temperatures it combines with carbon, forming pig or cast iron; it combines readily with silicon and it alloys with most metals. It forms two stable oxides, red or ferric oxide, Fe_2O_3; and black or magnetic oxide, Fe_3O_4; the former being converted into the latter by heat. Both the oxides are readily reduced by carbon, hydrogen, or other reducing agents, the iron being left in a spongy form, unless there is an excess of carbon and the temperature be very high.

Forms of Iron.—Iron comes into the market in four forms differing from one another in physical and chemical properties and in the methods by which they have been produced. The essential distinction is the amount of carbon which is present, though other almost equally important differences exist.

1. Pig or Cast Iron, . .	Carbon above 1·5, usually with considerable quantities of other impurities.
2. Wrought Iron, . .	Carbon below ·5, nearly free from other impurities. Has not been fused, and, therefore, contains intermixed slag and oxides.
3. Steel, ...	Carbon between ·5 and 1·5, usually almost free from other impurities. May or may not have been fused.
4. Mild Steel (Ingot Iron),	Carbon below ·2, nearly free from other impurities. Has been fused, and is, therefore, free from slag and oxides.

Any one of these forms can be prepared directly from the ore or from any other; but the almost universal method is first to prepare pig iron, and then to convert it into the form required.

The preparation of pig iron will, therefore, be first considered.

Properties of Pig Iron.—Pig iron is the most impure form of iron, often containing as much as 8 per cent. of impurities. The quantity of carbon varies from 2 or a little under, up to 4·5, usually being between 3 and 4, the highest recorded being about 4·6. The carbon is present in two forms:—

(*a*) *Graphitic.*—This consists of flakes of graphite which are only mechanically mixed with the iron, having separated during solidification, and which are left in the insoluble residue when the iron is dissolved in dilute acids.

(*b*) *Combined.*—This is in chemical combination with the iron and is not left in the residue on solution.

Pig irons are classified into several grades, according to the appearance of the fractured surface when a pig is broken, the properties of the different grades being very different. The grading varies a little in different districts, but usually there are six classes, Nos. 1, 2, 3, 4, mottled, and white. Nos. 1, 2, 3, 4 are often called foundry pig, and the two last forge pig. No. 4 is sometimes subdivided into No. 4 foundry and No. 4 forge. Nos. 1, 2, 3, 4 are also spoken of as gray irons. This classification depends on the appearance of the fracture only, and the line of demarcation between the grades is somewhat uncertain. It is quite independent of the presence or absence of deleterious impurities, so that there may be, and in fact are, several qualities of each grade.

No. 1 Pig has a dark gray or black colour, and a very highly crystalline fracture, large shining flakes of graphite being very distinct. When melted it makes a very fluid liquid, flows freely and so fills the most minute interstices of the mould. It makes sound castings, which have a low density and are weak or only moderately tough. As it is poured from the ladle there are neither sparks nor flashes, and as it cools a thick scum or kish collects on the surface. It usually, though not always, contains a high percentage of carbon, most of which is invariably in the form of graphite. The silica is usually high (about 2 per cent.) and the sulphur is low.

No. 2 Foundry Pig.—This has a closer grain than No. 1, is harder, stronger, and denser, splashes a little as it is poured from the ladle, and throws up less kish as it solidifies. This is the iron most generally used for foundry work, No. 1 being added if greater fluidity is required.

No. 3 Foundry Pig.—This is close grained, the facets of graphite are small and almost indistinguishable, the metal is harder, more compact and brittle than the lower numbers. It does not melt to such a fluid liquid as No. 1 or 2, it throws off abundance of sparks as it flows from the ladle, and throws up no kish as it solidifies. It is used for large and heavy castings.

No. 4 Foundry Pig.—This is still finer grained, harder, flows less readily, and is only used for the heaviest and roughest castings, and in No. 4 forge pig the grain is finer still.

White Iron.—This stands at the other end of the series from No. 1. It is white in colour, close grained, showing a fine granular fracture with no trace of graphite visible. It melts readily, but forms a thick pasty liquid, which does not flow freely and which, therefore, cannot be used for casting; and it throws off bright sparks when the liquid is poured from a ladle. It is very hard, not easily filed, and takes a fine polish. It is usually, though not always, low in carbon, and almost the whole of this is in the combined form, there being very little graphite. The silicon is usually low and the sulphur is often high.

Mottled Pig.—This consists of a matrix of white iron with specks of gray disseminated through it.

Chemical Composition of Pig Iron.—It is quite impossible to give a chemical classification of pig or to decide what chemical composition corresponds to any particular grade. No. 1 contains a very small quantity of combined carbon, usually about ·2, all the rest being in the form of graphite, whilst in white iron there is very little graphite, nearly all the carbon being combined. The system of grading by inspection is so uncertain that it is of little value, and from the grade as so determined it is impossible to deduce any chemical composition. The only satisfactory system of grading would be one based on the percentage of combined carbon and silicon. The following analyses are merely illustrations of the variations in the composition of pig iron:—

	No. 1 Coltness.	No. 1 U. S.	No. 2 Bessemer.	Clarence No. 3.	Mottled.	White.
Graphitic carbon,	3·30	3·43	2·57	3·36	3·02	·87
Combined carbon,	·2	·56	1·17	·65	1·43	2·46
Silicon, . .	3·50	2·15	1·76	1·79	·92	1·12
Phosphorus, .	·98	·36	·038	1·80	·04	·91
Sulphur, . .	·02	trace.	·016	·1	·04	·25
Manganese, .	1·58	...	1·13	·56	2·02	2·72

The lower the number the more valuable is the pig.

Other Varieties of Pig Iron.—*Bessemer or Hæmatite Pig* is pig iron made from hæmatite or other phosphorus free ores. It may be of any grade, but is usually No. 1 or 2; it contains very little phosphorus or sulphur and is usually high in silicon.

Basic Pig is a variety of pig iron used in the basic Bessemer process of steel making. It must be low in silicon, containing 1 per cent. or less, and is, therefore, often white. It must contain a large quantity of phosphorus, usually as much as 2 or 3 per cent.

Cinder Pig is made from tap cinder from puddling furnaces, and similar materials; it often contains a large quantity of phosphorus, and very frequently is "blown," *i.e.*, full of small holes or cavities.

Many other varieties of pig are in use known by the name of the district or works where they are made.

Change of Grade.—Iron can under some circumstances be changed from one grade to another. If a gray iron not containing too much silicon be melted and suddenly cooled the graphite will not have time to separate as the metal solidifies and the iron will become white. Advantage is taken of this in what is called chill casting. Long continued heating to a temperature short of fusion will convert white iron into gray.

Influence of Impurities on Pig Iron.—*Silicon* is always present and has considerable influence on the quality of the iron. Iron will take up almost an unlimited quantity under suitable conditions, but when it reaches 10 per cent. the metal becomes extremely brittle and shows a light lustrous fracture, whence it is called "glazed" pig. A small quantity of silicon is advantageous for most purposes. Where tensile strength is required about 1·8 per cent. is best, whilst for a soft easily workable iron up to 2·5 per cent. is allowable, anything over 3 per cent., however, being dangerous. An important property of silicon seems to be the power of throwing carbon into the graphitic condition and thereby making the iron gray; hence No. 1 pig, which is grayest, is rich in silicon, whilst white iron is poor. If white iron be melted and a little silicon-iron be added it becomes gray; on the other hand, removal of silicon from gray iron by oxidation, as in remelting, tends to make it white.

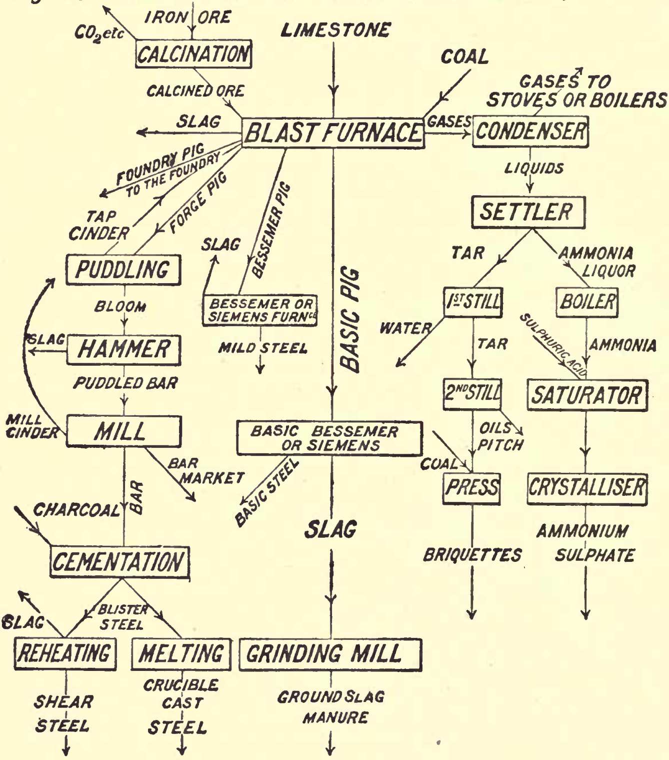
Phosphorus is always present. If a pig is to be used for making Bessemer or Siemens steel the quantity present must be small. Ordinary Scotch pig contains up to 1 per cent.; this increases the fluidity of the metal and improves it for foundry purposes. Large quantities tend to produce blow holes.

Sulphur is usually present in very small quantities (under ·05 per cent.) in gray pig, but may reach ·25 or higher in white pig; indeed, it is sometimes thought to be the cause of whiteness. A good foundry pig should not contain more than ·1.

Manganese and other impurities are of little importance.

Alloys of Pig Iron.—Pig irons containing large quantities of other metals are made, and some of them are of considerable importance.

Fig. 27.—DIAGRAM OF THE IRON AND STEEL INDUSTRY



Spiegel-eisen is a pig iron containing 20 to 25 per cent. of manganese. It has a yellowish colour and breaks with a highly crystalline fracture, showing very large bright facets with a brilliant lustre; whence the name, which means "mirror-iron."

Ferro-manganese.—This is a similar alloy containing up to

80 per cent. of manganese; it has a yellowish colour, a granular fracture, and is very brittle; such alloys may contain up to 7 per cent. of carbon, and the silicon may be very high.

Ferro-silicon or *Silicon-eisen* is a pig iron containing up to 15 per cent. of silicon; it is white and hard.

Silicon Spiegel-eisen and *Silicon Ferro-manganese* are alloys of iron, silicon, and manganese.

Ferro-chrome is a pig iron containing a good deal of chromium.

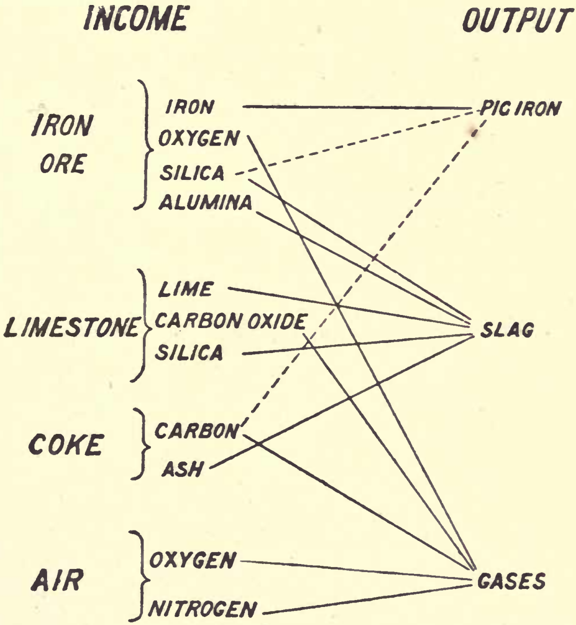


Fig. 28.—Income and Output of Furnace.

Materials used in the Manufacture of Pig Iron.—Iron ores have been already described. They are broken to a convenient size, and all except the oxide ores are calcined in heaps or in kilns. The calcination expels volatile matters, so that whatever was the original condition of the iron it enters the furnace as oxide and also makes the ore more porous. Limestone is required as a flux. It should be as free as possible from silicon and must not be fossiliferous, or it will contain phosphorus, which is objectionable except in the manufacture of basic pig. Occasionally the limestone is burnt to lime before use, but this does not seem to be any advantage. Coke is usually used as a fuel, and it must be hard and dense. In some districts raw (*i.e.*, uncoked) coal is used; it is always splint or hard non-caking coal.

Air is blown in.

Arrangement of the Furnace.—The furnace used is always a blast furnace; the top is now always worked closed, and the gases are drawn off by a pipe and used for firing the boilers. The hearth is usually closed; the number of tuyeres varies from three in very small furnaces to about eight in the larger ones; these may be arranged in pairs, or equally spaced all round, the latter being the preferable arrangement. The blast is heated, and is delivered under a pressure varying with the nature of the fuel and ore and the size of furnace, usually being about 3½ lbs. with coke, up to 7 lbs. with anthracite, or down to 1 lb. with charcoal.

Chemical Changes in the Blast Furnace.—The changes in the blast furnace are comparatively simple. The air blown in from the tuyeres coming in contact with the hot coke, its oxygen is at once converted into carbon monoxide, $C + O = CO$; any carbon dioxide formed at first being at once decomposed. As there will be some moisture in the air it also will be decomposed, and a little hydrogen will thus be added to the gas. The hot gas, which consists mainly of carbon monoxide and nitrogen, is rapidly cooled, as it ascends, by the descending cold charge, and, if the furnace be sufficiently high, will escape at a temperature of about 600° C. As this gas comes in contact with the oxide of iron near the top of the furnace, the carbon monoxide acts on it and reduces the iron, $Fe_2O_3 + 3CO = 2Fe + 3CO_2$, the carbon dioxide being added to the gas and the iron being left in a spongy condition. If the furnace be sufficiently tall and the descent of the charge sufficiently slow, the whole of the iron will be reduced by carbon monoxide in about the first 10 feet of its descent; should any oxide escape reduction it will afterwards be reduced by the solid carbon of the fuel, $Fe_2O_3 + 3C = 2Fe + 3CO$, but this is wasteful, since each 56 lbs. of iron so reduced uses up 18 lbs. of carbon, which would otherwise be burned and evolve heat in the furnace. As the charge descends the limestone is decomposed, $CaCO_3 = CaO + CO_2$, more carbon dioxide being thus given to the gases, and as the temperature rises the lime combines with the silica and alumina present to form the slag. The iron also combines with some carbon to form cast iron, which then melts; some silicon is reduced from the silica and passes into the iron; nearly the whole of the phosphorus, a small part of the sulphur, and a good deal of the manganese in the charge are also reduced and pass into the iron, the remainder passing into the slag. The alkalies in the charge acted on by the carbon and the nitrogen of the air yield cyanides—*e.g.*, $K_2CO_3 + 4C + 2N = 2KCN + 3CO$, which pass up with the gases, but being condensed before the top is reached, descend again and are re-volatilised.

Products of the Blast Furnace.—Pig iron has been already described (see p. 92).

Slag.—The slag is made up of the gangue of the ore, the ash of the fuel, and the fixed portions of the fluxes added. In practice it is always a double silicate of lime and alumina, though both oxides may be to some extent replaced by those of other metals, such as magnesia, manganese, &c., according to the nature of the gangue of the ore. In a coke furnace the slag will usually be of the monosilicate type, approaching to the formula, $3(2CaO,SiO_2) + 2Al_2O_33SiO_2$ = lime, 37 per cent.; alumina, 23 per cent.; silica, 40 per cent.; whilst from a charcoal furnace it will approach more nearly to the bisilicate type.

The nature of the slag will vary much with the conditions of furnace working; indeed, change in the slags is often the first indication that the furnace is out of order. If the furnace be working cold from deficiency of fuel or otherwise, or if the quantity of lime be insufficient, a black fusible slag containing silicate of iron, called "scouring cinder" may be produced. A blue slag is usually produced by the presence of manganese; a yellow one by sulphide of manganese; a very stony slag indicates excess of lime, whilst an opalescent one points to alumina being in excess. As a rule, the more siliceous the slag the higher must be the temperature of working.

Uses of Slag.—Slag is not of much use; it is mainly used for road metal, and ballast for railway lines. It has been suggested to mould it into blocks for building. Some basic slags can be made into a good cement, and slags blown into wool by a jet of steam or air have been used to some extent for non-conducting linings. Some coloured slags have been used for ornamental purposes.

Dust.—A great deal of dust is carried over with the gases, and is collected in dust boxes or in the flues; it is of no use.

Gases.—The gases escaping from the furnace are combustible. They consist of nitrogen, carbon monoxide, carbon dioxide, a small quantity of hydrogen, a good deal of steam, and, where coal is used as fuel, coal gas, tarry matters, and a little ammonia. The water condenses in the flues, and the gas is used for firing boilers, and for heating the blast. The calorific power of the gas is low, owing to the quantity of carbon dioxide present (see

p. 42), which, when the furnace is working properly, will be about one-half the quantity of the carbon monoxide. If the quantity of carbon dioxide falls much below this it shows that carbon is being uselessly consumed in the furnace.

out

Tar and Ammonia.—When, as in Scotland, coal is used in place of coke for fuel, the gases contain enough tarry matters and ammonia to be worth recovery. Ordinary splint coal contains about 1·4 per cent. of nitrogen, of which about one-fifth escapes with the gases in the form of ammonia, the remainder going off as nitrogen. This is equivalent to about 26 lbs. of crystallised ammonium sulphate for each ton of coal consumed; a small quantity, but one which becomes important when the enormous quantity of coal consumed in an iron work is taken into account; one work in Lanarkshire, for instance, uses about 3,000 tons of coal per week, which yields over 30 tons of ammonium sulphate, in addition to 120,000 gallons of tar.

Many forms of apparatus have been suggested for the recovery of these products. As an example, a brief description is given of the Alexander & M'Cosh process, the first patented and erected. The principle on which it is based is that of condensation by cooling and washing with water, and the plant used is very similar to that used in a gas work, except that it is on a much larger scale, necessitated by the fact that each ton of coal yields about 125,000 cubic feet of gas, so that for the quantity above mentioned 375,000,000 cubic feet of gas would have to be treated per week. The gas passes from the furnaces by a large gas main and thence to the condensing apparatus.

The condensing apparatus consists of three parts—(1) the atmospheric condenser; (2) the water condenser; (3) the scrubber.

The gas is first led into the atmospheric condenser, which consists of 200 iron pipes, each 2½ feet in diameter and 40 feet high, arranged in twenty rows of ten pipes each. These are alternately connected at the top by short cross pipes, and they stand on a water box with divisions between each pair of tubes, so that the gas has to traverse twenty of them in its passage. The gas enters this condenser at about 350° F., and is considerably cooled in its passage through it, and leaves it at about 120° F.; a little tar is condensed.

The water condenser is a large iron chamber, 40 feet high, 40 feet long, and 20 feet wide, divided into a series of chambers by iron partitions, stopping short alternately of the bottom and top, so that the gas passes up one division, down the next, and so on, till it has traversed them all. These chambers are crossed by a large number of iron pipes (2,000 are used) about 3 inches in diameter; these are joined outside by bends, first at one end and then at the other, so that cold water delivered at the top flows backwards and forwards across the chamber till it reaches the bottom and flows away. In this condenser the gas is further cooled to 80° F. and deposits more of its tar.

The scrubber through which the gas next passes is a square tower, 80 feet high, and 25 feet square; it is traversed by a large number of perforated shelves placed in an inclined position and terminating at the lower edges, a little short of the wall of the scrubber. As the gases ascend this they are met by a descending stream of water, by which the tar and ammonia are washed out.

These condensers offer so much resistance to the passage of the gas that the furnace blast is not sufficient to carry it through; a Roots' blower or a fan is, therefore, used which draws the gas through the apparatus and delivers it to the furnaces for combustion. As this might draw air into the mains while the furnaces are being charged, a signal is always given from the furnace top, and

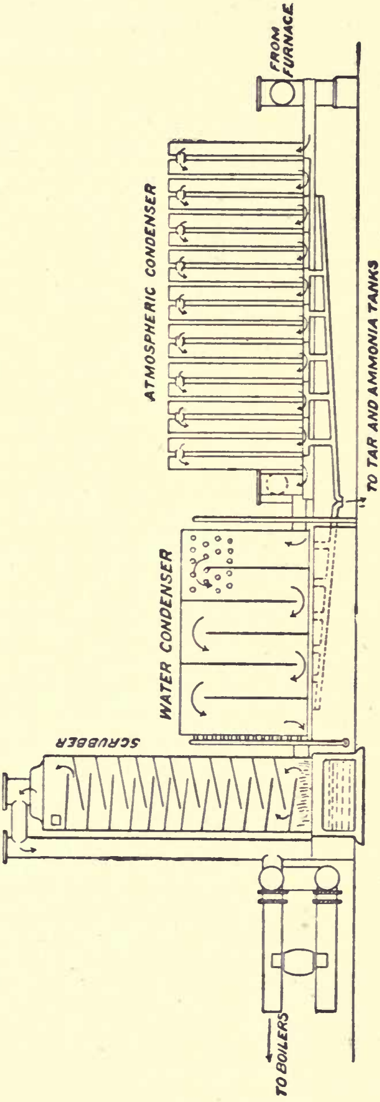


Fig. 29.—Ammonia and Tar Recovery Plant.

the suction is reduced before the bell is lowered. The gas is of excellent quality; its value as a fuel is not seriously diminished; and, being cool, it does not carry very much water with it.

The tar and ammonia liquors are run into a large tank and allowed to separate; the tar rising to the top. The ammonia liquor is pumped into boilers (a little lime being added to decompose any ammonium salts that may be present) the ammonia is boiled off, and passed into sulphuric acid till the acid is saturated; the liquid is then evaporated by steam and the crystals are fished out and dried, preferably in centrifugal driers, and are ready for sale. The yield is about 25 lbs. of ammonium sulphate per ton of coal consumed.

The tar is very different from that obtained in the manufacture of coal gas owing to the very low temperature at which it is produced. It is heated in a still to drive off water, and is then distilled. The oils obtained are sold for lighting, lubricating, and other purposes, and the pitch is used for making briquettes. The yield of tar is about 16 gallons of anhydrous tar per ton of coal consumed.

The Best Form and Size of Blast Furnace.—The modern blast furnace differs considerably from the older forms. It is much larger, being from 60 to 100 feet in height, and with a diameter at the bosh of 16 to 25 feet. The ratio of the height to the diameter is about 4 to 1. The diameter at the top of the charge is about $\frac{2}{3}$, and at the hearth about $\frac{1}{3}$ that of the bosh, though these proportions are subject to considerable variation. The interior contour is a continuous curve without either sharp edges or re-entering angles. The hearth is almost always closed, the tuyeres being evenly spaced, so that there is one just over the taphole, and the blast is used as hot as possible.

The best height for the furnace varies with circumstances. It must not be so high that the weight of the charge above will crush the fuel or ore into powder. Therefore, the harder and stronger the materials the taller can the furnace be. A furnace working, for instance, on hard Durham coke can be used of greater height than one working on raw coal. In the former case 80 feet or over being satisfactory, whilst in the latter about 60 feet is the maximum. The furnace must not be so high that the charge will prevent the free passage of the blast—*i.e.*, the more porous the ore and fuel, and the larger the fragments in which it is used, the taller may be the furnace.

The furnace must be so high that the gases leave at a temperature not much above 600° F., or there will be undue loss of heat; and that the ore remains so long in contact with the gases that the oxide of iron is almost entirely reduced by carbon monoxide before it becomes hot enough to be reduced by carbon, or there will be a waste of fuel. Therefore, the less porous the ore; and the more rapidly the charge is made to descend, the taller should be the furnace. It must be borne in mind that increase in the height of the furnace entails considerably increased cost in construction, and also in raising the materials for charging.

The diameter of the furnace at the hearth must not be so great that the blast cannot penetrate to the centre The form of the furnace must be such that the charge shall descend and the gases ascend uniformly, and great care must be taken in arranging the charging not to use a method by which the larger lumps of ore and fuel are thrown to the side, as this will facilitate the creeping of the gases up the furnace wall, and leave the middle of the furnace without its proper supply.

6 **Rate of Working, &c.**—The working burden of the furnace depends on the ratio of the fuel to the other parts of the charge; with a heavy burden the proportion of fuel is low, with a light burden it is high. The amount of material passed through the furnace or the rate of driving depends on the amount of air blown in. Other things being equal, the quicker the charge is made to descend, provided it be not too quick, the better. The average time in Great Britain is about forty-eight hours; in America with quick driving it is twenty-four hours or less.

The lighter the burden and the hotter the blast the hotter will be the furnace, and the grayer will the iron, and the more siliceous will the slags, tend to be. Hot working tends to the reduction of silicon, and, therefore, to the production of gray irons. In the manufacture of ferro-manganese, silicon-eisen, and similar materials, a very high temperature is required, and the furnaces are, therefore, worked with a light burden and a very hot blast. A heavy burden and a cool blast tend to the production of non-siliceous white iron and basic slags.

Increase of temperature can be attained by increasing either the temperature of the blast or the amount of fuel, but sufficient fuel must always be used to give the necessary quantity of reducing gases.

The Hot Blast.—The introduction of the hot blast led to an enormous saving in fuel. There are several reasons for this. When cold air is blown into a furnace it becomes heated, and, expanding, absorbs a large quantity of heat, thus tending to cool the furnace where the highest temperature is required. A greatly larger quantity of fuel being required to supply the heat, with the cold blast more air must be blown in, and, therefore, the gas current is more rapid; it is more difficult to cool the gases, and to allow the carbon monoxide to exert its full reducing power. The heat of the blast is available immediately the air enters, whilst the cold air will probably spread considerably through the charge before combustion is complete. Thus the hot blast tends to give a much higher temperature at the zone of fusion, and this is what is required for good working. The amount of saving produced by the use of the hot blast depends very much on the conditions of working.

Out

Blowing in, &c.—The mass of masonry of which the furnace is composed must be heated very slowly, and therefore the lighting up of a new furnace or "blowing in" takes some time; a fire is lighted, fuel is gradually added, then fuel and ore, and lastly the blast is turned on, so that it may be some weeks before the furnace is in full working order. Blowing out also takes a considerable time. When any obstruction takes place in the furnace, it is often indicated by the furnace working too cool, and running white iron. This is often due to the accumulation of a mass of material which fails to descend, and forms a "scaffold." Such obstructions are often very difficult to remove.

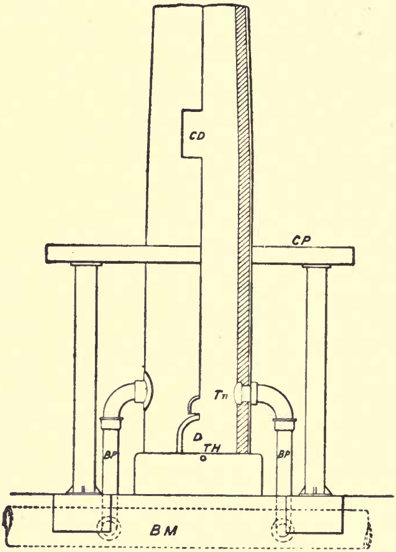


Fig. 30 —Foundry Cupola.

B M, Blast main.
B P, Blast pipe.
Tn, Tuyere.
C D, Charging door.
D, Door.
T H, Tap hole.

Out

The Foundry.—Pig iron is largely used for making castings. For this purpose the lower numbers of "foundry" pig are used, the number varying according to the nature of the work and the taste of the founder. As a rule, a mixture of several brands is used, the idea being that in this way better results can be attained.

The Cupola.—This is used for melting the iron, and is simply a small blast furnace, usually cylindrical and cased with iron, the ratio of height to diameter being from 5 : 1 to 6 : 1, and usually built to melt from 5 to 10 tons of iron. The blast is supplied through one or more tuyeres; in large cupolas several are used, often placed in two rows, one above the other. As the temperature will not be very high, and the cupola is worked intermittently, there will always be some unfused material left; this is raked out through the door, D, which is bricked up while the furnace is at work, or, in some modern cupolas, is got out through a "drop bottom." The consumption of fuel is from 100 to 200 lbs. of coke per ton of iron melted, depending much on the conditions of working. The fuel, iron, and a little limestone for flux are introduced through the charging door, and the fused metal is tapped from the tap hole, either into a ladle or direct to the moulds. The blast is supplied by a fan or Roots' blower at a pressure of 1 to 2 inches of mercury.

Moulds, &c.—The moulds are made in what is technically called "sand," which is a mixture of moulding sand with from $\frac{1}{15}$ to $\frac{1}{8}$ its weight of coal dust, and they are used either just as they are made, in which case they are said to be of "green sand," or of "dry sand," after artificial drying. The moulding sand is not a pure siliceous sand, but always contains a small amount of alumina or other binding material. A model or pattern of the article to be cast is prepared in wood; from this the mould is made in "sand" contained, for small work, in iron boxes or flasks. The mould is usually made in two or more pieces which are separated by a layer of "parting sand." Openings or "gates" for the admission of the molten metal and holes for the escape of the air are made, and the molten metal is run in. When cool the mould is taken off, the surface of the casting is cleaned, and projecting pieces from the gates, &c., are removed.

In "loam" moulding, which is used for large work, no pattern is used, but the mould is built up with loam, a very binding sand, mixed with water till it is plastic, the surfaces being finished to the required shape by means of "striking boards" or "templates."

CHAPTER X.

MALLEABLE IRON.

Malleable or Wrought Iron.—This is undoubtedly the form of iron with which man was first acquainted. Of all the forms in which iron comes into the market, it is the one which approaches most nearly to purity. It contains not more than ·2 per cent. of carbon and very small quantities of other impurities, so that not unfrequently the iron is as high as 99·7 per cent.

Wrought iron has a gray colour, and breaks with a granular or crystalline fracture, unless the piece has been subjected to rolling, in which case the structure becomes very fibrous, as rolling tends to induce a fibrous structure, whilst vibration tends to destroy this and make the metal crystalline. It is not hardened by being heated to redness and quenched in cold water. It is very easily magnetised, but does not retain its magnetism. It does not melt at any temperature which can be attained in ordinary furnaces, but passes into a soft condition in which it can be welded. Its specific gravity varies from 7·3 to 7·9. It does not oxidise in dry air, but in moist air in presence of carbon dioxide it rusts readily. When heated to redness in air it oxidises and becomes covered with a black layer of oxide called "hammer scale," which, however, is not adherent, and which has the approximate composition Fe_3O_4.

When heated in an oxidising atmosphere, wrought iron becomes brittle, unworkable, and unweldable, and is said to be "burnt." This burning is probably due to the formation of oxide of iron in the mass of the metal.

Wrought iron readily takes up carbon when heated with it, passing first into steel, then into cast iron. It is soluble in dilute acids, but strong nitric acid renders it passive, probably owing to the formation of a thin layer of black oxide on the surface.

Defects in Malleable Iron.—There are two principal defects in malleable iron, due to the presence of small quantities of impurities. When the metal is brittle at ordinary temperatures so that it cannot be worked cold, it is said to be **cold short.** This defect is usually due to the presence of phosphorus or excess of silicon, though other less common elements have a similar effect. If the metal be brittle at high temperatures so that it cannot be worked red hot or welded, it is said to be **hot, or red short.** This defect is usually due to the presence of sulphur, ·03 of which will render the metal unforgeable. A piece of metal may have these defects either singly or together.

Tensile Strength.—Since wrought iron is principally used for purposes where it is required to resist great strain its strength is of very great importance. The tensile strength is from 17 to 26 tons, with an elongation up to 25 per cent. on an 8-inch test piece and a contraction of area before fracture of from 20 to 50 per cent. Owing to the fibrous structure of rolled iron the strength is considerably greater along the fibre than across it.

out

Forms of Malleable Iron.—Malleable iron is rolled into various forms "or sections," such as bars, rounds, squares, or flats, usually from 14 to 18 feet long, angles, tees, channels, and various other forms, rods for wire drawing (about 6 B.W.G.). Sheets (under $\frac{3}{16}$ inch), plates (over $\frac{3}{16}$ inch), are made of the required sizes up to the capacity of the mill where they are made, the largest plates readily obtainable being about 30·0 feet long and 6 feet 6 inches wide, though a few works produce up to 10 feet wide. The qualities of the iron, apart from the value attached to the productions of special works, are—

Common,	branded,	👑	
Better,	,,	👑	Best.
Superior,	,,	👑	Best Best.
Best of all,	,,	👑	Treble Best.

Preparation of Wrought Iron.—Wrought iron may be prepared directly from the ore, indeed this is the way in which it was first obtained; or it may be made from pig iron, which is the method usually adopted. Taking the composition of pig iron and wrought iron to be approximately—

	Pig.	Bar.
Carbon,	3·5	·2
Silicon,	2·0	·1
Phosphorus,	·5	·04
Sulphur,	·2	·02
Manganese,	2·0	Trace.

It will be seen that the preparation of wrought iron from pig will be a refining, or removal of impurities, and this can be done by oxidation.

Puddling.—The process of conversion usually adopted is called

Start

puddling, and was invented in 1784 by Henry Cort. The furnace used is a small reverberatory furnace, having a tall chimney so as to produce a good draught, and a fireplace very large in proportion to the hearth area ($\frac{1}{2}$ to $\frac{1}{3}$); it has only one working door, placed at the side. The hearth is made of iron plates so arranged as to allow a free circulation of air underneath; on this is spread a layer of hæmatite, tap cinder, or hammer scale, which is heated till it is just pasty; over this is spread the fettling or working bottom, which is made of some material con-

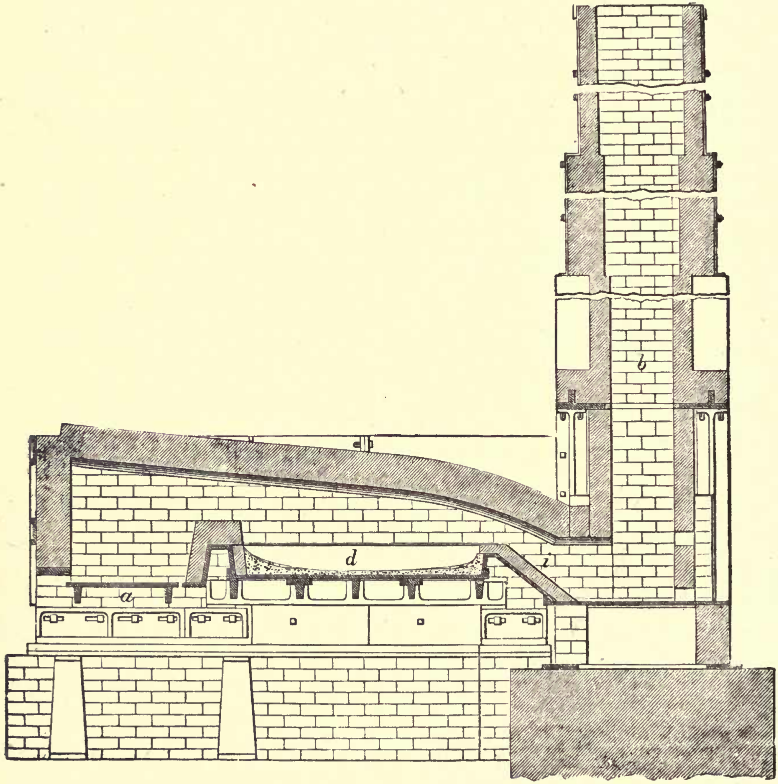


Fig. 31.—Puddling Furnace.

a, Fireplace; *d*, Heart of furnace; *i*, Flue slope; *b*, Chimney.

taining a good deal of oxide of iron. Hæmatite, magnetite, ilmenite, burnt ore or blue billy, and other materials have been used, but the most general is the mixture of oxide of iron and silica, known as bull dog. This is obtained by roasting tap cinder, usually in heaps. Tap cinder, which is obtained in the puddling process, is a fusible black basic silicate of iron, $x\text{FeO}, \text{SiO}_2$. On roasting, this takes up oxygen, forming a mixture of ferric oxide and silica, $2(x\text{FeO}, \text{SiO}_2) + x\text{O} = x\text{Fe}_2\text{O}_3 + 2\text{SiO}_2$. As both these are difficultly fusible and do not combine, the mixture is very refractory unless the atmosphere becomes powerfully reducing, when oxygen would be taken from the ferric oxide, and ferrous silicate would be re-formed. The furnace being fettled and hot, a charge of pig iron is introduced, the door closed, the fire made up and the charge melted down. The door is now opened and the damper slightly lowered so as to reduce the temperature. Air has access through the open door, oxidation begins, slag is formed, and the workman stirs this into the molten iron, from time to time adding hammer scale or mill cinder to increase the action if required. At first the liquid is tranquil; after a time carbon monoxide is formed and given off, burning as it comes to the surface with a blue flame, the evolution of gas soon becomes so rapid that the liquid seems to boil, from which the process is often called "pig boiling." As the evolution of carbon monoxide slackens, the metal becomes thicker and thicker, till at last it is pasty. The slag is now run off, and the temperature raised to the highest possible point. The slag or cinder which is left becomes very liquid and the workman continuously mixes this up with the metal, which soon becomes quite thick and passes into a spongy mass of malleable iron or "comes to nature." The mass of iron thus produced is soft and incoherent, the granules of metal being separated by slag. The puddler breaks it up into pieces or balls of a size suitable for handling (60 to 80 lbs.); these are lifted out with tongs and carried at once to the hammers to be consolidated into masses of malleable iron.

Theory of the Process.—The process may be divided into four stages—(1) melting down; (2) formation of slag by oxidation; (3) boiling; (4) balling up.

During the first stage there is little chemical action, except, perhaps, a little surface oxidation. During the second stage the air acting on the iron oxidises it, forming (say) magnetic oxide, Fe_3O_4, mixed with a considerable quantity of other oxides; $3\text{Fe} + 4\text{O} = \text{Fe}_3\text{O}_4$. When this oxide is stirred into the mass it is at once attacked by the silicon in the iron forming a silicate of iron, which, being fusible, rises to the surface as slag. The reactions which take place are no doubt complex, but the following may be taken as indicating the nature of the changes :—

$$Fe_3O_4 + Si = 2FeO, SiO_2 + Fe$$
$$Fe_3O_4 + 2Si = 3Fe + 2SiO_2$$
$$2FeO + SiO_2 = 2FeO, SiO_2, \&c.$$

As the silicon is removed, the oxide of iron begins to act on the carbon, $Fe_3O_4 + 4C = 3Fe + 4CO$. The ultimate result of these reactions is that the silicon is oxidised to silica, which forms a fusible slag with oxide of iron; and the carbon is given off as carbon monoxide, the evolution of which causes the boil. It must be remembered that the oxidation is due to atmospheric oxygen, not directly but indirectly, for it goes on even when the surface of the metal is covered with a layer of molten slag. The ferrous silicate which forms the slag is easily reduced, and is capable of taking up larger quantities of silicon or of losing iron without becoming infusible—thus

$$2(2FeO, SiO_2) + Si = 2FeO, 3SiO_2 + 2Fe$$
$$2FeO, SiO_2 + C = FeO, SiO_2 + CO + Fe.$$

The silicon is completely removed, and the carbon is reduced to about ·1 per cent. It is usually said that the silicon goes first, then the carbon, but this is not strictly correct; the oxidation of the two takes place simultaneously, but during the earlier stages the silicon is removed much more rapidly than the carbon. The manganese is almost completely removed during the early stages; the phosphorus is reduced by about 80 per cent., a good deal being squeezed out as a fusible phosphide during "shingling;" the sulphur is only removed to a small extent.

The cinder produced must be fusible, hence the flame must not be too oxidising or "cutting," or too much ferric oxide will be formed; this being infusible at the temperature of the furnace will remain mixed with the slag making it "dry;" it will also be mixed up with the iron and will not be squeezed out during "shingling." If the quantity of silicon in the iron is not enough to form a sufficient quantity of fusible cinder, the slag will contain ferric oxide and be infusible. If the quantity of silica be too large, the slag will be too siliceous and infusible or cold, and before it comes right there will be undue oxidation of the iron; the process will also be prolonged, and the iron may come to nature (this depending only on the removal of carbon) before the silicon is all removed. A pig for puddling should contain a considerable amount of carbon, at least 1·5 per cent. of silicon, little phosphorus, sulphur, or manganese, and it should be gray, so as to yield a fluid mass when melted.

As a result of the process, puddled bloom and tap cinder, a basic ferrous silicate often containing a considerable quantity of ferric oxide, are produced.— end

Charges, &c.—The charge of pig used varies from 3 cwts. (Bowling) to 4½ cwts. (Staffordshire). The number of heats worked in twelve hours varies from nine (Bowling) to four or five (Scotland). The consumption of coal is from 20 to 26 cwts. per ton of bar made, and the loss of weight in conversion from 7 to 15 per cent. In general the more siliceous the pig, the longer will the process take, the more coal will be consumed, and the greater will be the loss of metal.

Modifications of the Process.—In the process as invented by Cort the charge was worked on a sand bottom, and white iron was used; the charge never became very fluid, but remained more or less pasty, so that the oxide of iron had to be worked into it as the process went on. This form of the process was called "dry puddling."

Gas furnaces are largely used, and various arrangements have been suggested for mechanically rabbling the charge. In the Dank's process the body of the furnace is made cylindrical, and is kept in rotation; this keeps the metal well mixed with the oxide and hastens oxidation; this apparatus has been very favourably reported on, and is said to give good results, but it is not in use in this country.

The Refinery.—At one time puddling was preceded by a process of refining, the object of which was to remove most of the silicon and part of the carbon, and thus convert the gray iron into white; this is, however, now generally abandoned.

The finery is simply a rectangular hearth, the bottom being of sandstone and the sides formed of cast-iron water blocks; at the front is a dam plate; above the hearth is a stack carried on columns, the space between which is closed by iron plates with openings for the passage of the tuyeres; in front of the hearth is a casting bed made of flat iron plates with flanges at the edge, and kept cool by the circulation of water beneath. Two or three tuyeres are used, placed on each side, and so arranged as not to be directly opposite one another.

The metal may either be run into the finery directly from the blast furnace or it may be added in the solid form, the latter plan being the more usual. Some broken sandstone is put into the hearth, coke added, lighted, and the blast turned on; the charge of about 2 tons of metal is then added. In two to two and a-half hours it will be melted down; more coke is added, as required, to keep up the temperature. The blast of air being directed downwards on to the coke burns it, but as the layer of coke is thin a good deal of air penetrates through it, and coming in contact with the metal oxidises it. A basic, and, therefore, powerfully oxidising, slag is produced, which removes the silicon almost completely and the carbon partially. When it is judged that the operation is complete, the metal is tapped out. As it flows down the casting plate it spreads into a thin sheet, and the cooling is hastened by throwing water on the surface. When cold, the cake of white iron is removed and broken up. The loss of weight is about 2 to 5 per cent., being greatest with siliceous irons, and the consumption of coke is about 2 cwts. per ton of iron.

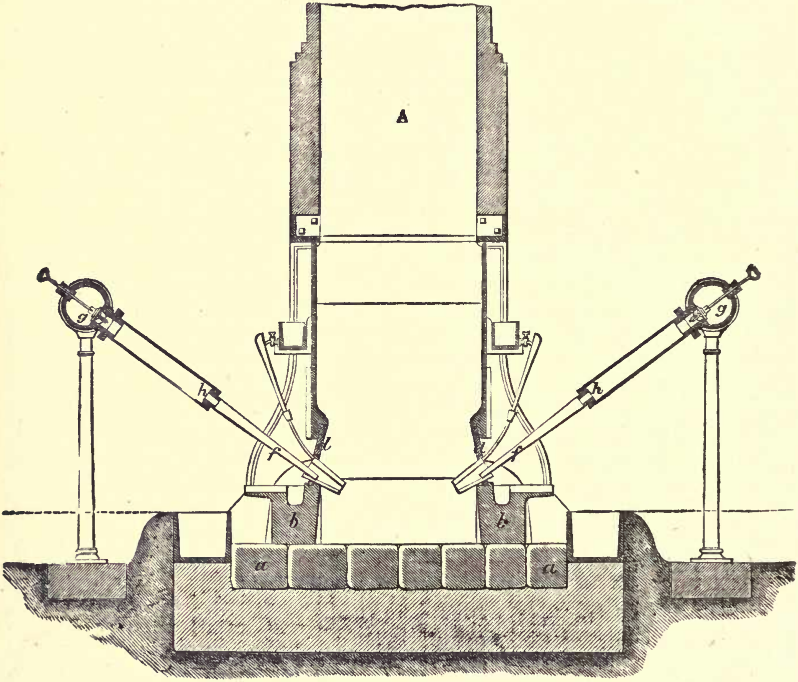


Fig. 32.—Refinery.

A, Shaft. *b*, Side blocks. *f*, Tuyeres. *g*, Air joints.

out **Production of Wrought Iron in Open Fires.**—The finery was used for the production of wrought iron before the invention of puddling, and continued in use in South Wales till wrought iron was replaced by mild steel in the manufacture to tin plates. The process consisted of two operations.

1. *Running in the Coke Finery or Running-out Fire.*—The coke finery was a hearth about 18 inches square, similar to the refinery described above. In it the iron from the blast furnace was refined.

2. *The Charcoal Finery.*—Attached to each coke finery were two small rectangular hearths called charcoal fineries, into which the metal was tapped from the coke finery, the slag being kept back as completely as possible, and what did pass being solidified and removed. Charcoal was then thrown on the surface of the metal, damped with water to prevent it being blown away, and the blast turned on. The charcoal ignited and soon fused the iron; more charcoal was added from time to time as required to keep up the temperature. The blast playing downward oxidised the iron, and the oxide formed in its turn oxidised the silicon and carbon; the mass was kept well stirred till it became pasty; then it was broken up, the lumps brought to the surface to allow of further oxidation, then put back, and so on till the conversion into malleable iron was complete. The Lancashire hearths and many others were identical in principle, differing only in details of working.

Production of Malleable Iron directly from the Ore.—This has been carried on from the very earliest ages, and many modern methods have been suggested for making it applicable to present conditions. The ore, usually hæmatite, is reduced by carbon, and the temperature is kept so low that the carbon does not combine with the iron. In all the process, however, there is great danger that some of the carbon may so combine and produce a steely iron.

Catalan Forge.—This form of furnace, which is still in use in the Pyrenees, has been described on p. 60, and will serve as a type of similar furnaces. About $9\frac{1}{2}$ cwts. of ore (45 per cent. iron) will yield about 3 cwts. of bar with the consumption of $10\frac{3}{4}$ cwts. of charcoal. The reactions are almost identical with those of the blast furnace; the slags are very basic silicates of iron, so that there is a considerable loss of iron.

The American Bloomery.—This is a more modern application of the same principle, and is still in use in the United States. The hearth is rectangular, and made of thick cast-iron plates. There is a single D-shaped tuyere, and the blast is heated to 550°–800° F., by circulating over the top of the hearth. The hearth is filled with burning charcoal and pulverised ore and charcoal are thrown in at short intervals. The slag is tapped intermittently and the metal collects into a pasty mass. Ater three or four hours the mass of iron is pressed together and brought towards the tuyeres in order that it may be more strongly heated; it is then removed and taken to the hammer. Eight charges are worked in twenty-four hours. Each blow yields 300 to 400 lbs. of iron, equivalent to 80 per cent. of the iron in the ore.

✓ **The Stückofen or High Bloomery.**—This furnace marks a step in the passage from the shallow hearth to the blast furnace. It was a shaft furnace, 10 to 16 feet high and 3 or 4 feet in diameter, with one tuyere. In place of a tap hole a drawing door was used, which was loosely bricked up when the furnace was at work. Ignited charcoal was put in, then charcoal and ore, and the blast was turned on, more charcoal and more ore being added till the furnace was full; and then it was kept full till enough ore had been added. The slag was tapped off through a hole left in the door and the iron collected into spongy masses at the bottom. The furnace was allowed to burn down, the drawing door was taken down, and the spongy mass of iron was drawn out and taken to the hammer. The interior of the furnace was cleaned and was then ready for a fresh charge. From a furnace of this type the passage to the blast furnace was very easy, for as soon as the furnace was made so large or the blast so powerful that the temperature was high enough for the iron to combine with the carbon, pig iron would be produced.

Out **The Chenot Process.**—This process was used some years ago for the manufacture of iron sponge, and in a modified form is still in use for the same purpose. The ore is powdered, mixed with charcoal, and heated in vertical retorts, the lowest part of the retort being below the fireplace, and being provided with a water-jacket in which the sponge can cool before being withdrawn. Blair's process, used in America, is similar, the retorts being heated by gas.

✓ **The Siemens Direct Process.**—This can only be used with rich ores. The furnace is a rotating cylinder, with a very refractory lining, the casing being kept cool by the circulation of water in pipes. The furnace is fired by gas, which enters and leaves at the same end. The charge of ore and charcoal with sufficient limestone is introduced, and the furnace is set in rotation; after about three and a-half hours the slag is tapped out, and the temperature is raised as high as possible. The iron is reduced, but the excess of carbon is too small to allow of the production of cast iron, so the metal is left in a spongy form, and the charge is balled up and withdrawn. The loss of iron in the slags is about 20 per cent.; and the fuel used is about 200 lbs. for each 100 lbs. of iron obtained. About 5 heats can be worked in a day.

✓**Shingling.**—The wrought iron obtained, either by puddling or by a direct process, is in the form of a spongy incoherent mass quite unfit for use, and which has to go through several more processes before it is ready for the market. The first is shingling, by which, either under the hammer or in squeezers, the spongy mass is consolidated and any liquid slag is squeezed out.

0 ✓**Tilt Hammer.**—This is the simplest form of hammer. The beam is supported in the middle, the head being at one end and the working power at the other. These hammers are always of small size, as the strain on the axis is very great, and were mainly used in connection with the older direct processes.

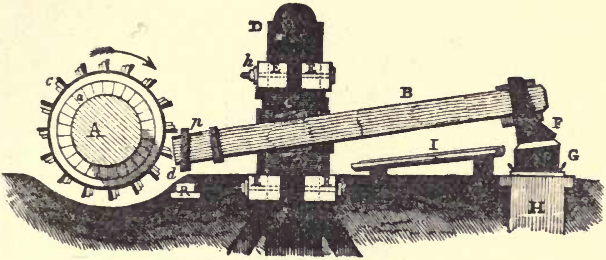


Fig. 33.—Tilt Hammer.

A, Cam shaft. B, Hammer shaft. F, Hammer head. G, Anvil.

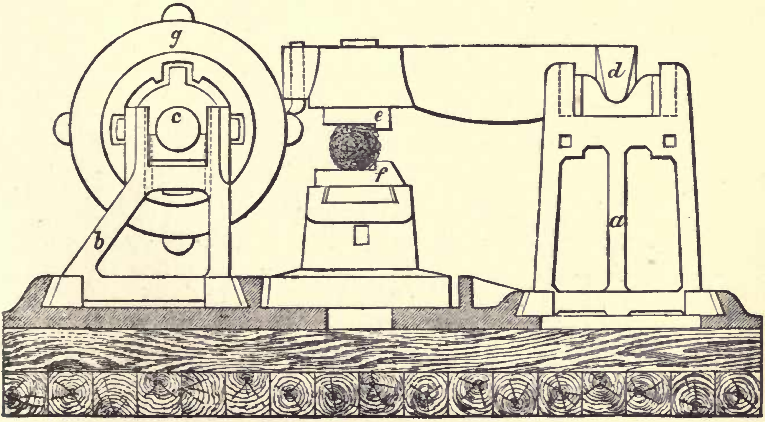


Fig. 34.—Helve Hammer.

d, T knife edge. *e*, Hammer head. *g*, Cam wheel. *f*, Anvil.

out

Helve Hammer.—In this form the fulcrum is at one end, which is usually much extended so as to give a very broad bearing, and the head is near the other end. The head of the hammer is lifted by a cam acting either on a piece at the end of the shaft (*Nose-helve*), or on a projection between the fulcrum and the head (*Belly-helve*). These hammers often have a moving mass weighing up to 10 tons, a fall of 18 inches, and are driven up to 60 blows a minute; they are very efficient, and at one time were in general use.

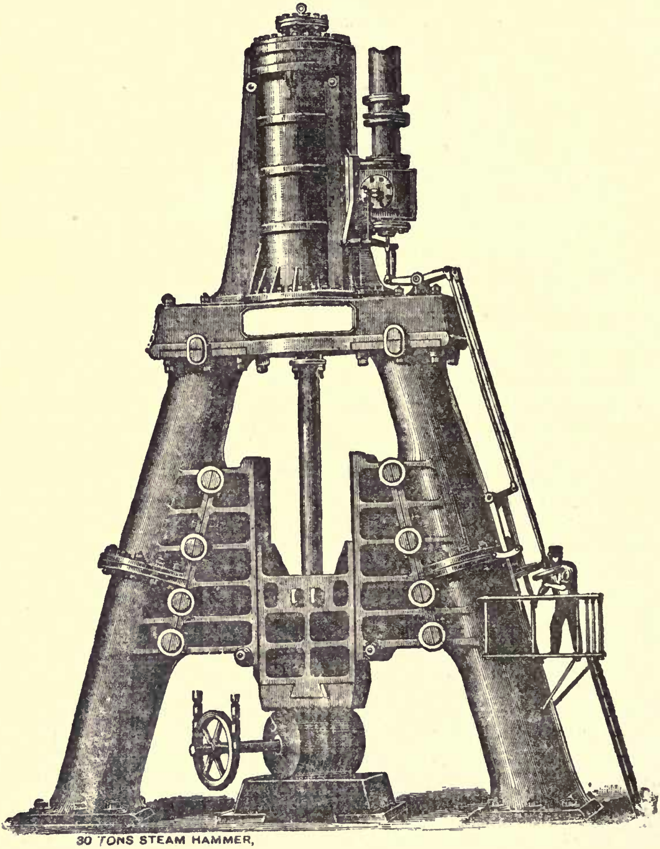


Fig. 35.—Thirty-Ton Steam Hammer, by Messrs. Thwaites Bros., Bradford.

ot

0 **Steam Hammers.**—These are now almost universal, a heavy hammer head is lifted by the action of steam on a piston working in a cylinder, and either allowed to fall by its own weight (single-acting hammers), or forced down by steam (double-acting hammer). The hammer head works in guides; it may weigh anything from a few hundredweight up to many tons, and the fall may vary from 1 up to 8 or 10 feet. The anvil must be very massive and the framework strong. With a steam hammer the force of the blow is perfectly under control.

0 ot **Squeezers.**—In these the compression is produced by steady pressure, not by impact; they are but little used, except for working the very heavy blooms obtained in the Danks puddling furnace.

✓ **Effect of Shingling.**—The primary effect of shingling is to compress the spongy "bloom," squeeze out any liquid slag, and weld it into a solid mass. The removal of the slag can never be complete, and, therefore, wrought iron always contains more or less slag and oxide disseminated through it, and the less fusible the slag the more is likely to be retained. A considerable quantity of liquid iron-phosphide is squeezed out during shingling.

The relative advantages of hammers and squeezers can hardly be discussed here; for light work the former seem to have an advantage, but for very heavy blooms the inertia of the upper portions may take up so much of the energy of the blow that the pressure in the centre may be very small. With a squeezer the pressure is likely to be much more uniform. the

0 ot **The Forge Train.**—From ~~the hammer or~~ squeezer the bloom is passed to the forge train. This consists of two sets of rolls, the roughing and the finishing rolls. The rolls are 3 feet to 5 feet 6 inches long and 18 inches to 22 inches in diameter, and are provided with grooves through which the bloom is passed. The shingled bloom is in the form of a roughly square prism, and the first grooves are of such form, usually gothic, as to grip this and carry it through; they are also roughened so as not to let the metal slip. The rest of the grooves are square. The finishing rolls are very similar, and the grooves are so arranged that the last one will produce a bar 3 inches to 7 inches wide and $\frac{3}{4}$ to $1\frac{1}{2}$ inches thick. The rolls are carried in massive bearings; the lower roll is driven by power, the upper one by the friction of the metal. The metal is passed through the groove and is returned over the top roll, back through the next groove, and so on. The metal is now puddled bar.

0 ot **The Mill.**—The puddled bar is now passed on to the mill for further treatment. This consists of three stages—(1) cutting up

or shearing; (2) heating in the mill furnace; (3) rolling in the mill.

Shearing.—The bar is cut up into lengths; these are piled up into a pile, tied up with iron wire, and passed into the mill furnace.

out **The Mill Furnace.**—This is a reverberatory furnace of small size, the bed is of sand resting on cast-iron bed plates, and slopes down towards the base of the chimney. The furnace being hot the pile of metal is placed on a flat iron paddle (peeler) and introduced into the furnace, the piles being placed so that the bars are parallel to the fire bridge and sprinkled with sand. The door is closed and luted with a shovelful of coal, the fire is made up, and in about half an hour the piles reach welding heat; they are then withdrawn and taken to the rolls.

The iron always oxidises to some extent, and the oxide combining with the sand forms a very fusible slag which flows down to the foot of the flue and is called flue- or mill-furnace cinder. It is a very basic silicate of iron, often containing as much as 40 to 45 per cent. of iron. Gas furnaces are now very generally used for re-heating.

out **The Mill Rolls.**—There are two sets of rolls—the roughing or billeting rolls and the finishing rolls—together constituting the mill-train.

The billeting rolls are about 6 feet 6 inches long and 22 inches in diameter. They do not run in continuous collars, but rest on bearers which are capable of adjustment by screws; the lower roll is driven from the upper one by friction.

The finishing rolls are somewhat shorter and smaller. The rolls are grooved, the first grooves being square, and the last of the form into which the bar is to be rolled. The rolls are provided with stripping plates; these are curved plates of metal resting in the grooves of the rolls so as to prevent the bar adhering and wrapping round the roll.

out **Three High Rolls.**—Ordinarily the bar is passed through the first groove, the end is then lifted with tongs and it is passed back over the upper roll, and then through the second groove. Except for light work, a rising table is arranged to lift the bar so that it may be more easily returned. Very often three high rolls are used; in these there are three rolls instead of two, the middle one being driven. The bar is passed through the first groove between the lower and middle rolls and back through the second groove between the middle and upper rolls, so that it is rolled while travelling in both directions.

In very heavy work the rolls are reversed between each passage of the bar.

out **Merchant Bar.**—The bar thus produced is called merchant bar, and it is the common form of malleable iron; if branded at all it is simply marked with a crown. If the process of cutting up, re-heating, and rolling be repeated, best iron is obtained; another repetition gives best best; and a third gives treble best. The rolling does not alter the chemical composition of the metal; but simply makes it more homogeneous by more evenly distributing the fragments of slag and other impurities that are always mixed with it, and also develops a fibrous structure.

Plates and Sheets.—These are rolled in rolls 4 to 9 feet long and 20 inches to 36 inches in diameter, the finishing rolls being of chilled and the roughing rolls of ordinary cast iron. They have no grooves, and the distance between them is adjusted by screws; usually the rolls are reversed for each passage of the plate, so that it is rolled in both directions of its travel, the rolls being brought a little closer together for each passage. The top roll is usually a trifle larger than the bottom one so as to deflect the plate a little downwards, and thus prevent wrapping. The extension is almost all in the direction at right angles to the rolls; the billet is therefore passed through until its length is equal to the width of the rolls, then it is reversed and the rolling is completed.

out **Forging Press.**—Masses of wrought iron are sometimes forged by pressure instead of by impact, and hydraulic pressure is usually adopted. As there is no fear of fracture, the anvil and pressure block may be made of any required form.

Black Plates.—These are the plates which were, until recently, made for tinning. The iron made in the charcoal finery (see p. 113) was hammered into flat plates, and these were broken up into stamps each weighing about 28 lbs., which were placed on the flattened end of a staff, heated to welding temperature and hammered into a solid mass, nicked and doubled over, so that the surfaces which formed the bottom of the flat paddle formed the top and the bottom; the whole was then again heated, welded into a solid mass, and rolled. For heating, the **hollow fire** was used; this was simply a deep rectangular hearth with an arched top, and holes in the sides closed by sliding doors. The article to be heated was placed on a paddle and held through one of the openings in the flame above the fuel in the hearth, and, therefore, was not in contact with the fuel.

✓ **Welding.**—One of the most valuable properties of malleable iron is that of softening at high temperatures so that masses can be welded together. In welding, two surfaces, which must be absolutely clean, are brought into contact and pressed together till they unite. To ensure the absence of oxide from the surfaces in contact, a flux is always used. The usual flux is sand, which combines with the oxide forming a very fusible slag which is squeezed out when pressure is applied. Borax is sometimes used, as it is a good solvent for oxide of iron. Dissimilar irons are often troublesome to weld, as their welding temperatures may be very different.

out

Electric Welding.—In this process the metals to be united are brought to the welding temperature by means of electricity, and thus all contact with fuel is avoided, and there is less danger of the surfaces becoming contaminated. In the **Thomson** process the two pieces to be welded are placed in contact, in the position in which they are to be united, and a very powerful current, sufficient to heat the metals to the welding temperature, is sent across the junction, and at the same time gentle pressure is applied. By this process, rods, wires, or tubes can be welded end on, but it is only suitable for small articles. In the **Bernardos** process the surfaces to be united are heated by the electric arc. The negative pole of a powerful dynamo is attached to the metal, and the positive is attached to a carbon pencil which is brought over the spot to be heated, the current passes and the heat is sufficient to bring the metal to welding heat. Plates are readily welded by this process; the edges are feathered and the space between is filled with turnings which are melted by the heat of the arc. Defects in mild steel castings may also be removed by cutting them out, filling the cavity with steel turnings and melting them in. The heat and light are so very intense that, for the protection of the face of the worker, the work must be viewed through red glass.

CHAPTER XI.

STEEL.

Steel.—It is not at all an easy matter to give a definition of steel, as the term is used for two entirely different classes of material, "ordinary steel" and "mild steel," which have very little in common. In this chapter ordinary steel only will be considered.

Steel for our present purpose may be defined as an iron containing between ·5 and 1·5 per cent. of carbon, with very small quantities of other impurities, and which, on being heated to redness and cooled in water, becomes hard.

Steel has a bluish-gray colour, breaks with very finely crystalline or granular fracture, has a specific gravity of 7·62 to 7·97, being lower in the hardened than in the soft state. Wire drawing and cold rolling also tend to diminish its specific gravity. It is malleable, forgeable, and weldable; though these properties are much better shown in specimens with low than in those with high carbon. It is much more difficult to magnetise, but retains the magnetism much better than wrought iron; that is, it has greater coercive force. On exposure to the air it oxidises, but less readily than wrought iron.

Hardening and Tempering.—One of the properties of steel on which its value largely depends is that of being hardened by sudden cooling; and of being subsequently tempered by gentle heating and cooling. If a piece of steel be heated to redness and then cooled rapidly, it becomes hard, the hardness depending on the nature of the steel and on the rate of cooling. For hardening, different cooling liquids are used. Mercury, if used in large quantity, produces the greatest hardness; water, which is the general hardening agent, next; and oil the least.

Steel, if at all high in carbon, when hardened is too brittle for use; it must, therefore, be tempered or have its hardness reduced. This is done by gentle heating and then cooling; whether suddenly or slowly makes no difference. The higher the temperature to which the metal is heated in tempering, the more of the hardness will be removed and the softer and tougher will be the resulting metal; if the temperature be raised sufficiently high all the hardness may be removed, the metal being annealed. When steel is heated in air it becomes covered with a film of oxide, which, owing to its extreme thinness, is transparent, and by the interference of the reflected rays of light appears coloured. The colour of the film enables the workman to judge of the temperature to which the steel is heated. A tempering scale is given as a rough indication of the nature of the colours, but no description could enable any one to hit exactly the right point in actual practice; it is one of the things which only experience can teach.

TEMPERING SCALE.

OXIDE TINT.	TEMPERATURE.		USES
	Centigrade.	Fahrenheit.	
Pale yellow, .	220	428	Lancets.
Straw, . .	230	446	Surgical instruments, razors.
Full yellow, .	243	469	Penknives, cold chisels for steel.
Brown, . .	255	491	Cold chisels, scissors, hatchets.
Brown with purple spots, . .	265	509	Axes, planes.
Purple, . .	277	531	Table knives, shears, &c.
Blue, . .	290	555	Swords, coiled springs, augers.
Dark blue, .	316	600	Hand and pit saws, cold chisels for wrought iron.
Red just visible in the dark.	...	...	Clockmakers' purposes, softest.

It must be remembered that the lower the tempering temperature the less hardness will be removed, and, therefore, the harder and more brittle will be the metal; also, that for the high tempers steels rich in carbon must always be used.

Theory of Hardening and Tempering.—A great deal of study has been devoted to this subject, and much has been done towards clearing up the great difficulties with which it was surrounded. Combined carbon can exist in iron in two forms—(1) hardening carbon, in which form it seems to be uniformly diffused through the metal, and, therefore, combined with all of it; and (2) carbide carbon, in which it is combined with a portion of the iron to form a more or less definite carbide (Fe_3C) which remains mixed with the excess of iron.

At high temperatures, say above redness, the carbon in steel seems to be all present in the "hardening form." If the metal be allowed to cool slowly, the carbon passes into the condition of carbide carbon and the steel is soft; if, however, it be cooled rapidly the carbon remains in the hardening form and the steel is hard. If now the steel be moderately heated, a certain play of the molecules is allowed, part of the carbon passes into the carbide condition, and the steel is partially softened.

This does not seem to explain all the facts, and there is some evidence to show that iron may exist in two allotropic modifications—α, a soft iron, and β, a hard iron—the β being the stable form at high temperatures, and existing at low temperatures if the metal has been suddenly cooled, or in presence of certain other elements; the α form being stable at low temperatures.

Both facts are probably of importance, and have influence on the phenomena of hardening and tempering.

Methods of Preparing Steel.—Steel may be made in three ways:—

(1) By carburising malleable iron. wrought Fe.
(2) By decarbonising pig iron.
(3) Direct from the ore.

Method (1) is now almost universally used.

The Cementation Process.—This is the process in general use for the manufacture of tool and other hard steels. Best malleable iron is used, rolled into bars 8 to 15 feet long, 5 or 6 inches wide, and ½ inch thick.

The converting furnace is a rectangular chamber covered with an arched roof and with a fire grate running along the centre. On each side of the grate is placed a box, "chest or pot," made of stone or fire-clay slabs, 8 to 15 feet long, 2 feet 6 inches to 4 feet deep, and about 3 feet wide, resting on fire-bricks so as to allow free circulation of the fire underneath and around it. At the end of the chamber is a fire door, an entrance door or manhole, which is bricked up when the furnace is in operation, and a small opening communicating with the chests by which trial bars can be inserted and withdrawn. When the furnace is ready a layer of charcoal, from which the dust has been sifted, is spread over the bottom of the chest; on this is put a layer of bars; this is covered with charcoal, then another layer of bars, and so on till the chest is full; the top is then covered and made airtight with an impervious coating, the material used in Sheffield being the mixture of finely divided steel and sandstone from the grindstones known as *wheel-swarf*. When both the chests are full the manhole is built up and the fire lighted, and in about forty-eight hours cementation temperature is reached. Trial bars are withdrawn, broken, and examined from time to time, to ascertain how the operation is progressing. In from seven to nine days the operation will be complete; the fire is then drawn, and a few days allowed for cooling; after which the manhole is unbricked, and the workmen enter and discharge the chests.

Blister Steel.—The bars when withdrawn are found to have undergone several changes. They have increased in weight by about 1 per cent., and the surfaces, which before were smooth, are now covered with blisters from the size of a pea up to an inch or more in diameter, whence the metal is called *blister steel.* The blisters should be small and evenly distributed. On breaking the bar the appearance of the fracture will vary according to the stage to which conversion has been carried. In an ordinary case there will be a central portion or sap of unaltered malleable iron surrounded by an edge of highly crystalline steel; and the two should merge one into the other without any sharp line of demarcation. In the lowest "heat" the layer of steel will be only an outer skin; in "double shear heat" the sap and steel will be about equal; and in "steel through heat" the sap will have disappeared; whilst in the highest or "melting heat," the steel all through will have become highly crystalline.

If the bars show signs of fusion or are "glazed" on the surface, the temperature has been too high; if air has obtained access the surfaces are rough and the bars break with difficulty, owing to a skin of malleable iron having been formed; they are then said to be "*aired.*"

Theory of the Process.—Two points have to be considered—(1) The transference of the carbon into the iron; and (2) the cause of the blisters:—

(1) As soon as the temperature rises high enough the oxygen in the air mixed with the charcoal will be converted into carbon monoxide. At a red heat iron is very porous to this gas, which, therefore, diffuses itself into the bars, and in contact with the hot iron is dissociated into carbon dioxide and carbon, the latter being fixed by the iron, $2CO = CO_2 + C$. The carbon dioxide then diffuses out, and as it comes in contact with the hot carbon is reconverted into carbon monoxide, and so the process goes on. As soon as a certain amount of carbon has been deposited the carbon dioxide diffusing out tends to combine with the carbon, and so equilibrium is set up, no more carbon being deposited at that point, until, by the partial saturation of the iron beyond it, the amount of carbon dioxide passing is diminished; then the deposition of carbon can again go on, and so on, till, ultimately, if the process were not stopped, the metal would be converted into cast iron.

(2) Wrought iron is never homogeneous, but contains scattered through it, particles of slag and oxide of iron. If a particle of

carbon be deposited in contact with a particle of ferric oxide, or even silicate, the oxide will be reduced, $Fe_2O_3 + 3C = 2Fe + 3CO$; the carbon monoxide liberated will occupy an enormously larger space than the solids from which it was formed, and, therefore, will momentarily exert a very great pressure. If the liberation takes place in the mass of iron the gas must remain under pressure till it can escape by diffusion; but if it be near the surface the thin skin of metal will be unable to resist the pressure, and a blister will be formed. If the iron has been well worked, so that the particles of oxide and slag are broken up and evenly distributed, the blisters will be small and scattered pretty evenly over the surface; if, on the other hand, the iron has not been well worked, the blisters will be fewer and larger. If a metal be used for conversion which contains no slag or oxides, no blisters are produced.

Case Hardening.—This is very closely allied to the method of making blister steel. An article of malleable iron is placed in an iron box or chest surrounded by some carbonaceous material, and is heated to redness for from twelve to twenty-four hours. The surface of the metal becomes converted into steel, and is thus hardened. For case hardening some organic substance containing nitrogen is almost always used in place of charcoal; leather cuttings, hoof parings, and the like being favourite materials. *Mild centered steel* is mild steel which has been case hardened.

Malleable Castings.—As the reverse operation, the manufacture of the so-called malleable castings may be mentioned. An article of cast iron is embedded in an oxidising material, such as hæmatite, and heated for some time. The carbon is gradually removed from the surface, leaving a more or less spongy mass of malleable iron. Such articles are always deficient in strength owing to the porous nature of the metal and the absence of fibre, and are rarely malleable, as, except in the case of very small articles, the conversion cannot be carried through.

Treatment of Blister Steel.—The steel made by cementation is not homogeneous, the carbon not being evenly distributed. The bars are cut up, piled, welded, and rolled, exactly as in the manufacture of bar iron, and then constitute "single shear steel." If the bars be again cut up piled, welded, and rolled into bars, "double shear steel" is produced.

Crucible Cast Steel.—For many purposes shear steel would not be sufficiently homogeneous, and, therefore, the blister steel is frequently melted. This yields crucible cast steel, so called to distinguish it from mild steel which is also melted, but not in crucibles.

The pots or crucibles for the melting are made on the premises by moulding the clay in a flask having the shape of the outside of the crucible, the inside being shaped by a core. They are made about 18 inches high and 7 inches in diameter, and are dried, but not fired, before use. Each crucible lasts for three fusions, different quantities of steel being melted each time, so that the surface of the metal, where the corrosion of the pot is greatest, may come at different levels.

The melting holes or fires are ordinary crucible furnaces, elliptical in form, 26 inches by 19 inches and 30 inches deep, each holding two pots; they are placed below the working floor, so that the pots can be easily handled.

The pots are placed on bricks in the furnace and heated to redness. The steel is carefully selected so as to be of the required quality, broken up into pieces of a convenient size and placed in the pots; after which the fire is made up. In forty-five to fifty-five minutes the first fire will have burnt off, the coke is pushed down and the fire again made up. By the end of the second fire the steel will have begun to melt, and in four or five hours will be completely melted. The temperature is carefully regulated, and the progress of the operation watched, and when the steel is "dead melted," the crucible is lifted out with a pair of tongs, scum removed from the surface, and the metal poured at once into the mould. If the mould is so large that it requires the contents of more than one pot, care must be taken to commence pouring the second pot before the first is emptied, so that the molten steel may run into the mould in a continuous stream.

If the steel has not been long enough melted it will seem "fiery," throwing off sparks as it flows, and will yield honeycombed ingots; if it be left too long in the fire it will pour "dead," and will be brittle and lacking in tensile strength.

Working the Metal.—The ingot having been cast, is hammered or rolled to the size and form required. The very greatest care is necessary, and during re-heating the ingot must be continually turned in order to ensure uniform heating, and its surface must be sprinkled with sand or borax to protect it from oxidation. It must only be left in the fire just long enough to allow it to become properly heated or "soaked" through, as if it be left too long in the fire its quality may be injured.

Steel heated too strongly or too long becomes very brittle, and is said to be "burnt."

Qualities of Steel.—The grade of a steel depends on the quantity of carbon it contains; the quality depends on many conditions, and steel of any grade may be either good or bad. The usual grades or tempers are:—

Razor	temper,	1½	per cent.	carbon.—	A steel capable of taking a very high temper, and yielding very keen-edged cutting tools, but which is very difficult to work, being easily burned in the fire.
Sawfile	,,	1⅜	,,	,,	More easily worked, but still requires to be treated with very great care.
Tool	,,	1¼	,,	,,	The most useful quality for tools; can be welded, but only with great care.
Spindle	,,	1⅛	,,	,,	Useful for large turning tools, taps, &c.
Chisel	,,	1	,,	,,	A strong useful metal, hardens at a low temperature, is capable of taking a keen edge, and in the unhardened state is strong and tough. Used for making cold chisels and similar articles.
Sett	,,	⅞	,,	,,	The unhardened metal is very hard and tough, used for cold sets.
Die	,,	¾	,,	,,	A very useful metal when power to stand great pressure is required.

The two last weld easily.

Wotz.—This is a hard steel prepared by the natives of India by a process similar in principle to cementation. One pound of malleable iron is put into a small crucible with 10 per cent. of the stem and leaves of a species of cassia, the whole being covered with green leaves, and heated in a small charcoal hearth till the metal is fused.

Puddled Steel.—If steel be intermediate between pig and wrought iron, then by stopping the puddling process at the right point steel should be obtainable. This has been attempted, and steel so made is called puddled steel. A furnace is used similar to, but smaller than, that used for puddling iron. The pig, which should be low in silicon and rich in manganese, is charged as usual and melted down; during the early stage the temperature is kept high, but when the boil commences it is reduced by lowering the damper. As the whole of the carbon is not to be burnt out the metal will not "come to nature," but when it is judged that the operation has gone sufficiently far, the temperature is reduced till the metal becomes pasty, when it is balled up, and shingled. These operations require the greatest care and skill, and it is impossible to prepare steel of absolutely uniform quality by this process, as there is nothing to indicate the moment at which to stop the oxidation. Steel is no doubt intermediate between pig or wrought iron as regards its percentage of carbon, but in no other respect; for other impurities should be less in quantity than in either; and it is only with a pig iron free from impurities that it would be possible to obtain a good steel by puddling.

CHAPTER XII.

MILD STEEL.

Mild Steel.—The metal now known as mild steel is iron containing not more than ·5 per cent. carbon, and very small quantities of other elements. In composition it is, therefore, very similar to malleable iron, often containing little, if any, more carbon, but differs from it, in having been fused, and, therefore, being free from intermixed slag and oxide. Whether the absence of intermixed slag or the preparation by fusion should be made the distinguishing characteristic is of little moment, since the two always go together.

Mild steel either does not harden, or hardens only very slightly, when it is heated and quenched in water. It has a tenacity of 26 to 35 tons per square inch, with an elongation up to 30 per cent. on an 8-inch test-piece. The higher the percentage of carbon the greater is the tensile strength and the less the extension before fracture; hence in specifications where toughness is required the maximum as well as the minimum tensile strength is usually mentioned. Chemically, it has nearly the same composition as wrought iron, but is free from intermixed slag and oxide, and, therefore, has no fibre. It is stronger than wrought iron and has for most purposes superseded it, and, when necessary, it can be made softer than any wrought iron.

Effect of Impurities on Mild Steel.—Very small quantities of some impurities have a great effect on the quality of the metal. The higher the *carbon*, the less malleable and weldable it is, also the harder and the greater is its tenacity. As the carbon approaches ·5 per cent. the metal hardens perceptibly on being heated to redness and cooled in water.

The effect of *silicon* is somewhat uncertain; it usually produces hardness and red shortness; it slightly increases the tensile strength, increases the fusibility and fluidity, and decreases the welding power, and in castings diminishes the tendency to form blowholes. It is possible that silicon may exist either as intermixed silica or as combined silicon.

Manganese has a direct and an indirect action; hence its effect is somewhat uncertain. By preventing blowholes it increases the tensile strength and ductility, but it is said to render the metal brittle under shock.

Sulphur makes the metal red short, the red shortness being less as the temperature rises, and it diminishes the welding power.

Phosphorus is very objectionable, as it diminishes the ductility and makes the metal liable to fracture under shock; it is also said to interfere with the forgeability. The influence of phosphorus is worse on mild steel than on wrought iron.

Manufacture of Mild Steel.—Mild steel is always made by the addition of carbon to pig iron from which the carbon and silicon originally present have been oxidised out. There are four processes in use for its manufacture:—

1. The Bessemer process.
2. The Thomas-Gilchrist or basic Bessemer process.
3. The Siemens or open-hearth process.
4. The basic open-hearth process.

THE BESSEMER PROCESS.

The principle of this process, which was patented by Sir Henry Bessemer in 1855, is that if air be blown through molten pig iron, the carbon and silicon present are oxidised together with some iron forming carbon monoxide and a silicate slag, and enough heat is evolved by the oxidation to keep the metal in a liquid condition till the end of the operation. When the metal is completely decarburised, the requisite amount of carbon is added in the form of spiegel-eisen or ferro-manganese.

The operations are—(1) melting the pig and running it into the converter; (2) blowing the metal in the converter; (3) teeming into the ladle, and addition of ferro-manganese or spiegel; (4) casting into moulds.

The Converter.—This consists of a wrought-iron shell lined with a very refractory lining, almost always of ganister, and is usually of the form shown in Fig. 36. It is carried on trunnions, so that it can be turned over to receive and discharge the metal; to one trunnion is fitted the turning gear, the other is hollow and serves for the transmission of the blast to the blastpipe, by which it enters the air-chest placed at the bottom of the converter. The top of the air-chest is a perforated iron guard plate, through the holes in which the tuyeres are passed. These are fire-clay cylinders about 2 feet 6 inches long perforated with holes about $\frac{3}{8}$ inch in diameter; they pass through the bottom lining of the converter and are held in place by buttons, below the guard plate, so that they can be readily removed when worn out.

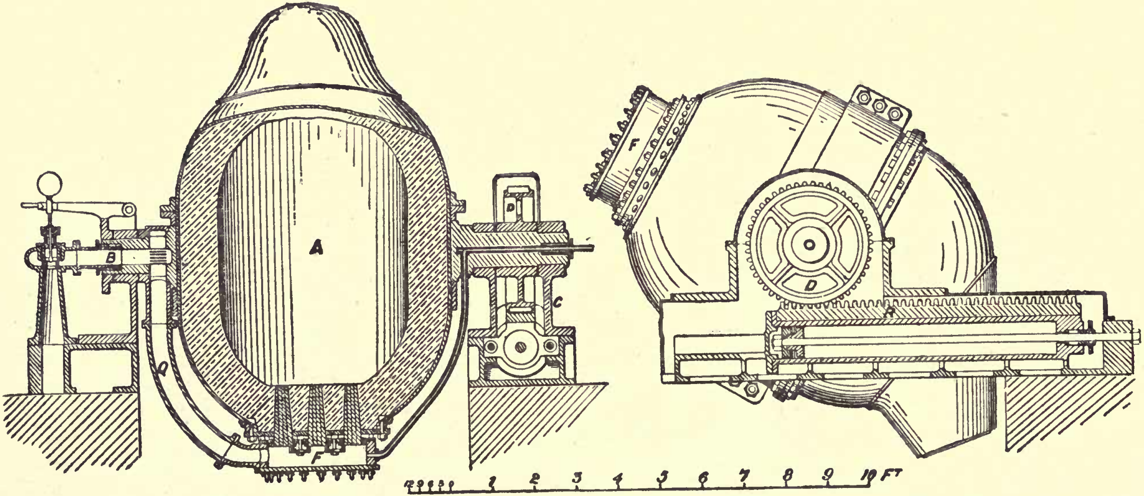


Fig. 36.—Bessemer Converter.

A, Body of converter.
B, Blast pipe.
Q, Blast pipe.
F, Air-chest.
D, Turning pinion.
R, Rack.

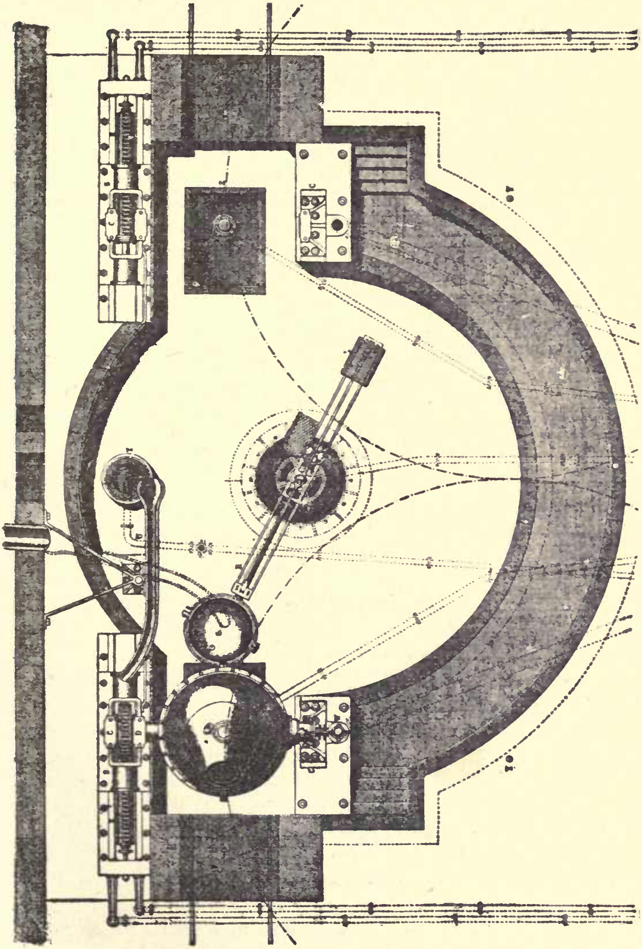


Fig. 37.—Plan of Bessemer Plant.

A, Converter. G, Ladle. C D, Converter tipping gear. H, Crane.

Arrangement of the Plant.—There are usually two converters, and these are placed at opposite sides of the casting pit. In the centre of the pit is the crane, a long platform carrying the ladle at one end, and so arranged that, by rotating it on its axis, and raising or lowering it as required, usually by hydraulic power, the ladle can be brought under the converter or over the moulds. The ladle is of iron, thickly lined with ganister, and is large enough to hold the whole charge of metal. It is provided with a taphole at the bottom, which is closed by a plug operated by a handle outside, so that the steel can run from it into the moulds. Round the casting pit are arranged the moulds, and outside it is a crane for lifting the ingot moulds off the ingots; removing the ingots; and replacing the moulds. Near the converters, usually at a higher level, is placed a cupola of the ordinary type for melting the pig iron. At some convenient place an engine, usually of the direct acting horizontal type, is arranged for supplying air at a pressure of about 25 pounds on the square inch. All motions of the converter, the supply of air, &c., are regulated by the foreman from a platform or "pulpit," from which a good view of the whole plant can be obtained.

The Blow.—The converter having been cleaned out is brought up into position, so that its mouth is horizontal and the charge of 5 to 10 tons of molten pig is run in, either direct from the cupola, or from the cupola into a ladle, and thence into the converter, as may be most convenient. The quantity of metal must be such that it does not reach the level of the tuyeres when the converter is over. As soon as the charge is in, the blast is turned on, and the converter is slowly brought up into the vertical position. As the metal passes over the tuyeres a shower of sparks is ejected, and a yellowish slightly luminous flame appears at the mouth of the converter. The temperature rises and in from four to six minutes the boil begins, the flame increases in brilliancy and size, and sparks of burning iron are ejected. This stage lasts from six to eight minutes; and then the last or fining stage begins, the flame diminishes, the sparks become fewer, and in about fifteen to twenty minutes from the start the flame drops, or becomes much smaller and less luminous, and the blow is over. The converter is immediately turned over on its side, and as soon as the metal is clear of the tuyeres, the blast is stopped, and from 7 to 10 per cent. of spiegel-eisen or ferro-manganese is added solid, or, having been melted in a small cupola is run in from a ladle, and the contents of the converter are poured into the casting ladle. The converter is then turned bottom upwards, cleaned of some very refractory slag, which remains after the metal has been poured out, and turned up ready for another charge. The steel is run from the ladle into the moulds, and the slag is tipped out of the ladle. As soon as the metal has set, the moulds are lifted off and the ingots removed, the moulds being at once replaced for another charge.

Chemistry of the Process.—The air blown in at once becomes heated and attacks the oxidisable constituents, at first mainly silicon and iron, forming a ferrous silicate,

$$2Fe + Si + 4O = 2FeO, SiO_2;$$

the carbon is also attacked, slowly during the first few minutes, but more rapidly as the quantity of silicon decreases. These oxidations evolve a large quantity of heat. The calorific power of silicon is 7,830, that of iron (to FeO) 1,185, and that of carbon (to carbon monoxide) 2,400; and, as the quantity of these substances oxidised is very large, the heat evolved is sufficient to keep the iron in a molten condition even after the impurities are removed, the temperature rising nearly to the melting point of platinum. Sulphur and phosphorus are not removed; indeed, the latter *seems* to increase, because the total quantity of phosphorus remaining the same, the mass of metal through which it is distributed is diminished. When the operation is at an end, practically the whole of the carbon and silicon will have been removed; but the metal will have also taken up some oxygen either as oxide of iron or occluded gas, which renders it brittle and unworkable or "burnt." To remove this, and at the same time to add the quantity of carbon which is required for the steel being made, ferro-manganese or spiegel-eisen, containing the right amount of carbon, is added. The manganese becomes oxidised and passes into the slag, whilst the carbon enters the steel. It is essential that sufficient manganese be added to remove the oxygen, and the smaller the quantity of carbon combined with this the milder can the steel be made. The end of the operation, marked by the drop of the flame, is the point at which the oxidation of the carbon is completed. In this country it is always judged by the eye, but in some Continental works the spectroscope is used, the end being indicated by the disappearance from the spectrum of certain lines and bands, probably due to manganese.

Bessemer Pig.—The pig used must be free from sulphur and phosphorus, as these would remain in the steel. It should be a gray pig, so as to be readily fusible; and must contain about 2½ per cent. of silicon, because on the oxidation of the silicon depends the heat evolution during the early part of the blow, and if there be too little the blow will be cold. Too much silicon is objectionable, because it entails loss of iron; and because, if very high, some may be left after the carbon has been removed, for the drop of the flame depends on the removal of the carbon. The best limits of silicon are between 2 and 3·5 per cent. The higher the temperature at which the pig is run into the converter, and the hotter the converter the less silicon is necessary, and in some American works blows are made with iron containing only 1·5 per cent. or even less of silicon. The quantity of carbon should be slightly in excess of the silicon; manganese is objectionable as it delays the process.

Products.—The products are mild steel and slag; the latter is a silicate of iron containing manganese, and is re-melted for the iron which it contains.

Modifications of Working.—The working of the process may be modified in various ways. The speed at which the plant can be worked will depend on many conditions, but to a large extent on the rapidity with which the ingots can be got away and the moulds replaced. In British practice the time occupied by each blow is from fifteen to twenty minutes, about sixteen heats being blown in eight hours, with two converters. In America the blowing is much quicker; at one work sixty-one heats were blown in eight hours, giving an average of 7·87 minutes per blow. For quick blowing the iron must not be too siliceous.

Forms of Converter.—The converter is usually of the form described—rotating, bottom blowing, eccentric. It is made in three parts, the middle part being attached to the trunnions, whilst the lower and upper parts are removable to facilitate lining; but some modern converters are so arranged that the body can be lifted out of the trunnion ring and quickly replaced by another. The lining is of ganister, and is usually put in by ramming it between the casing and a central wooden core; or, sometimes, ganister, made plastic by mixing with water, is pressed by hand into position, the interior being afterwards smoothed. The lining is examined after each blow, and requires frequent patching. It will need renewal about every five hundred blows, though sometimes it may stand longer. The bottom requires replacing much oftener, probably after each twenty blows or so, and it is so arranged that it can be quickly removed and replaced.

Other forms of converter have been suggested. In some—*e.g.*, the Clapp-Griffiths—the converter is fixed, and the metal is tapped out; the tuyeres are placed horizontally in the sides of the converter (*i.e.*, it is side blown), an ingenious arrangement being used so that the blast can be stopped during tapping without the metal running into the tuyere holes.

Converters of this type are mainly used for blowing small charges, and seem to be more wasteful than the ordinary form.

THE BASIC BESSEMER PROCESS.

In the Bessemer process with ganister lined converters the phosphorus is not removed; and, therefore, only very pure pig iron can be used. The reason for the non-removal of the phosphorus is that the silica of the lining at once decomposes phosphates, forming silicates, and the phosphorus passes into the iron. If a basic lining containing but little silica be used, this does not occur, and the phosphorus is eliminated in the slag. The best material for lining a basic converter is the lime obtained by burning magnesian-limestone (dolomite), a double carbonate of lime and magnesia containing some silica (p. 46). This was suggested by Messrs. Thomas & Gilchrist, and the process in which it is used is, therefore, often called the Thomas-Gilchrist process.

The converter is much the same as that used for the Bessemer process, except that for the same charges it must be larger. It is often made upright, with the mouth parallel to the base (a concentric converter), so that the very pasty slag can be more easily removed.

The Process.—The converter being ready for work, some lime is thrown in, then the pig iron is run in from the cupola or ladle, the blast is started, and the converter is turned up. The blow takes place much as with the Bessemer process, except that more sparks are thrown off and the flame is denser and yellower. When the flame drops the operation is not over ; for, if a sample were taken and analysed, it would be found that whilst all the carbon and silicon were gone the phosphorus had hardly been removed at all. The blow is, therefore, continued for another two to five minutes, this period being called the "after blow." During this stage a good deal of red smoke escapes indicating the oxidation of iron. There is no external indication as to when the after blow is over. When the workman judges that it has gone on long enough, the converter is turned down and the blast stopped, a sample is taken from the converter in a ladle, cooled, hammered flat, and broken, and by its fracture it is judged whether or no the blow has been sufficient. If the fracture be largely crystalline the phosphorus is not all out, but if the crystals be small and close so that the fracture is granular the operation may be said to be over. If necessary the blast is turned on again, and the converter put up for a minute or two, then turned down, and another sample is taken, and so on. As soon as the blow is over, the slag is poured off as completely as possible, then the metal is poured into a ladle, ferromanganese or spiegeleisen added, and the steel is run into the moulds in the usual way.

Chemistry of the Process.—The chemistry of the process is much the same as that of the ordinary Bessemer process. The silicon is removed, more completely than in the acid converter, and so is the carbon. The phosphorus does not commence to go till the silicon and carbon are almost completely removed. Then it goes very rapidly, passing into the slag as a calcium phosphate, probably tetra-calcic phosphate,

$$4CaO, P_2O_5 \quad (Ca_4P_2O_9).$$

Pig Required.—The pig may be white or gray, preferably the latter. The less silicon it contains the better, about 1 per cent. being probably the best quantity. The quantity of phosphorus may be large, anything up to 3 or 4 per cent. Such pig is called "basic pig."

The Products.—These are "basic steel," a mild steel which can be made milder than is possible by the acid process, as the removal of carbon during the blow is more complete; and "basic slag," which contains a large quantity of phosphorus, and which is ground and sold as manure.

The Converter.—This is made in three portions, like the acid converter. It is lined with burnt magnesian limestone mixed with anhydrous tar. The body is usually lined with basic bricks set in a magnesian-lime-tar cement. To line the bottom it is placed over an iron plate fitted with a number of vertical pins, and the magnesia-lime-tar mixture is rammed in hot; when the bottom is lifted the pins are withdrawn, and the holes left serve for the passage of the air, the use of fire-clay tuyere tubes being avoided.

THE SIEMENS OR OPEN HEARTH PROCESS.

In this process the decarburisation is brought about by the addition of ferric oxide, usually in the form of Spanish hæmatite. The theory of the process is identical with that of puddling, the principal differences being (1) that the oxide is supplied by addition of iron ore, not by atmospheric oxidation; and (2) that a sufficiently high temperature is maintained to keep the metal liquid, even after the carbon is all removed.

The Furnace.—This is a reverberatory furnace; always heated with gas, and provided with regenerators; it may be made capable of working anything from 10 to 30 tons of metal, or even more. The working bottom of the furnace is made of very refractory sand, the hearth being made to slope towards a tap hole at the back from which the metal is tapped out. The furnace has three working doors all on one side, and is usually built above the regenerators; and, therefore, at a considerable elevation, which allows the ladle to be run at a suitable level for receiving the metal when it is tapped out, and the working platform at the front must be suitably elevated.

The Process.—The charge will consist (for a 30-ton furnace) of about 20 tons of pig iron, 9 tons of scrap, and 8 tons of hæmatite.

The pig iron is charged on to the hearth, and the scrap is put on it, the charging taking about one and three-quarters hours; in about five hours the charge melts down, a good deal of oxidation taking place during melting, so that some slag is formed. As soon as the metal is completely melted, the addition of the ore is commenced, and this is added in small portions at a time as the process goes on. The boil commences, at first small bubbles appear, then flames of carbon monoxide. In about five hours the boil begins to subside, and when it is over, samples are taken, and the carbon determined by the "Eggertz" test. If the carbon be sufficiently low, the temperature of the furnace is increased as much as possible, the tap hole is opened, and the metal is run into the ladle, 3 or 4 cwts. of ferro-manganese being thrown into it as it flows. A layer of slag forms which protects the surface of the metal from oxidation. As soon as all the metal is out the slag is run into a slag mould, or it may be run into the ladle with the steel and allowed to flow over the top. The quantity of steel obtained is usually a little less than the weight of the metal charged, though under some circumstances, owing to the reduction of the iron ore, it may be a little more.

After the charge is out the furnace requires fettling; this is done by spreading sand in the places where it is required, and hardening it by exposure to a full temperature. The total time, including fettling, required for a 30-ton charge is about fourteen hours.

Attempts have been made to expedite the process by blowing air through the bath, and by the use of rotating furnaces, but these have not become general.

Materials Used.—The pig should be No. 1 or 2, so as to melt freely; the quantity of silicon should not be high, since an excess causes loss of iron and delays the process. Phosphorus and sulphur must be absent, as they are not sensibly reduced. Manganese is of little moment, except that, if present in large quantity, it may delay the process.

The iron ore should be as free as possible from silica, and must be free from sulphur and phosphorus; Spanish hæmatite is usually employed.

Limestone is often added; it also must be free from sulphur and phosphorus.

THE SIEMENS-MARTIN PROCESS.

In this, known as the pig and scrap process, no iron ore is used, the quantity of scrap is much larger than the pig iron, and the purification depends entirely on atmospheric oxidation. It is mainly used in conjunction with other processes for utilising the large quantities of steel scrap which are unavoidably produced in manufacturing operations.

THE BASIC OPEN HEARTH PROCESS.

If a basic lining can ensure the removal of phosphorus in the Bessemer process, the same result should be obtained by using a basic hearth for the Siemens furnace. This has been tried, and the process, though it has not yet come largely into use, has been found to give excellent results. The hearth or working part of the furnace must be of a basic material, such as magnesia-lime, or lime, but owing to the friability of basic-bricks these cannot be used for the sides and roof. As the acid and basic materials would flux one another if they were in contact they must be separated by a layer of neutral material; the material always used being graphite, a layer of which is built in between the basic hearth and the walls.

The process is identically the same as the acid process, except that a large quantity of lime is added to keep the slag as basic as possible, and thus to ensure the removal of the phosphorus.

Materials.—The pig used should be as free as possible from silicon; phosphorus, whilst not essential as in the basic-Bessemer process, is not objectionable.

The **limestone** and the **iron ore** should be as free as possible from silica.

Products.—These are steel and a slag containing a large quantity of phosphates, which is valuable as a manure.

COMPARISON OF METHODS.

These four methods of making mild steel are in general use, and each one has its own advantages and disadvantages.

The Bessemer processes are very quick, and, by the acid processes with a suitable pig, steel of perfectly uniform quality can be prepared. With the basic-Bessemer this is not so easy; the end of the operation can only be ascertained by testing the steel, and the effect of the blow is so extremely rapid that there is great danger of either leaving the metal a little underblown, or overblowing it considerably.

The open hearth processes are slower, but they are much more under control; time can be allowed for testing the metal so as to make sure that it is exactly what is required; with the basic open-hearth process, a steel can be made as mild as is required, milder, in fact, than by any acid process, and pig of almost any quality, provided it does not contain much sulphur, can be worked satisfactorily.

Equally good steel can be made under favourable conditions by all the processes; which will be best in any particular case must depend on the nature of the materials to be had, and of the steel which it is required to make. Where great uniformity of product is required the open hearth processes seem to have the advantage.

Use of Ferro-manganese or Spiegel-eisen.—In all the processes ferro-manganese or spiegel-eisen is added at the end, and this with a twofold object—(1) to remove any oxygen which may have been taken up by the metal; and (2) to supply the requisite amount of carbon. Enough must always be added to fully effect the first object, and, therefore, the less carbon is contained in the steel before addition of the manganese-alloy, and the richer in manganese and poorer in carbon this alloy is, the milder can the resulting steel be made. The manganese-alloy always contains silicon and other impurities, so that the less of it, it is necessary to add the better. Owing to the presence of these impurities it is found impossible to make a good high carbon steel by the addition of manganese-alloy to decarburised iron. In the basic process the action of the manganese-alloy on the slag might cause a reduction of phosphorus, and, therefore, the slag is removed as completely as possible before it is added.

Casting the Mild Steel.—The steel made by any one of the processes is run into a ladle, which is carried on a truck or casting crane. The ladle is an iron vessel capable of holding the whole of the charge. It is lined with some very refractory material, and is provided with a tap hole at the bottom; which is closed by a plug attached to an iron rod, well protected by fire-clay, and which can be raised or lowered by levers from outside. The ladle is brought over the moulds, the plug is lifted, and the metal is run into the moulds.

The moulds are of cast iron, open bottom and top, usually square with rounded corners and slightly tapered, being larger at the bottom, so that they may be lifted off the ingot; they are placed on a flat metal plate, and the bottom is luted with sand. The moulds may be arranged in a ring if a circular casting pit be used, or in a row if a longitudinal pit. In the former case the ladle is carried on a rotating crane; in the latter it is usually carried on a truck running on rails, between which the moulds are placed.

When the mould is filled, it is stopped, by covering the top with an iron plate, or by throwing on sand. As soon as the ingot is sufficiently set the mould is lifted away, and if, as sometimes happens, the ingot sticks it is lifted with the mould, and pushed out by an hydraulic ram.

Soaking Pit.—When the ingot is stripped it will be very hot, the interior will be liquid, and the outside will be much cooler. If the ingot be allowed to cool, it will have to be reheated before it can be rolled. To avoid this it is transferred to a soaking pit. This is a square brick underground chamber large enough to hold one or two ingots, and which is hot, from previous ingots which have been in it. The ingot is placed in, and the top closed; the ingot radiates heat to the walls, and these radiate it back, so that the heat of the ingot becomes gradually distributed, and it becomes equally heated all through; it is then taken to the rolls. During soaking combustible gases are given off so that the atmosphere of the pit is always reducing. It is found difficult to keep up a sufficiently uniform supply of ingots to maintain the pits at a sufficiently high temperature; they are, therefore, now usually supplied with producer gas, and are, in fact, converted into "soaking furnaces."

Rolling the Ingot.—The ingot from the soaking pit or the reheating furnace is passed through the roughing rolls, and rolled down into a billet. Owing to the weight of the ingot (·5 to 5 tons) it cannot be returned over the rolls, and the rolls are, therefore, always made reversing, the engine usually being reversed at each passage. To facilitate moving the heavy ingots, rollers are let into the floor, the tops of which come just above the floor level, and an arrangement is used for turning over the ingot so that it may not always be rolled in the same position. For large work the roughing rolls are not provided with grooves, the squareness of the billet being maintained by turning it over at each passage, the rolls being brought a little nearer together by means of screws. When rolling is complete the ends of the ingot are sheared off, and it is cut into convenient lengths.

The further rolling into rails, sheets, plates, &c., is conducted exactly as for malleable iron, except that the sheets rolled being much larger, heavier plant is required.

Imperfections in Ingots.—When an ingot of hard steel is cast it shrinks on cooling, and leaves a conical depression or pipe at the end; on the other hand, very mild steels usually rise, often scattering abundance of sparks; the upper part of the ingot after solidification, being full of cavities, or "honeycombed." This is due to the evolution of gas; the quantity of evolved gas is very large, in some cases amounting to more than ten times the volume of the steel. A steel which evolves much gas usually contains cavities or "blowholes," and the milder the steel the more likely is this to be the case. The "blowholes" are due to the escape of gas as the metal is solidifying. Many methods have been suggested for their prevention; as, for instance, agitation in the ladle, or otherwise, so as to facilitate the escape of the gas, and casting under great pressure (Whitworth process). Addition of silicon-eisen or ferro-aluminium also promotes soundness.

Cracks due to unequal contraction are also not uncommon in ingots.

Removal of Sulphur.—None of the processes described have much effect on sulphur. A process has recently been invented by Mr. Saniter, of Wigan, by which it seems that this most troublesome element can be completely eliminated. The process may be used either for the desulphurising of cast iron as it comes from the blast furnace, or in the manufacture of steel in the basic open hearth or basic-Bessemer process from sulphury materials.

In purifying pig iron a mixture of calcium chloride and lime, with fluorspar and limestone, is put in the bottom of a ladle and rammed hard, and the molten metal is run on. In the open-hearth furnace a mixture of lime and calcium chloride is added during the process. In either case the sulphur passes into the slag.

Various Steels.—*Tungsten steel* or *Mushet special steel* contains a small quantity of tungsten. It is so hard that it will scratch glass, does not need tempering, and is very fine grained.

Chrome Steel.—The presence of chromium is said to increase tensile strength and hardness; such steels are said to weld readily.

Manganese Steel.—This alloy is very tough, difficultly forgeable, and is also very hard. Hadfield's manganese steel contains 7 to 13 per cent. of manganese. It is very fluid, and solidifies with little contraction and without blowholes.

Silicon steel has been made with 7 per cent. silicon; it is very hard, but welds well.

Aluminium Steel.—The addition of ferro-aluminium to a mild steel which is boiling in the ladle or crucible quiets it at once, and enables small castings to be made of very mild metal. The quantity of aluminium which is present in the resulting castings is very minute. Castings made in this way are called "mitis castings."

Nickel Steel.—This alloy may contain up to 2 per cent. of nickel. With 1 per cent. of nickel it welds readily, but not with more; it is hard and strong, and resists atmospheric corrosion better than any other form of iron.

High Carbon Steel from Mild Steel.—As already stated, this cannot be made directly by addition of ferro-manganese. Mr. Darby has proposed a method by which it can be done, which consists simply in bringing the liquid mild steel in contact with the requisite amount of carbon, in the form of charcoal.

CHAPTER XIII.

COPPER.

Copper.—This metal is next in importance to iron. It has a specific gravity of 8·5 to 8·88 depending on its condition. It melts at 1054°, and on exposure to the air in a molten condition becomes oxidised, the oxide, cuprous oxide (Cu_2O), dissolving to some extent in the fused metal. On solidification such a copper is "dry;" it is extremely brittle, and breaks with a dull brick-red fracture. On exposure to reducing gases most commercial copper becomes brittle or burnt; pure electrotype copper, however, will not "burn." Copper is very tough, and is, therefore, difficult to work with cutting tools. When liquid it absorbs gases readily, and as these are given out on solidification it is very difficult to get castings free from blowholes. It is very malleable and ductile, it can be forged both hot and cold, and is one of the best conductors of heat and electricity; the latter property, however, being very much diminished by the presence of even the smallest quantities of impurities. The tensile strength of cast copper is about 11 tons per square inch; after rolling it may be double this; and wire is stronger still. The finer the wire the greater is its tensile strength.

Copper does not tarnish in dry air; in moist air it becomes covered with a film of basic carbonate commonly, but wrongly, called verdigris. Heated to redness in air it becomes covered with a film of oxide (copper scale) consisting mostly of the red oxide, but containing also some black oxide. It is dissolved by nitric acid in the cold; the other acids only attack it when heated, except in presence of air when even weak acids will slowly convert it into green basic salts.

Commercial Copper.—This is often very impure, almost always containing arsenic, antimony, iron, lead, nickel, cuprous oxide, and frequently tin, bismuth, silver, and other impurities. The most deleterious impurities are arsenic, antimony, and cuprous oxide; the last named of which, when present in any quantity, making the copper dry. The purest form of copper is that refined by electricity, and called "*electrotype copper;*" then comes that known as *best selected*, or, for short, B.S., which should not contain more than 0·5 per cent. of arsenic and a trace of antimony. The varieties known as *cake* or *tough cake* copper may contain a considerable quantity of impurities, the arsenic often reaching 1 per cent. *Chili bar* is an unrefined copper, imported from Chili in bars each weighing about 2 cwts. It contains 1 to 1·5 per cent. of sulphur; some brands being free from deleterious impurities, others containing considerable quantities.

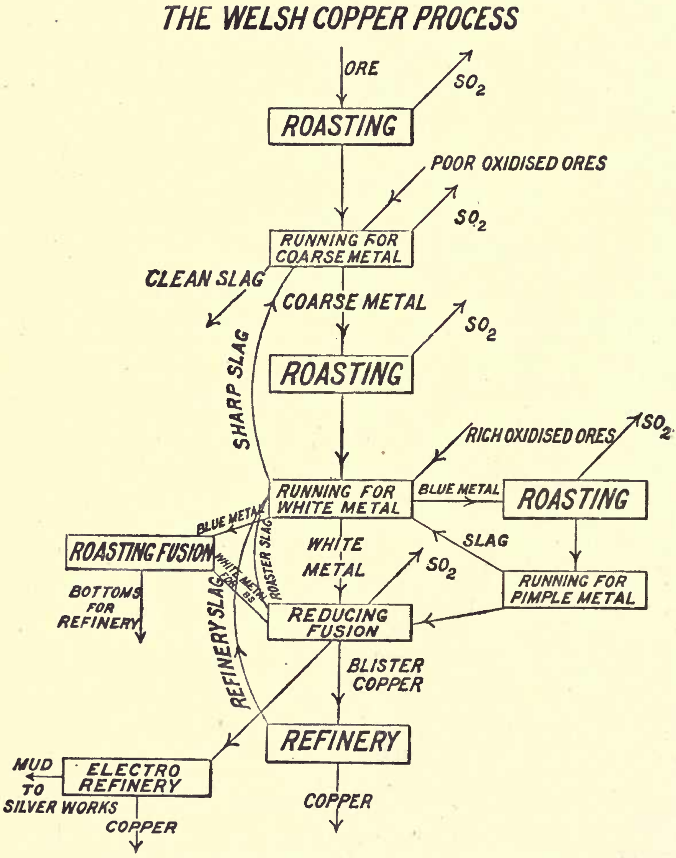

THE WELSH COPPER PROCESS

Fig. 38.—Plan of the Welsh Copper Process.

THE WELSH COPPER PROCESS.

This process, which has grown up in the Swansea district, forms the basis of the smelting methods in use all over the world. The copper ores raised in Cornwall, for working which it was designed, are sulphides, copper pyrites predominating. Owing to the extreme brittleness of these ores it is impossible to dress them; and as they are delivered to the smelter they may contain anything from about 4 to 20 per cent. of copper.

The process is a somewhat elaborate one, and consists of six stages, the object being to gradually concentrate the copper in a regulus, and flux off the earthy impurities and the iron.

The Ore Mixture.—The ore will be so mixed that it contains about 12 per cent. of copper. If more, there is apt to be loss in the slag; if less, the bulk of slag will be too large. The other constituents should be present in such quantities as to yield a fusible, but not too fluid slag. The ore will consist probably of copper pyrites, $CuFeS_2$, or as it may be written Cu_2S, Fe_2S_3, iron pyrites, FeS_2, and silica, with smaller quantities of other substances. It is crushed, and delivered to the smelter in the form of a coarse powder.

Operation 1.—Roasting or Calcination.—About 3 tons of the powdered ore is charged on to the hearth of a reverberatory furnace (Fig. 20) and roasted, with free access of air, for from twelve to twenty-four hours at a low temperature. During the roasting large quantities of sulphur dioxide, some sulphur trioxide, and arsenious acid (anhydride) are evolved. The principal changes which take place are that the copper and iron sulphides are partially converted into oxides, and that the residual iron pyrites loses some of its sulphur.

$$Cu_2S + 3O = Cu_2O + SO_2$$

$$Fe_2S_3 + 9O = Fe_2O_3 + 3SO_2$$

$$2FeS_2 + 2O = Fe_2S_3 + SO_2$$

It must be remembered that in this, as in most other cases, the reactions which actually take place in metallurgical operations are very complex, and that, therefore, the equations which are given can only be taken as types of the changes.

The roasted charge is raked through openings in the hearth into the arched space beneath the furnace.

Operation 2.—Melting for Coarse Metal.—A charge of about 3 tons of the roasted ore is let down on to the hearth of a reverberatory furnace, arranged for melting—*i.e.*, with the hearth sloping in all directions to a well at one side, from which a tap hole communicates with the exterior. The doors are closed, the fire made up, and the charge is run down.

The products of the operation are *coarse metal*, which is tapped out through the tap hole into sand moulds; and *slag*, which is raked out through the door at the end of the furnace.

The coarse metal is a regulus containing about 30 per cent. of copper, with iron and sulphur; and, therefore, resembles copper pyrites in composition. It should not contain much less than 30 per cent. of copper, or it will need too much further treatment; nor should it contain much more, or the slag will be too rich.

The slag is mainly silicate of iron; it is black, usually containing fragments of silica, and fuses to a pasty mass; it must not be too fluid, or it will not be possible to rake it out; and it must not be too thick, or the regulus will not settle out properly. A good slag will not contain more than ·5 per cent. of copper, much of which is disseminated through the mass in the form of shots or "prills" of regulus.

The roasted ore contained sulphide of iron, Fe_2S_3, or perhaps FeS; oxide of iron, Fe_2O_3; oxides of copper, Cu_2O and CuO; and sulphide of copper, Cu_2S. When these are melted together the affinity of the copper for sulphur is so great that all the oxygen from the oxides of copper passes over to the iron in exchange for sulphur; thus—

$$Cu_2O + FeS = Cu_2S + FeO$$

$$3Cu_2O + Fe_2S_3 = Cu_2S + Fe_2O_3, \text{ \&c.}$$

Any ferrous oxide formed at once combines with the silica, and the atmosphere being reducing, any ferric oxide present will be reduced to the ferrous condition, so that all the oxide of iron passes into the slag as ferrous silicate.

The object of the first roasting is to supply enough oxygen to combine with the iron and slag off the silica, leaving a regulus of sufficient richness. If oxidised copper ores are available these may be added and the roasting dispensed with.

Operation 3.—Roasting the Coarse Metal.—The coarse metal is either granulated by running into water, or, more usually now, is cast into blocks in sand moulds, and then ground to powder in a mill. A charge of about 3½ tons is let into a furnace similar to that used for Operation 1, and is roasted with free access of air for about twenty four hours. The chemical changes produced are exactly similar to those of the first roasting.

Operation 4.—Melting for White or Blue Metal.—A charge of roasted coarse metal, together with slags from Operations 5 and 6, and rich sulphide or oxide ores is let down on to the hearth of a furnace exactly like that used in Operation 2, and the charge is melted. The products are white or blue metal, according to the extent to which the roasting has been carried, which is tapped out into moulds; and a slag, which is raked out through the end door.

White Metal is a regulus containing very little iron, and, being nearly pure cuprous sulphide, it often contains 75 per cent. of copper; it is hard and brittle, and has a steel white colour. *Blue Metal* is not so free from iron, it contains about 65 per cent. of copper; it is bluish in colour, and is softer than white metal. Which of the two will be produced will depend on the amount of sulphur left in the charge after the roasting. If only enough sulphur is left to combine with the copper, white metal will be produced; if more, some iron will be retained, and blue metal will result. If the whole of the iron is slagged out some copper will pass into the slag.

Slag.—The slag called sharp slag is mainly silicate of iron, it is black and glassy, and as it contains 2 or 3 per cent. of copper, it is returned to the furnace for Operation 2.

Operation 5.—Roasting for Blister Copper.—A reverberatory furnace is used similar to that used for Operations 2 and 4. Four or five tons of white metal are introduced in pigs; the temperature is very gradually raised, so that the charge takes four or five hours to melt, air being allowed to circulate freely. The slag formed is skimmed off from time to time, the metal seems to boil from the evolution of sulphur dioxide, and copper is separated. The products are blister copper and roaster slag.

Blister Copper is an impure copper containing about 99 per cent. of copper; it derives its name from the fact that the surface is always blistered, by the escape of gas during solidification.

Roaster Slag is a dark reddish-brown vesicular mass, consisting mainly of cuprous silicate, and containing shots of metallic copper disseminated through it.

The chemical changes which take place are simple. Any sulphide of iron present is decomposed, and the iron oxide combining with the silica present from the sand moulds in which the pigs were cast, or from the furnace bricks, forms slag, which is removed. As the action goes on the copper sulphide oxidises; $Cu_2S + 3O = Cu_2O + SO_2$. The oxide thus formed reacts on the unaltered sulphide, and copper is separated, $Cu_2S + 2Cu_2O = 6Cu + SO_2$. If the quantity of oxide be insufficient, sulphide will be left undecomposed; if it be in excess, cuprous silicate will be formed. It is usually better in this process to pass some copper into the slag rather than to leave undecomposed sulphide.

Operation 6.—Refining.—About 10 tons of blister copper is stacked on the hearth of a large reverberatory furnace, the well of which is under the end door; the charge is melted down, and is kept melted for about fifteen hours, air being allowed free access, and the slag being skimmed off from time to time. Oxidation takes place, the sulphur and iron are completely removed, and a good deal of oxygen is taken up, cuprous oxide being formed, part of which dissolves in the copper, making it "dry," and part combines with the silica to form cuprous silicate. When the roasting has gone far enough, the slag is raked off; coal or charcoal is thrown on the surface of the metal, and ebullition takes place from the escape of carbon monoxide. A long pole of wood, preferably the stem of a tree, is plunged beneath the surface of the metal and held in position. A violent evolution of reducing gases takes place, by which the cuprous oxide is reduced. The foreman takes samples from time to time in a small ladle; these he cools and breaks; and from the fracture judges when the operation has been carried sufficiently far. As soon as this is the case the pole is removed, the copper is ladled into copper ingot moulds, and as soon as the ingots have set they are turned out into a vessel of water, "the water bosh."

The products are *refined copper* and *refinery slag.* The latter is a red slag consisting of a basic silicate of copper containing a large quantity of metallic copper disseminated through it.

The reactions are simple; during the roasting stages oxygen is absorbed and cuprous oxide is formed, which is partly dissolved and partly slagged off; during the poling this is reduced by the gases given off.

Dry, Tough Pitch and Overpoled Copper.—Melted copper can dissolve a considerable quantity of cuprous oxide, which renders it "dry." Dry copper is excessively brittle, and breaks with a granular dull-red fracture. An ingot of dry copper sets with a longitudinal depression along the top of the ingot.

When the oxide is removed by reducing agents the metal becomes tougher, till, when it breaks with a fibrous fracture, it is said to be at "tough pitch."

If the reduction be carried too far the metal again becomes brittle and breaks with a copper-coloured fracture with many bright specks, it is then said to be "overpoled." The more arsenic and antimony the copper contains, the more easily is it overpoled. An exactly similar result can be obtained by heating tough pitch copper in a reducing gas, such as hydrogen or coal gas.

The usual explanation of overpoling is that certain impurities have a less deleterious effect on copper when they are present in combination with oxygen than they have if all the oxygen is removed so as to leave them free.

Moss Copper.—As blue metal solidifies, metallic copper almost always separates in the form of fine filaments known as moss copper; much less separates from the richer white metal.

Removal of Impurities.—The impurities in the ore are partially removed during the various stages of the process. Iron and sulphur are completely removed, as is also lead; arsenic and antimony are partially evolved as oxides during the roastings, but the remainder passes into the metal and is not eliminated by subsequent operations. Nickel and cobalt pass partly into the metal and partly into the slag; gold and silver pass entirely into the metal.

Modifications of the Process.—If the blue metal obtained in Process 4 contain too much iron, it is ground, re-roasted, and run for pimple metal (a rich regulus), this being then run for blister copper, exactly as in the case of white metal. This adds two additional operations.

In Chili and other districts where fuel is dear, the process is shortened by running a richer coarse metal, then roasting that as completely as possible, and melting down, when bar copper and white metal are obtained. The bar copper in contact with the regulus dissolves about 1 per cent. or more of sulphur as copper sulphide, and is sold as "**Chili Bar**," whilst the regulus is also exported as "**Chili Regulus.**"

If oxidised ores are available, the roastings can be to a large extent or completely dispensed with.

The blast furnace is sometimes used for the fusions, but cannot be used for the roastings, as the atmosphere is always strongly reducing. Neither charcoal nor iron reduces copper sulphide.

THE MANSFELD PROCESS.

This is a very good example of a modification of the Welsh process under conditions where fuel is expensive and where the ores contain silver. The process is a little more complex than the ordinary form, on account of the necessity for the separation of the silver.

The ores are bituminous schists containing about 3 per cent. copper in the form of pyrites scattered through it.

(1) The ore is broken up and roasted in heaps to remove part of the sulphur.

(2) The roasted ore is melted in a small blast furnace with coke. The products are coarse metal similar to that produced in the Welsh process, and a clean slag, which is thrown away.

(3) The coarse metal is calcined in stalls (rectangular chambers with three sides). A layer of brushwood is spread on the floor, and on this the lumps of ore are stacked, the front is loosely bricked up, and the mass is lighted. The regulus burns for some weeks.

(4) The calcined ore is looked over; any lumps not properly burnt are put back for re-roasting, and the roasted regulus is run for blue metal, either in a reverberatory or blast furnace, the products being blue metal and a slag, which is returned to the furnace for the first operation.

(5) The blue metal is very finely ground.

(6) The ground residue is treated for the extraction of the silver by the Ziervogel process (p. 199).

(7) The residue, consisting mainly of copper oxide and some iron oxide, is in too fine a state of division for blast furnace treatment; it is, therefore, moulded into blocks with a little clay; these are charged into a blast furnace with a little silica, some pyrites, and coke, and run down. The products are coarse copper for refining, a small quantity of regulus, and clean slag, which is thrown away.

(8) The copper is refined on a small hemispherical hearth. The bars are broken up and charged with charcoal into the hearth, the blast is turned on, and the copper soon melts, more charcoal being added as required to keep up the temperature. The air acting on the surface of the metal oxidises the impurities in it, producing a rich slag (which is put aside for further treatment) and refined copper (which is usually dry).

BESSEMERISING COPPER MATT.

One of the most promising of modern processes is that of treating the regulus by blowing air through it in a converter similar to that used in the Bessemer steel process (p. 130), except that the tuyeres are placed horizontally through the sides of the converter, instead of vertically through the bottom. Coarse metal is very quickly blown to white metal, or white metal to blister copper. Coarse metal cannot conveniently be blown to copper in one operation on account of the large quantity of slag produced. This process is coming rapidly into use in America.

THE B.S. PROCESS.

This process was invented many years ago for the purpose of producing a pure copper from the impure ores of Cornwall. At one time all B.S. copper was made by this process, but now the name is given to pure copper made by any process.

The ore is run for blue metal exactly as described above. This is then roasted, till a small portion of the sulphide is converted into oxide, and the charge is melted down; the oxide formed reacts on some of the sulphide, producing metallic copper, whilst the bulk of the sulphide is unchanged. When the molten mass is tapped the copper, being heavier, falls to the bottom of the moulds, and is, therefore, called "bottoms," whilst the regulus remains above. The bottoms contain most of the impurities, and are used for preparing a very impure copper; whilst the regulus, which is nearly free from impurities, is converted into B.S. copper.

Smelting Oxide Ores.—These may be readily smelted in blast furnaces in the usual way, some basic material being used to form a suitable slag. As copper is very apt to pass into the slag as silicate, it is always well to add sulphur in some form, so that whilst the bulk of the copper will be obtained in the metallic form, there will be some regulus, and the slags will thus be kept clean.

WET COPPER PROCESSES.

Many of these have been suggested. Oxidised ores may be dissolved in acids, and the copper afterwards precipitated by means of iron.

Extracting Copper by Wet Methods from Cupriferous Pyrites.—This is the only wet process in general use, and though modified in detail in various works the principle is always the same. Pyrites containing 2 to 4 per cent. of copper is largely imported from Spain for the manufacture of sulphuric acid. It is burnt in kilns in the usual way, and the residue or "burnt ore" is sent to the copper extractor for treatment.

Burnt Ore.—This will be mainly ferric oxide containing about 2 to 4 per cent. of copper and 1 to 5 per cent. of sulphur. The quantity of sulphur should not be less than the copper; or addition of pyrites, "raw" or "green ore" will be necessary.

Grinding.—The burnt ore is ground to a coarse powder with from 10 to 15 per cent. of its weight of salt.

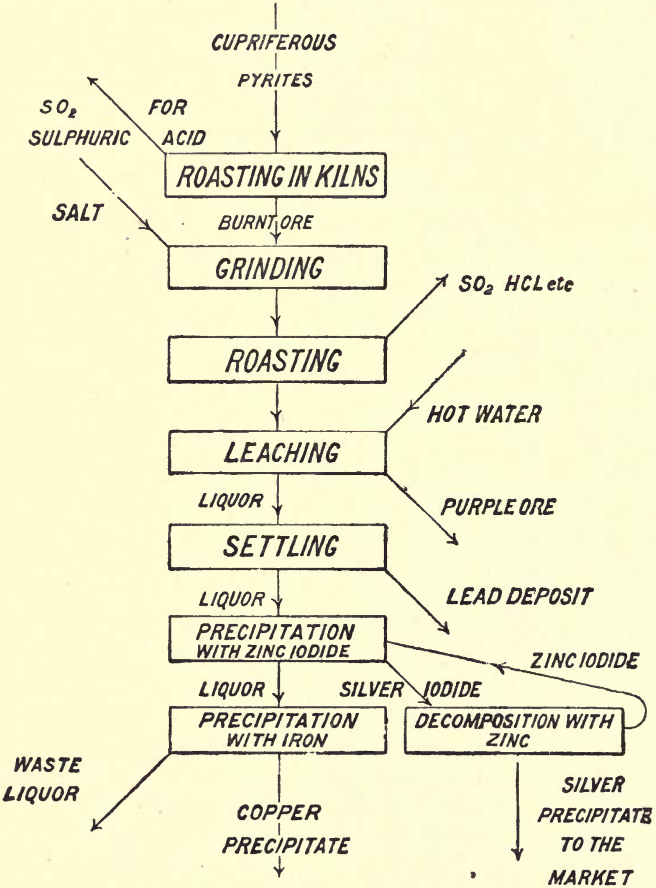
Roasting.—The mixture is charged on to the hearth of a reverberatory furnace, preferably of the close roaster type (p. 66), and is roasted at a low temperature till all the copper is rendered soluble. The charge is then drawn and allowed to cool.

The reactions which take place are very complex, so that only type equations can be given. The sulphide of copper in the charge is no doubt partially roasted to sulphate, which then reacts on the salt, forming sodium sulphate and copper chloride,

$$Cu_2S + 5O = CuSO_4 + CuO,$$
$$CuSO_4 + 2NaCl = CuCl_2 + Na_2SO_4.$$

Some sulphuric acid or sulphur trioxide is formed, probably by the action of oxygen on sulphur dioxide in contact with ferric oxide; this reacts on copper oxide, forming sulphate, which is in its turn converted into chloride. Whatever may be the reactions the ultimate results are—(1) the copper is mainly converted into cupric chloride, $CuCl_2$, unless the temperature be too high, when some cuprous chloride, Cu_2Cl_2, is formed; (2) the salt is partially converted into sodium sulphate, a considerable

Fig. 39.— THE WET COPPER PROCESS



quantity, however, remaining unaltered; (3) any silver present is converted into silver chloride. Hydrochloric acid and sulphur dioxide are evolved in large quantities.

Leaching.—The roasted ore is transferred to tubs, provided with perforated false bottoms and a filtering bed of pebbles; and is treated with water, containing a small quantity of acid, usually obtained by the condensation of the hydrochloric acid evolved during the roasting. The copper chloride dissolves in the water, together with the soluble alkaline salts; and the silver chloride is dissolved by the excess of salt. The residue which is left in the vats, called *purple ore*, is nearly pure ferric oxide, and is used for fettling puddling furnaces, or it may be smelted, though its fine state of division unfits it for treatment in the blast furnace.

Settling.—The green liquor is run into a settling tank, where a considerable amount of deposit, principally lead sulphate, settles out, and the clear liquor is run into another vat for the separation of the silver.

Separation of the Silver.—The ore contains a minute quantity of silver, and as this passes into solution with the copper it is worth recovery. The process always used is that known as Claudet's. The quantity of silver contained in the solution is determined, and the necessary amount of a solution of a soluble iodide is added to precipitate it. The silver is present as chloride dissolved in excess of salt, in which the iodide is insoluble; it is, therefore, precipitated, $AgCl + NaI = NaCl + AgI$. Care must be taken not to add too much iodide, or cuprous iodide would be precipitated and iodine liberated,

$$2CuCl_2 + 4NaI = Cu_2I_2 + 2I + 4NaCl.$$

When the precipitate has subsided the liquor is drawn off, the precipitate is washed and treated with zinc, which decomposes the iodides, $2AgI + Zn = ZnI_2 + 2Ag$. The residue is washed, dried, and sold as silver precipitate or silver mud; and the solution of zinc iodide is standardised and used for precipitating a fresh quantity of silver.

Precipitation of the Copper.—The copper liquor is run into wooden tanks, in which a quantity of clean iron scrap, preferably malleable iron, has been placed. Precipitation begins at once, and as it slackens is accelerated by heating the solution by blowing in steam. The copper is soon precipitated, and the waste liquor is run away. The precipitated copper is thrown out of the vat on to a coarse screen, by which any fragments of iron are separated; and is afterwards spread on a drying floor, which is heated by flues, to dry. It is then ready for the market as "copper precipitate," a red-brown powder consisting mainly of metallic copper coated with oxide and mixed with a certain quantity of iron oxide. After drying it usually still contains as much as 10 per cent. of moisture. It should contain when dry from 80 to 86 per cent. of copper, and only needs melting with a reducing agent, and refining. If it be dried at too high a temperature it turns black and oxidises to cupric oxide, CuO, its percentage in copper being thereby decreased.

Copper Refining by Electrolysis and Reduction of Matts by Electricity.—See p. 229.

CHAPTER XIV.

LEAD.

Lead.—Lead is a bluish-gray metal; it is very soft, can be readily scratched by the nail, and marks paper. It is very malleable, but it is the least tenacious of the common metals, and, therefore, cannot be drawn into very fine wire; it is tough and flexible; not sonorous when hammered, but if cast in the form of a spherical segment it is somewhat sonorous. It melts at about 325°, and just below its melting point becomes very brittle. The specific gravity is about 11·4. It is not acted on by dry air at ordinary temperatures; but, exposed to air and moisture, becomes covered with a white film of basic carbonate. Heated to redness in air it oxidises readily, forming litharge. It is dissolved by nitric acid, to a slight extent by strong sulphuric acid, but is not acted on by dilute hydrochloric or sulphuric acids.

Commercial Lead.—This is one of the purest of the metals in commerce, rarely containing more than traces of impurities. It sometimes contains traces of copper and other substances, which are objectionable when it is to be used for the manufacture of white lead or of red lead for certain purposes.

Lead Ores.—The most widely-distributed ore of lead is Galena (PbS), and almost all smelting processes are arranged for its treatment. The ore is crushed and dressed, and being very heavy, most of the gangue is easily removed, so that the ore is delivered to the smelter in a fine state of division, and usually contains from 70 to 84 per cent. of lead. Galena always carries more or less silver, so that the processes for the smelting of lead ores usually involve also the separation of the silver.

LEAD SMELTING PROCESSES.

Of these there are a very large number; they can be best classified according to the form of furnace used.

1. Smelting in reverberatory furnaces:
    - α The Flintshire or Air-reduction process.
    - β The Cornish or Iron-reduction process.

2. Smelting in blast furnaces:
   - α Small furnaces—as the slag hearth.
   - β Water-jacket furnaces.

3. Smelting in shallow hearths:
   The Scotch ore hearth.

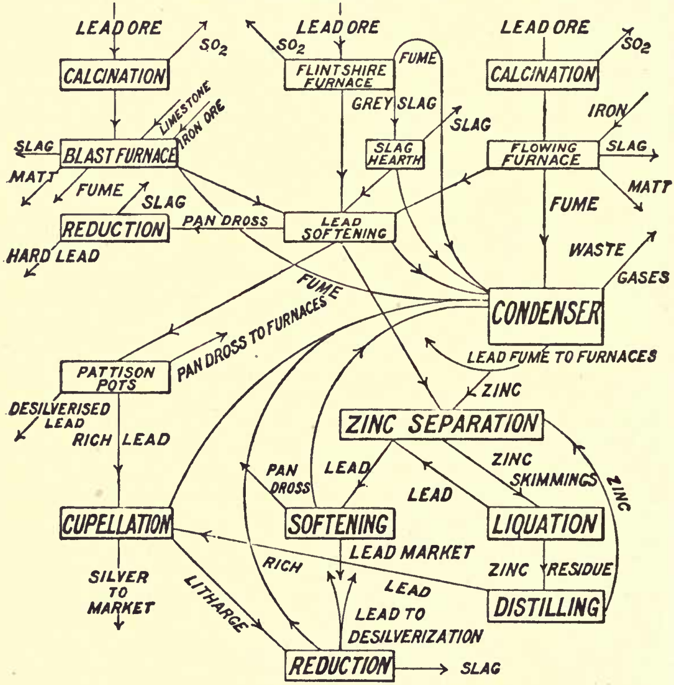


Fig. 40.—PLAN OF LEAD AND SILVER PROCESS

THE FLINTSHIRE PROCESS.

This process has been used from an early period for smelting lead ores in North Wales. The ores are very pure, and the gangue is mainly carbonate of lime.

The furnace used is a large reverberatory furnace, the hearth of which is made to slope to a well, from which a tap hole leads to a lead pot outside; the working bottom is made of gray slag, so that the ore does not come in contact with the brickwork. A charge of 1 ton (21 cwts.) of galena is let down into the hearth of the furnace, which should be at a very dull-red heat; the charge is spread evenly, and then roasted for about two hours at a very low temperature, during which time it is frequently turned, and great care is taken not to allow the temperature to rise high enough to cause it to clot. The fire is then made up,

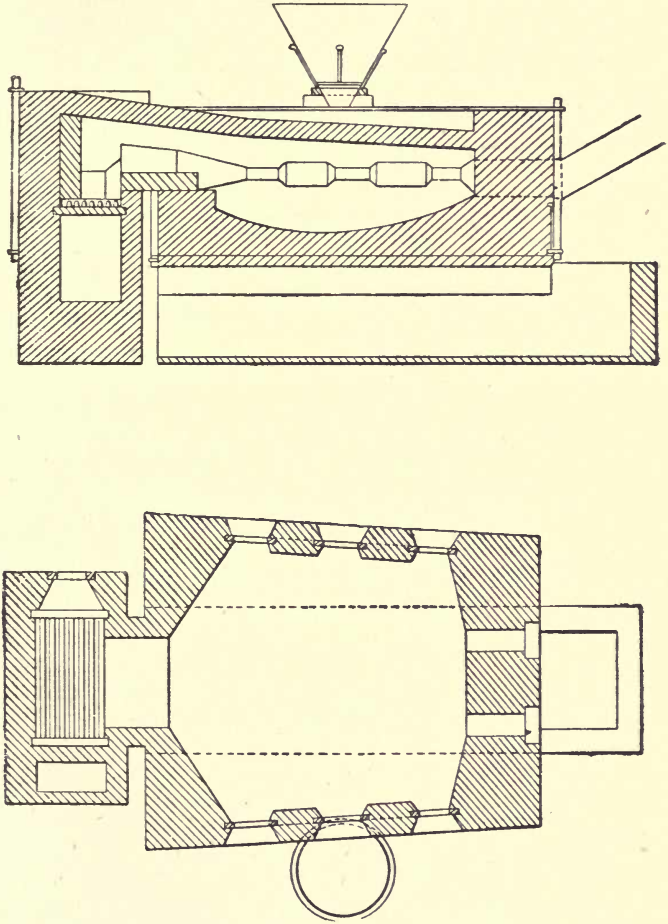


Fig. 41.—Flintshire Lead Furnace.

and the doors are closed, lead begins to run freely from the mass, and when this ceases the temperature is still further raised, so as to melt the charge down. The molten charge in the well will consist of a layer of melted lead, above which is a mass of melted galena and gangue. The doors are opened; lime is now thrown in and mixed up with the mass of galena which, being rendered stiff by the cooling and admixture of the infusible lime, is lifted with paddles and spread over the upper part of the hearth. The temperature is lowered, the charge roasted for a short time, and again run down; any unreduced material is again stiffened with lime, set upon the hearth, allowed to drain, and raked out as gray slag; a little coal slack is thrown into the well, and the lead is tapped out.

The time occupied by the process is about six hours; about $14\frac{1}{2}$ cwts. of lead will be obtained (about 80 per cent. of that in the ore), and about 3 cwts. of gray slag. The gray slag, which contains about 40 per cent. of lead, is a pasty unfused mass, and is put aside for further treatment. The lead in the pot is skimmed, the skimmings are thrown back into the furnace, and the lead is ladled into moulds.

Chemical Reactions.—The process is often called an air-reduction process because no reducing agents are used, the separation of the metal being brought about by a somewhat complex series of reactions, in which the oxygen of the air plays a very important part.

When galena is roasted with free access of air, the reactions which take place are:—

$$(1)\ PbS + 3O = PbO + SO_2$$
$$(2)\ PbS + 4O = PbSO_4$$

The roasting is never more than very partial, a large quantity of lead sulphide remaining unchanged. When the temperature is raised this unacted-on galena reacts on the oxidised products thus:—

$$2PbO + PbS = 3Pb + SO_2$$
$$PbSO_4 + PbS = 2Pb + 2SO_2$$

and lead is separated. The same series of reactions is produced by the second roasting and running down, and at the end there is always a good deal of unreduced galena left in the gray slag.

Influence of Impurities in the Ore.—As above remarked, this process can only be used with pure ores. Impurities may act in three ways—(1) They may be inert, and may by their bulk interfere with the roasting. This is the case with lime, or barytes, if in large quantity. (2) They may make the mass so fusible that it clots or softens before the roasting is sufficiently complete for a proper amount of reduction to take place. This is the case with many sulphides. Antimonite (antimony-sulphide) is very fusible, and forms a still more fusible double sulphide with galena. Iron and copper sulphides also form fusible double compounds with galena, and thus cause a softening of the charge. (3) They may combine with the lead to form compounds not easily reduced at the temperature of the furnace, and thus cause a large amount of lead to pass into the slag. This is the case with silica, which combines very readily with oxide of lead, forming normal or basic silicates, say $2PbO,SiO_2$. These silicates are very fusible, cause the charge to clot, and are not reduced by charcoal at the temperature of the furnace. If even a silicate no more basic than the one mentioned above be formed, each pound of silica will combine with about 7 lbs. of lead, and carry it into the slag.

There are other impurities which are objectionable because the metal which they contain is reduced and passes into the lead. Antimony sulphide is partially reduced and makes the lead hard. Copper sulphide is also partially reduced, the metal alloying with the lead. Zinc is partially reduced and volatilised, carrying with it a considerable quantity of lead. In this, as in other lead processes, the silver and gold pass completely into the lead.

Modifications.—The process is modified in various ways in different parts of the world, according to local requirements, but the principle is always the same—viz., (1) a roasting to form oxide and sulphate, and (2) the reaction of these on unaltered sulphide to separate lead. In some cases, if the gangue is of such a nature as to yield a fusible slag, the roasting is made as complete as possible before the charge is run down, and the slag and lead are both tapped out. This is the practice in Derbyshire where lead smelting is still carried on.

THE CORNISH PROCESS.

In Cornwall lead ores were abundant, and were at one time worked, but, as the gangue was silica, and as they were often mixed with copper sulphides, the Flintshire process could not be used, and a process called the flowing furnace or iron-reduction process was introduced. The process consisted of two stages.

(1) *Roasting or Calcining.*—The charge of about 2 tons of ore was let down on to the hearth of a reverberatory furnace with a flat bed, and roasted at a low temperature for about fifteen to twenty hours; it was then raked through holes in the bed into a cavity under the furnace and allowed to cool.

(2) *Flowing.*—For this a reverberatory furnace was used, the hearth of which sloped towards a well, and the working bottom of which was made of slag. A charge of about 2 tons of the calcined ore was thrown in through the doors, the doors closed, and the charge quickly melted down. Some lead separated, especially in the case of rich ores, and this was tapped out. The residue was dried with lime (as in the Flintshire process), set up on the hearth, 1 to 2 cwts. of scrap iron thrown in, and the charge again melted down and tapped out. The mass in the pot separated into three layers. At the bottom the lead, next a a slurry or regulus, and at the top a black slag; the regulus and the slag were directed into different moulds.

The lead obtained was very hard, owing to the reduction of other metals by the iron, which pass into the lead.

The slurry or regulus or matt consisted mainly of ferrous sulphide, and contained any copper that was in the ore as sulphide; also most of the zinc and some antimony, silver, and other metals.

The slag was mainly silicate of iron; it contained very little lead, and was thrown away.

The Chemistry of the Process.—The reactions during the early stages were the same as those of the Flintshire process. When the iron was added a further set of reactions took place. Any unaltered galena was decomposed by the iron, $PbS + Fe = FeS + Pb$; the sulphide of iron forming the regulus; any silicate of lead which had been formed was also decomposed,

$$2PbO, SiO_2 + 2Fe = 2FeO, SiO_2 + 2Pb;$$

the ferrous-silicate thus formed combining with the lime-silicate formed by the action of the lime on the silica, and producing the slag.

Modifications.—The process has been modified in various ways, The preliminary roasting is sometimes dispensed with, but for a rich ore this involved the use of a much larger quantity of iron. The roasting and reduction are sometimes conducted in one furnace, and the iron is added before the charge is run down for the first time, in which case no lime is used, and the charge is not set up.

Reduction of Oxidised Materials.—Oxidised ores are readily reduced in a reverberatory furnace, though it is rarely used for the purpose. It is, however, commonly used for the reduction of litharge, dross, pan skimmings, and other materials produced in the works. The furnace is of the ordinary Flintshire type, the hearth being made of gray slag or some other not very siliceous material. The charge, mixed with coal slack, is thrown into the furnace; the doors are closed, and the oxides are rapidly reduced. The metal separates, and a small quantity of slag is formed, consisting mainly of silicate of lead, but varying with the nature of the material being treated.

SMELTING IN BLAST FURNACES.

THE SLAG HEARTH.

This small furnace, used in the North of England for smelting gray slags, may be taken as a type of the small blast furnaces used for lead smelting. The furnace consists of a brick shaft with an arched opening in front. The bottom of the furnace is an iron bed-plate made to slope towards the front, and outside is placed the lead pot and the slag pot to receive the lead and slag. The lower part of the furnace is lined with fire-brick to a height of about 4 feet; and the front is closed by a cast-iron plate, the forestone, the top of which comes to the same level as the fire-brick, and the bottom of which comes to within about 6 inches of the bed-plate. Through the back of the furnace, about a foot above the bed-plate, is placed a single tuyere. A space for charging is left above the forestone.

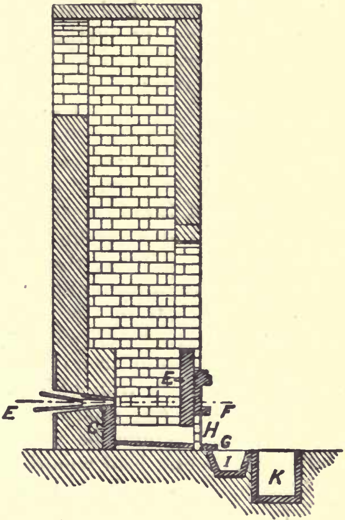


Fig. 42.—Slag Hearth (Percy).

E, Forestone.	I, Slag pot.
H, Tap hole.	K, Lead pot

The lead trough is an iron pan divided into two by a vertical partition which does not reach the bottom, and the larger half is filled up with coarse cinders. The slag trough is of brick.

The bed-plate is covered with a layer of coarse coal ashes to about 1 inch below the tuyere; this "ash-bottom" slopes so that at the front its upper edge is just above the bottom of the forestone, and the opening beneath the forestone is luted with clay. The fire is lighted with peats, then coke is put in, and the blast is turned on. When the furnace is hot, gray slag and coke are introduced, together with a little black slag and agglomerated ore or browse from a previous charge. As the lead is reduced it sinks through the ash-bed to the bed-plate, runs out through a tap-hole in the stopping into the lead trough, filters through the cinders in the large division, and finds its way into the smaller division, from which it is ladled into moulds. The slag accumulates on the top of the ash-bed till a tap hole is made for its escape in the stopping under the forestone, when it flows over the lead trough to the slag trough. The supply of gray slag and coke is kept up till near the end of the shift, about seven hours from lighting; then no more material is charged, but the charge is allowed to burn down. The blast is turned off, the ashes on the bed-plate are raked out, and the adherent slag or "browse" is put on one side for further treatment. Water is thrown into the hearth to cool it; and it is cleaned out, adherent masses of slag removed, and prepared for the next charge.

The products are slag lead (a hard, impure lead) and slag (a silicate of lime and other bases containing some lead), which is thrown away. In a shift about 80 cwts. of gray slag are treated, yielding about 18 cwts. of lead.

Chemistry of the Process.—The reactions are almost identical with those of the Flintshire process. The air blown in partly oxidises the sulphides; the hot coke acts as a reducing agent, decomposes any oxides, and, at the same time, the temperature being higher than in the reverberatory furnaces, the silicates of lead are decomposed by the carbon. Iron slag, tap cinder, or other basic silicate of iron is sometimes added, and is a great help to reduction. The iron oxide is reduced by the coke, and the iron decomposes any unaltered galena.

The process cannot be worked continuously, as the furnace would get too hot, lead would be lost by volatilisation, and the walls of the furnace would be slagged away.

Modifications.—The process may be modified in various ways. The length of the shift, the mode of working, and the form of furnace vary in different localities. In Spain a small circular furnace (the Spanish slag hearth) is used. The Carthagena or Economic furnace is a very similar circular furnace, in which the draught is produced by a chimney.

SMELTING IN BLAST FURNACES IN AMERICA.

The ores treated in the silver-lead districts of the United States are not by any means of uniform composition; some are rich, pure galenas, but most of them are more or less impure, containing considerable quantities of silica and frequently sulphides of other metals. The ore is delivered to the smelter in a fine state of division, and is, therefore, not well suited for blast furnace work, it is, therefore, roasted in order to remove part of the sulphur and, at the same time, to agglomerate the mass, so that it can be charged into the blast furnace. The reverberatory furnace used has a double hearth; the hearth nearest the fireplace is like a melting furnace, and beyond that is a long, flat roasting hearth made in several steps. The charge

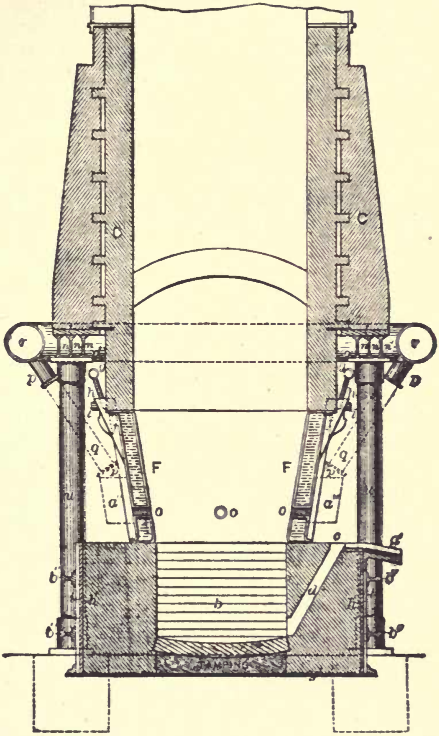


Fig. 43.—Blast Furnace for Lead Smelting. From Hofman's *Metallurgy of Lead.*

F, Water-jacket.
r, Blast main.
o, Tuyeres.
b, Hearth.
a, Air-jacket.
d, Syphon tap.
c, Shaft.

is delivered by a hopper at the end furthest from the fireplace, and is worked forward to the second hearth or "fuse box," where it is fused or agglomerated.

The blast furnace used is usually of the Rachette type, and is provided with a water-jacket, a closed top, and condensing apparatus. In order to avoid the necessity for tapping, the automatic tap is very often used. This is a pipe communicating with the bottom of the hearth, in which the lead rises to the same level as it is in the well of the furnace, and either runs or can be ladled out, the slag and matt being tapped at intervals through a tap hole.

The charge consists of agglomerated lead ore, oxidised iron ores (hæmatite), limestone, and rich slag, the constituents being so calculated that the iron will reduce the galena, and that the slag shall be a monosilicate with the approximate formula

$$(2\,FeO, SiO_2 + 2\,CaO, SiO_2).$$

If the ore be a mixed carbonate and sulphide, roasting can be dispensed with, and the quantity of iron required will be less.

The products are lead, which carries all the silver; matt or slurry, which contains the copper; and sometimes an arsenical speiss carrying any nickel and cobalt which were present in the ore.

The lead is usually hard. The matt may be very complex and will carry some silver. The slag is black, readily fusible, and should be nearly free from lead; it is run into deep moulds, so that any lead held in suspension may settle before it solidifies.

Chemistry of the Process.—The chemistry of the process is much the same as that of the other processes described. The oxides are reduced by carbon monoxide and carbon, the sulphides are reduced by iron, and the sulphates by the joint action of carbon and iron.

Oxide Ores can be reduced without the addition of iron ore, but if the gangue be silica, iron ore is always advantageous—(1) as it ensures the reduction of lead silicate; and (2) helps the formation of a fusible slag.

Modifications.—A combination of the Flintshire and the blast furnace processes is sometimes used for fairly pure galenas. The charge is roasted in the usual way and then run down, but instead of drying up with lime the whole charge is tapped out. A large quantity of lead is separated, and the slag, consisting of fused galena, lead silicate, &c., after cooling and being broken up, is in a good condition for being smelted in a blast furnace.

Sulphate Ores.—If ores containing lead sulphate be smelted in a blast furnace, the sulphate is reduced to sulphide, and unless iron be added lead will not be separated.

LEAD SMELTING IN SHALLOW HEARTHS.

THE SCOTCH ORE HEARTH.

This form of furnace is still in use in Scotland and the North of England. As worked at the Leadhills, Lanarkshire, it consists of an iron trough or sump, 24 ins. × 22 ins. and 6 ins. deep, set in masonry. This is surrounded on three sides by rectangular iron bars, 8 ins. square, placed so as to form a rectangular space. These bars are technically called the back and side stones; under the back stone passes the tuyere, an elliptical iron tube for admission of the blast. Over the space left in front, at the level of the top of the side stones, is a girder which carries the front of a shaft for the escape of the fumes. In front of the hearth is a sloping iron plate, the "work plate," across which a gutter runs to the lead pot, which is placed in front of it. The lead pot is divided into two parts by a partition which does not reach to the bottom; the lead flows into the larger division, and is ladled out from the smaller one. The blast is supplied by a Roots' blower. When the hearth is at work the sump is full of lead and the hearth is filled with ore and fuel up to the level of the top of the side-

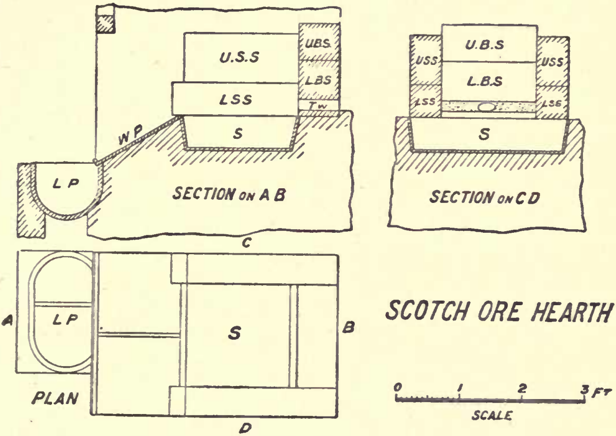


Fig. 44.—The Scotch Ore Hearth.

U S S,	Upper side stone.	U B S,	Upper back stone.
L S S,	Lower ,,	L B S,	Lower ,,
L P,	Lead pot.	Tw,	Tuyere.
S,	Sump.		

stones. The workman puts in a rod and draws out the red-hot mass on to the work plate; a good deal of lead separates and flows into the lead pot; the mass is turned over, any hard lumps of slag are picked out and thrown on one side, and the remainder is returned to the furnace; a shovel full of lime is thrown in, and a fresh charge of ore and fuel is added. These operations are repeated every four or five minutes.

The working shift is six hours, and the process is worked continuously through the week. The amount of lead obtained for each hearth is about 18 cwts. per shift.

The products are lead and gray slag. The slag is a porous mass containing galena, unburnt coal, and a considerable amount of lime, which rapidly disintegrates on exposure to the air.

The chemical reactions which take place are the same as in the Flintshire process, the low hearth allowing of oxidation being carried on by means of the blast.

The American Ore Hearth.—This is similar in principle to the Scotch hearth. It is made of larger size, and in place of the side and back stones, hollow iron castings are used, through which water is kept in circulation, and, in one form, in place of water air for the blast was made to circulate, thus producing a hot blast.

SOFTENING THE LEAD.

The lead obtained by any of the processes may be hard, from the presence of antimony, copper, and other impurities, which it is advisable to remove before desilverisation; this is done by "softening." If the lead be fairly pure, especially if the zinc process of desilverisation is to be used, the softening may be left till after desilverisation.

The softening is simply a purification by oxidation; the melted lead is exposed to the air, oxidation takes place, and the metallic impurities are oxidised together with part of the lead, and form a scum on the surface, which is raked off.

The softening pan or furnace is a reverberatory furnace, the bed of which is a pan made of cast or wrought iron, and capable of holding from 10 to 30 tons of lead. The lead is melted, either on the hearth, or in a separate pot and ladled in. Oxidation commences at once and proceeds slowly, the end of the operation being judged by ladling out a sample of the lead and testing it. The time occupied by the softening will usually be between six and forty-eight hours; the latter only in the case of very hard lead. The dross is skimmed off, and the lead is tapped out into moulds. The lead is now soft, and if it has been desilverised, is ready for the market. The "pan dross" is reduced, either in a reverberatory or blast furnace, and the "hard lead" thus obtained is softened. The dross from this when reduced often contains enough antimony to make it valuable for making alloys of antimony, such as type metal, &c.

SEPARATION OF SILVER FROM LEAD.

Almost all lead ores carry silver, some very considerable quantities; this all passes into the lead during smelting, and can be separated from it. There are two processes for the separation of the silver, or rather for concentrating it in a small quantity of lead—viz., the Pattinson or crystallisation, and the zinc, processes.

THE PATTINSON PROCESS.

This process depends on the fact that as a liquid crystallises it tends to reject substances held in solution. If lead containing silver be allowed to crystallise, the crystals which form at first will be free from silver, which will, therefore, concentrate in the residual lead. The plant required consists of a series of iron pots (the number for a full equipment being 12), arranged side by side in a row. Each pot is about 5 feet 6 inches in diameter, and nearly hemispherical in form; they are set in brickwork, and each is heated by a separate fire. Between each two pots is a crane for working the ladle. The ladle is 16 to 20 inches in diameter, and is pierced with ½-in. to ¾-in. holes, and is mounted on a long shaft; in addition, baling ladles, not perforated, and a flat perforated skimming shovel are used.

As the lead comes from the furnace a sample is assayed and the pigs are stamped with the number of ounces of silver per ton which it contains. Suppose a charge is sent down containing say 16 ounces to the ton, 9 tons of this will be melted in No. 6 pot. As soon as it is well melted it is stirred up, the fire is drawn, and the metal is allowed to cool; the cooling being often accelerated by throwing water on the surface, and stirring in any solid crust which may form. Any dross which collects on the surface is skimmed off with the perforated shovel, and put on one side for reduction. As the metal cools, crystallisation begins, and the crystals, being heavier than the liquid, sink to the bottom. The perforated ladle is now taken, and with it the crystals are fished up, allowed to drain, the draining being accelerated by shaking the ladle, and the contents of the ladle are tipped into pot No. 5, and this is continued until two-thirds of the contents of the pot have been transferred. The fire under the pot is then lighted to keep the residual lead liquid, and it is baled over into No. 7 pot. Thus, of the 9 tons of lead put into No. 6 pot, 6 tons will have gone downwards into No. 5, and 3 tons upwards into No. 7. The lead in No. 5 pot will assay about 8 ounces per ton, whilst that in No. 7 will contain about 32 ounces. Therefore, of the 144 ounces of silver contained in the No. 6 pot 48 ounces will have gone downwards, and 96 ounces will have gone upwards. Pot No. 5 is now filled up with 8-ounce lead, and the process is repeated, the 9 tons of 8-ounce lead giving 6 tons of 4-ounce in No. 4, and 3 tons of 16-ounce in No. 5; and so the process goes on. By the time the lead reaches No. 1, the market or D pot, it will not contain more than about ½ an ounce of silver per ton, whilst that which reaches No. 12 will contain 700 to 800 ounces or more.

If two-thirds of the contents of each pot are ladled as crystals—the method of thirds—the lead in each pot will be just about twice as rich as that in the pot before; the lead in the upper pots, however, usually falls below its theoretical richness for several reasons—(1) the increase does not keep up so well with the higher pots; (2) the lead which comes down to the desilverising house will probably not contain exactly the amount of silver required for any pot, and in that case it is better to put it into the pot which should be next richer rather than into the one which should be poorer. The theoretical values of the lead in the twelve pots are given below, with assays from Phillips of a set of pots in actual work—

Pot, . . . Nos.	1	2	3	4
Silver, ozs. per ton,	·5	1	2	4
,, ,, actual,	9 dwt. 15 grs.	19 dwt. 7 grs.	1 oz. 18·14	3·17·4
Pot, . . . Nos.	5	6	7	8
Silver, ozs. per ton,	8	16	32	64
,, ,, actual,	7·7·21	12·17·5	22·10·3	36·19·12
Pot, . . . Nos.	9	10	11	12
Silver, ozs. per ton,	128	256	512	1024
,, ,, actual,	59·9·16	96·9·4	167·3·2	273·6

No. 12 was crystallised, and the bottoms gave 514 ozs. 9 dwts. 18 grs.

It will be seen from this example that the falling off was very rapid, and as the higher numbers were reached, exceptionally so; No. 12 pot should, in practice, assay 700 ozs., beyond which it is practically impossible to concentrate by crystallisation.

A very large quantity of dross is produced, sometimes as much as 25 per cent. of the lead being oxidised. It will be seen also that a very large quantity of lead must be kept "floating," and, therefore, it will be a considerable time before the silver in any given quantity of lead can be realised, about 1,400 tons (say) of D lead being produced for 1 ton of lead in No. 12 pot.

THE ZINC PROCESS.

This process was first introduced by Parkes, and is known as the Parkes process. It was tried at various works, and abandoned. It has recently been reintroduced in a slightly modified form, and is now rapidly replacing the Pattinson process. It depends on the fact that if zinc be stirred up with molten argentiferous lead, it will dissolve the silver, and on cooling, owing to its lower specific gravity, it rises to the top, and can be skimmed off.

The plant used generally consists of four pots—two large ones (which may be of any size capable of holding 10 tons or more of lead) and two much smaller ones. The large pots should be provided with tap holes at the bottom, by which the lead can be run off, so as to avoid the necessity for ladling, and each pot is provided with its own fireplace.

To start the process, a charge of lead (say 10 tons) is introduced into one of the large pots, melted, and heated to the melting-point of zinc. About 1½ lbs. of zinc is then added for each ounce of silver present, well stirred in, and the fire damped down with moist fuel so as to allow the contents of the pot to cool very slowly. As cooling goes on, the zinc rises to the surface, and is skimmed off with a perforated shovel, the skimmings being transferred to one of the smaller pots. As soon as all the zinc has been removed, the lead is heated up again, another supply of zinc is added, and the charge is cooled and skimmed as before, the zinc skimmings being used as first zinc for the next charge. After the removal of the second zinc, the lead is tapped out into moulds. The skimmings in the small pot retain a very large quantity of lead; they are heated to a temperature above the melting-point of lead, so as to liquate out as much lead as possible; and, as this lead always carries a considerable quantity of silver, it is put back into the large pot for treatment with the next charge. The zinc scum is broken up, mixed with a little charcoal, put into a retort, and distilled. The condensed zinc, which carries over a little silver, is used over again; and the residue of rich lead, which will contain from 2,000 to 3,000 ounces of silver per ton, is cupelled. This process has several important advantages over the Pattinson process. It requires much less labour, does not require a large floating stock of lead, and can be used on lead of any richness.

Dezincification of the Lead.—The lead run off from the pots will contain about 1 per cent. of zinc, which must be removed before the lead is ready for the market.

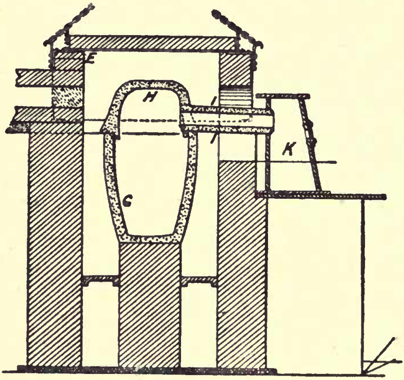


Fig. 45.—Zinc Retort (Battersea Form).

G, Graphite crucible.
H, Graphite cover.
I, Graphite tube.
K, Condenser.
E, Furnace cover.

The usual method of dezincification is identical with the process of softening; the lead is melted on the hearth of a reverberatory furnace holding 20 to 40 tons, and kept exposed to the air for from eighteen to twenty-four hours. Poling is sometimes resorted to; the metal is then run into a large pot, and a large piece of wood is put in and held beneath the surface.

In the Cordurié process the zinc is oxidised by blowing superheated steam through the molten metal contained in a covered iron pot.

CUPELLATION.

Either of the above-described processes yield a rich silver lead, which must be further treated by cupellation. The principle of this process has been already described; it is, therefore, only necessary here to give an account of the method in which it is carried out.

The furnace used is a small reverberatory furnace, without a fixed bed, the fireplace and chimney being at the long sides of the hearth. At one end of the hearth is an opening, through which passes a tuyere pointing downwards at an angle of about 30°, and at the other end is the working opening closed by sliding doors, over which a hood is placed to carry off the lead fumes.

The hearth of the furnace is made of an elliptical iron ring 4½ to 6 inches deep, with flat iron bars across the bottom. This is placed on the ground, and is rammed full of "bone ash" (calcined bones), which has been powdered and passed through a sieve of about 25 holes to the inch run, and moistened with water, to which a little potassium carbonate has been added. The mixture is rammed hard into the cupel, or "test" frame, so as to completely fill it. A cavity is then cut in it, so as to leave a hollow, as shown in the sketch, which will hold about 4 cwts. of metal. The cupel is dried, lifted into the furnace, and firmly wedged in its place, the edge of the iron ring being covered with bone ash, and the fire is made up. Lead is melted in the small melting-pot attached to the furnace, and is tapped into the cupel till the cavity is full. The blast is then turned on; the air, acting on the surface of the lead, oxidises it, and the litharge is blown in ripples towards the breast of the cupel; here a little channel is cut, by which it flows away and runs into an iron ladle put to receive it. The litharge solidifies in the pot, and is turned out in lumps, which soon fall to pieces. The supply of lead is kept up continuously, so as to keep the "test" full, and, as the lead only is oxidised, the residual lead gradually increases in richness.

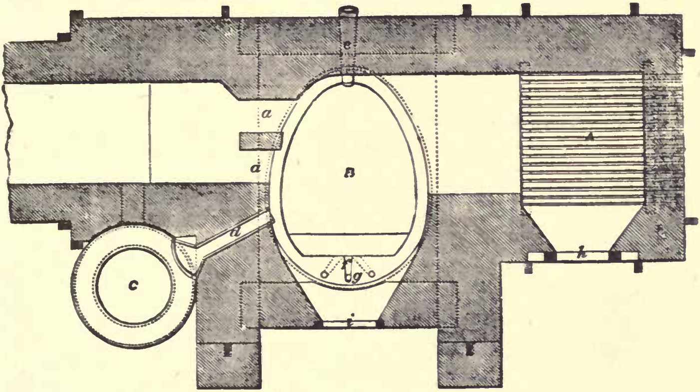


Fig. 46.—Cupel Furnace.

Assuming that the lead being treated is from the Pattinson process, and assays, say, 700 ounces to the ton, the process will be continued till about 2 tons of lead have been introduced, and, therefore, have been reduced by oxidation to about 5 cwts. The blast is now stopped, and an iron ladle on wheels is run under the middle of the cupel; by means of a special boring tool a hole is bored through the bottom, thus allowing the rich lead to run into the ladle. The hole in the cupel is then stopped with a plug of bone ash, and the process is resumed with fresh lead. The oxidation of 2 tons of lead down to 5 cwts. will take about sixteen hours.

The lead tapped will contain about 5,000 ounces of silver per ton. When six shifts have been worked, about 30 cwts. of this very rich lead will have accumulated. After each tap the cupel is carefully examined, and if it be found to show signs of giving way, it is replaced by a fresh one, and a new one is generally used for the last stage. The rich lead is now melted in the pot, run into the cupel, and the cupelling is continued, the litharge which comes over being usually kept separate from that which comes over during the earlier stages. When the whole of the lead has been introduced, the waves of litharge are blown forward and bright metallic patches are seen. As soon as the litharge clears off, leaving a bright surface of metallic silver, the blast is stopped, and the metal coming to rest, the surface looks bright and mirror like; if any impurities do come to the surface they are skimmed off. The fire is let down and the metal is allowed to cool. A crust soon forms over the surface, and this is soon blown up into blisters by the evolution of oxygen from the solidifying silver below. If these blisters or bubbles were left they would burst and scatter silver over the inside of the furnace; to prevent this, as soon as a bubble appears it is pricked with a pointed iron rod, and the pressure of the gas forces out the metal, which rises into a sort of cone. This is repeated until the metal has set, and by that time the whole surface will be covered with a mass of elevations. The cupel is now taken out, the bone ash broken away, and the cake of silver is ready for the market. Under the conditions described it will weigh about 8,000 ounces.

The time occupied in working, the size of the cake made, and other details depend very much on local conditions. The very rich lead from the zinc process is cupelled at once to silver.

German Cupellation.—The German cupellation is rather a scorification process; since the hearth is not of bone ash, but of non-litharge-absorbing marl. The furnace is circular, with a movable roof, so as to allow access for repairs. The resulting products are litharge, the first portions of which are very impure; and a crude silver, which must afterwards be refined on a bone-ash cupel.

CONDENSATION OF LEAD FUME.

Lead is so volatile that a portion is always carried away by the furnace gases. As these lead fumes are very poisonous and deleterious to vegetation, and as they are also of considerable value, they must be condensed. Very many methods of condensation have been suggested.

Long Flues.—This is one of the earliest forms of condenser used, but it is only fairly satisfactory, the escaping gases always carrying with them some "fume"; the advantage of this method is that no artificial blast is necessary, as is the case with most other forms of condenser. The longest flue of this type is that at Allen Smelt Mill, Northumberland, which is 4,451 yards long. The one at the Ballycorus Works, near Dublin, was about a mile long, but in this case it was not horizontal, but ascended the slope of a hill, at the top of which was a vertical shaft 60 feet high. These flues are of massive masonry, and are provided at intervals with doors by which access can be obtained for cleaning out. In some American works a suspended sheet-iron flue is used, which is cheaper to erect, and cools the gas much more efficiently.

Increase of surface facilitates condensation. Freudenberg suspends in the flue a large number of sheet-iron plates parallel to the direction of the current, and in some cases wires have been substituted for plates.

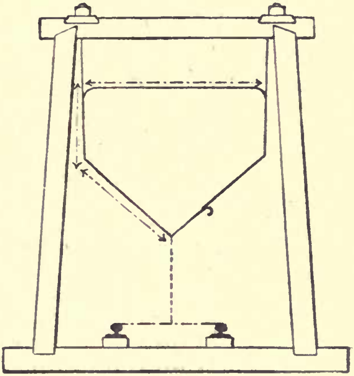


Fig. 47.—Iron Condensing Flue.

When the flues are enlarged they then form dust chambers, and these are very frequently used; usually cross-walls or baffles being arranged so that the gases shall not pass too rapidly through.

Water Condensers.—These were at one time more used than they are now; the gases are either made to bubble through water, or, more usually, are made to pass through scrubbers or chambers where they meet a descending shower of water. To facilitate condensation the chambers are usually filled with brushwood, or are provided with perforated shelves so as to break the current of gas and bring it into intimate contact with the water; they are usually so divided that the gases have to traverse them several times up and down. All condensers of this type require suction, by fan or otherwise, to draw the gases through.

The Bag Filter.—This is the most recently introduced, and probably the most efficient of the condensers. The fumes are drawn from the furnace by a suction fan, and pass through iron-cooling cylinders and through a long iron delivery-pipe, so that the temperature may be reduced below the point at which the filter-bags would be destroyed. The gases are then conducted into hoppers, and ascend into hanging bags made of unwashed wool 33 feet long. The gases filter through the bags and the fume is deposited. The bags are emptied every two days and the fume collected.

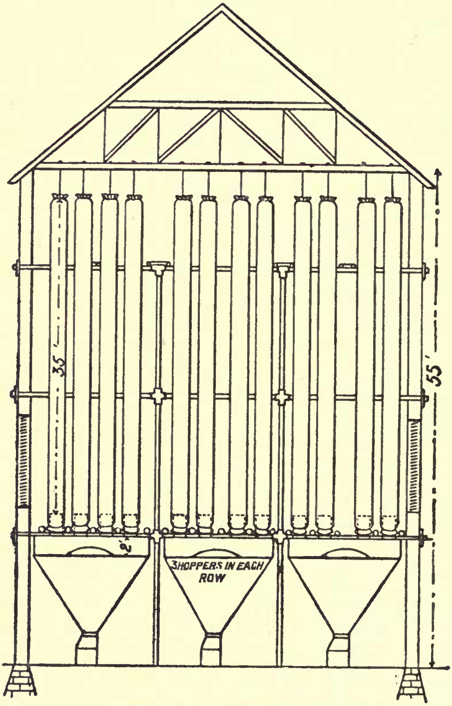


Fig. 48.—Bag Filters.

Electric Condensation.—This has been suggested, but has not proved a success. A powerful discharge of electricity into dust-laden air very quickly causes the dust to settle; if, however, the air be in motion, the settlement is not quick enough to be of practical value.

Lead Fume.—The dust or "lead fume" is a complex mixture of oxide, sulphide, and sulphate of lead, containing considerable quantities of other volatile materials present in the ore, such as zinc oxide, &c. There is always lime, other earthy impurities carried over mechanically, and only very small quantities, if any, of carbonaceous matter. Silver is always present; the richer the ores being treated the more silver will be in the fume.

The fume is smelted either in reverberatory or blast furnaces; in the latter case it is agglomerated by heating, so as to be more suitable for charging.

By the Bartlett process the fume is converted into a pigment. The fume is agglomerated, and is then charged into a blast furnace, fed with hard coke, and supplied with a large quantity of a very hot blast. Owing to the amount of air supplied there is but little reduction of lead; what is reduced, however, carrying with it the silver, but most of the lead is oxidised to sulphate or oxide and carried away by the gases; these, after passing through cooling tubes, are filtered through bag-filters, and the powder is ready for the market.

SHEET PIPE, &c.

Lead Sheet.—Lead being very malleable, is easily rolled into sheets. It is first cast into cakes, 5 feet or so square. These cakes are lifted on to a rolling table, provided with rollers to facilitate the moving of the heavy mass, and is passed backwards and forwards between a pair of rolls, the bottom one of which is fixed, but the top one adjustable, so that it can be brought down a little each draught. When the sheet has become too long for the table it is cut in two, and the two halves are rolled separately. Lead sheet for sulphuric acid chambers (chemical lead) should be as pure as possible, and lead for this purpose is always desilverised by the Pattinson process.

Lead Pipe.—The manufacture of pipe depends on the readiness with which lead, especially if hot, will flow under pressure.

A strong iron cylinder is used, which is carried on the ram of a hydraulic press; at the bottom of the cylinder is an opening or die the size which the pipe is to be made, the top of the ram being split to allow the pipe to pass out of the die; a core is fitted in the centre of the die, and this is carried on the cylinder so that its position relatively to the die remains fixed. The upper end of the cylinder is closed by a ram rigidly fixed to supports in the roof, and round the cylinder is arranged a trough in which a charcoal fire can be lighted to keep it hot.

The lead is melted in a pot, ladled into the cylinder, and allowed to set; the hydraulic press is then set in motion, the cylinder is raised, and, coming in contact with the fixed ram, the lead is subjected to great pressure, and is squeezed out of the annular space between the die and the core. The end is seized with tongs and wrapped round a reel on which the pipe is wound as it is squeezed out. When the cylinder is nearly empty, it is lowered, refilled, and the operation is resumed. The fresh lead added melts that left in the cylinder; the two cool together, and so a continuous length of pipe can be made of any required length.

Shot.—Shot is made of lead, to which a little arsenic has been added. The method of making shot is very simple; the molten lead is poured on to a perforated ladle or colander, in the bottom of which is placed a layer of pan dross or cinders; the lead filters through in drops, which fall down a tower 100 to 150 feet high. As they fall, being acted on by the cohesive power of the metal, the drops become spherical, and in that form solidify, and are received in tubs of water at the bottom of the tower.

Among the shot are many which are not round; these are separated by running down an inclined plane, in which there is a break. The round shot acquire sufficient impetus as they run down to jump over the break, whilst those which are not round fall through into a receptacle, and are re-melted. The shot are then sorted into sizes in cylindrical sieves, and are polished by being rotated in a barrel with graphite.

Larger shot, such as buckshot, about ·3 of an inch in diameter, are cast in moulds.

Litharge and Red Lead.—These are largely used in the arts, and as they are usually made at the smelting works their manufacture is commonly regarded as belonging to metallurgy.

Flake Litharge.—This is made by the oxidation of desilverised lead on a cupel; the molten litharge runs over and solidifies in masses, which soon fall to pieces, forming dark orange-yellow flakes.

Ground Litharge.—This is made by the oxidation of lead, the temperature not being allowed to become high enough to fuse the litharge.

The furnace used is called a "drossing oven." It is a very low reverberatory furnace with two fireplaces, which are at the same end of the furnace as the working door. Frequently there is no chimney, the products of combustion escaping by the central door and being carried away by a hood placed over it.

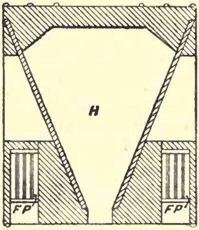


Fig. 49.—Drossing Oven.

About a ton of lead is placed on the hearth of the furnace; a dam is made across the front of the furnace of unoxidised material from a previous charge. The fire is made up, the lead melts, runs into a pool, and begins to oxidise; a little antimonial lead (about 1 cwt.) is added, which much facilitates the oxidation. As the lead oxidises, the litharge which is formed is pushed back, so as to expose a fresh surface of metal. In about ten hours oxidation is complete; any unoxidised lead is allowed to drain away, and the litharge is raked out.

The crude litharge, or "dross," is supplied to a mill consisting of a pair of horizontal mill-stones, where it is finely ground. A stream of water carries the material into the "puddler," a vessel, the contents of which are kept in agitation by revolving arms. There unoxidised lead separates, and the finely-divided litharge is carried over into settling tanks, where it is allowed to settle, and is thence removed, dried, and sieved. It now forms the buff-coloured powder known as "ground litharge."

Red Lead.—When litharge is heated to dull redness for some time, it absorbs more oxygen and passes into the condition of red lead, $3PbO + O = Pb_3O_4$.

The ground litharge is put into the "colouring oven," which is exactly similar in arrangement to the drossing oven, and is kept heated to dull redness for about 48 hours, with frequent rabbling. The mass becomes dark purple when hot, and when finished assumes, on cooling, the fine red colour of red lead. Samples are drawn from time to time, smoothed down on a piece of slate, and the tint judged. The red lead is ground and levigated; it is allowed to settle, and the thick sludge is pumped into an iron tank, and is thence run into the "drying-off" oven, which is exactly similar to the colouring oven in structure.

Orange Lead.—This has a paler colour than red lead, and is made by heating the carbonate (white lead) in the colouring oven in place of the oxide.

CHAPTER XV.

ZINC AND TIN.

ZINC.

THIS metal is one of the most recently discovered of those in common use, though its alloys have been known from a remote period. It has a bluish-white colour, is brittle at ordinary temperatures, but at about 150° C. it is malleable, and can be rolled into sheets; at higher temperatures it again becomes brittle. It melts at 412° C., and boils at a red heat. Exposed to moist air it tarnishes, becoming covered with a white crust of oxide, and, heated to near its boiling point in air, it burns with a very brilliant flame, depositing dense white fumes of zinc oxide known to the ancients as "philosopher's wool." It can be cast readily, and is largely used for making small ornamental castings. It dissolves freely in most acids; if very finely divided it decomposes water at about 100°, and it is attacked by alkalies.

Commercial zinc is always impure. Iron is always present; in larger quantities if the metal has been re-melted. A good zinc should not contain more than ·15 per cent., but the quantity is often very much larger. The presence of iron can be detected by the appearance of the fracture. If the spelter is free from iron the crystal faces will be smooth and bright; if iron is present in any quantity they will be covered with specks; and if the quantity is very large, as in "dross" spelter, the fracture will be granular. Lead is always present. When zinc and lead are melted together and cooled slowly, the metals will separate, but the zinc will retain about 1·5 per cent. of lead; and this quantity is very frequently present in commercial zinc. If more be present it will usually be found to be in the form of separated shots or layers. If zinc be redistilled, about ·2 per cent. of lead will remain in it, and this amount seems to distil over with the zinc, as it is not removed even by repeated distillation. Traces of tin, cadmium, arsenic, and copper are also often present.

Zinc comes into the market in two forms—in thick cakes, in which form it is called spelter; and in rolled sheets, known as sheet zinc.

Zinc Ores.—The ores of zinc have already been described (p. 78). Those generally used are blende and calamine. Whichever ore be used it is always subjected to a preliminary roasting or calcination. Calamine and smithsonite are usually calcined in heaps or kilns since it is only necessary to expel water and carbon dioxide, and they are much more readily crushed after calcination. Blende is always ground and then roasted on the hearth of a reverberatory furnace to expel the sulphur as completely as possible.

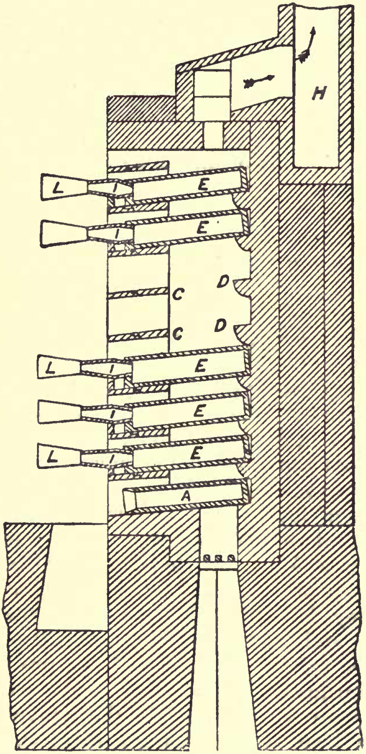
Methods of Smelting. –The zinc is converted into oxide by calcination or roasting before it goes to the smelter. The oxide is mixed with carbon and heated to a high temperature, by which it is reduced, $ZnO + C = Zn + CO$, and the zinc, being volatile, is distilled out and condensed in suitable condensers. The differences in the methods of working are, therefore, entirely differences in the arrangement of the plant, and not in principle.

THE BELGIAN PROCESS.

This process, which is now the one in general use, was introduced in 1810.

The retorts which are used are of fireclay; they are either round or oval; they are from 3 to 4 feet long, and about 6 to 8 inches in diameter, closed at one end and open at the other.

The furnace in which these are heated consists of a vertical chamber open in the front, with an arched roof, from which passes a flue, and at the bottom is a fireplace; the back is built with projecting ledges, and across the front are arranged transverse iron bars, one a little lower than each ledge. These furnaces are usually built in blocks of four, the fire doors being at the sides. The retorts are placed in position, the closed ends resting on the niches in the back wall, and the front ends on the transverse bars, which are so placed that the retorts slope downwards, each furnace holding from 40 to 50 retorts or pots. The spaces between the lower ends of the retorts are built up with clay and brick, the fires are lighted, and the temperature is gradually raised. The ore is intimately mixed with small coal, and the retorts are charged with a long semicircular shovel; the upper retorts, where the heat is less intense, receiving smaller charges than the lower ones, which are more strongly heated; and the lowest row of all is often left empty to protect the row above from the too intense heat of the flame. As soon as the charge is in a clay nozzle, I (Fig. 50), is luted on to each retort. As the charge gets hot the action commences, and carbon monoxide is given off, and burns with its characteristic pale-blue flame at the mouth of the condenser. As the temperature rises zinc begins to distil, the flame becomes "livid" from the burning vapours of zinc, and white fumes are deposited; or if much cadmium be present, the fumes will at first be brown from the formation of cadmium oxide. An iron cone or cylinder is now attached to condense any zinc that may escape condensation in the condenser. At intervals the cone, L, is taken off, the zinc which has condensed in the condenser is ladled out into an iron mould, and the cone is replaced. The distillation occupies about eleven hours; the charges are drawn, the retorts examined, and, if one has cracked, it is withdrawn and replaced by another, which is previously heated to redness in another furnace.



E, Retorts.
I, Clay condensers.
L, Iron cones.
D, Ledges to carry back ends of retorts.
C, Bars to carry front of retorts.
A, Empty retort placed at bottom.
H, Flue.

Fig. 50.—Belgian Zinc Furnace.

Products.—The products are zinc and residues; the former is remelted in iron pots and cast in moulds for the market. The fume which condenses in the cones is a dark gray powder, consisting principally of zinc and zinc oxide; it is re-treated, and, as it is very easily decomposed, is put in the upper retorts of the furnace, or it is sold for use as a pigment under the name of zinc gray. The residues are raked from the retorts, and often contain 10 per cent. of zinc. This process is only suitable for fairly pure ores. The silicates of zinc are not completely decomposed. If much iron be present, fusible iron silicates may be produced; and, if there be much lead, lead oxide is very likely to attack the retorts.

Chemistry.—The chemistry of the process is very simple. The zinc being in the form of oxide is reduced by the carbon, and the metal is distilled out; other metals present are also reduced, and, if volatile, distil out with the zinc.

Modifications of the Process.—The use of gaseous fuel for zinc distilling has now become common. The furnaces are made very much larger, often holding 100 to 150 retorts. In the Boëtius furnace (in which regenerators are not used) the producer is attached to the furnace and the air is heated while passing through passages in the brickwork. When the Siemens' producers are used the gas is passed direct to the furnace, and the air is heated by being passed through channels in the brickwork. The separate regenerators are found not to work well, owing to choking by oxide of zinc, caused by the escape of zinc from cracked retorts.

THE ENGLISH PROCESS.

This process was the first used in this country, having been patented in 1739, but is now abandoned. The distillation was carried on in pots or crucibles, the lids of which were luted on, the vapours being made to descend through a pipe fitting into a hole in the bottom of the crucible. The consumption of fuel by this process was very large.

THE SILESIAN PROCESS.

The retorts used in this process are larger than those used in the Belgian process, they are D-shaped, and the flat side rests on the bottom of the furnace. The retorts are arranged in two rows, the fireplace being between them, and the metal is condensed in clay condensers, to which iron cones are attached exactly as in the Belgian furnace. This form of furnace is specially adapted to heating by gas, the Siemens' process with or without regenerators being often used.

Zinc Refining.—The crude zinc carries over lead, which it is sometimes necessary to separate as far as possible. About 30 tons of metal is melted on the hearth of a reverberatory furnace provided with a well; and is kept melted for some days. Any lead undissolved sinks to the bottom, and the zinc is ladled off into moulds. If the lead is not to be separated the crude spelter is simply melted in iron pots and cast into moulds. Occasionally the zinc is re-distilled, and sold as re-distilled spelter, in which case it contains about ·2 per cent. of lead.

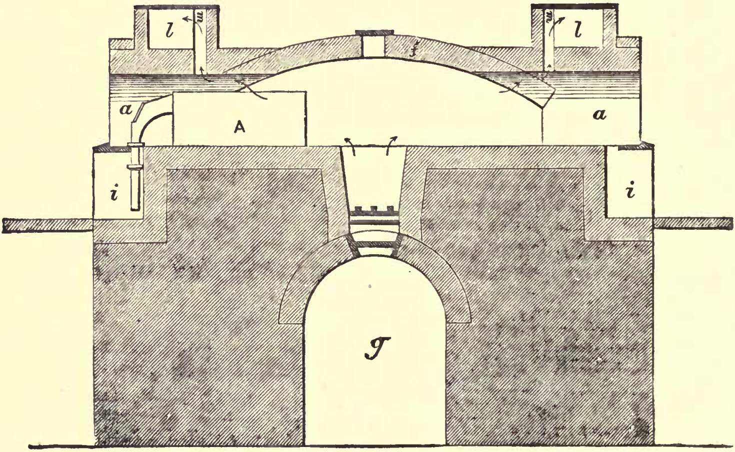


Fig. 51.—Silesian Zinc Furnace.

Zinc White.—Zinc oxide (ZnO) is largely used as a pigment under the name of zinc white. This may be made from zinc by burning it in air, but it is now largely made direct from the ore. In New Jersey a mixture of franklinite and zincite is used. The ore is finely ground, mixed with limestone and small coal, and spread on the hearth of a furnace, the hearth of which is an iron plate perforated with holes, through which air is forced. The zinc is reduced and volatilised, passing away with the furnace gases; as cooling takes place the zinc is oxidised, partly by excess of oxygen present, but mainly by the reduction of carbon dioxide, $Zn + CO_2 = ZnO + CO$. The gases are then carried through cooling tubes and passed through bag-filters exactly similar to those described on p. 174.

TIN.

Tin is a metal of almost silver whiteness, very malleable and flexible, but not elastic. When bent it emits a crackling sound, called the "cry" of tin. The purer the tin the better is the cry, though absolutely pure tin is said not to give any cry. It melts at 228° C., and is very slightly volatile. It crystallises very readily, either on cooling from fusion or by deposition by electrolysis. It does not tarnish in air at ordinary temperatures, but when heated to redness it is rapidly converted into oxide, SnO_2. It is readily dissolved by strong hydrochloric acid and by warm dilute sulphuric acid; nitric acid oxidises it, converting it into insoluble metastannic acid, $5H_2O, 5SnO_2$. Strong, hot alkaline solutions also attack tin, forming stannates.

Tin occupies a curious position almost intermediate between the metals and the non-metals; it displaces hydrogen from acids, forming salts, and its oxides can combine with acid oxides, also producing salts; but stannic oxide is rather acid than basic in its character, tending to combine with bases to form salts, in which the tin is in the non-metallic position. The behaviour of tin oxides to acid and basic oxides introduces difficulties in smelting.

Commercial tin is very fairly pure, though it contains traces of lead, iron, copper, arsenic, and other metals.

Tin Ores.—There is but one workable tin ore, cassiterite, the oxide, SnO_2. When this is extracted from the vein it is mixed with gangue, usually silica, galena, pyrites, copper pyrites, arsenical pyrites, often wolfram, and other substances, the tin-stone forming not more than 5 per cent. of the whole; and, as the impurities would interfere with the smelting, they are as far as possible removed by dressing before the ore is sent to the smelter. The ore is finely crushed and washed, by which means the earthy impurities are removed, and the concentrates contain the tin-stone and the other heavy minerals. These are roasted in a reverberatory furnace; this converts a considerable portion of the heavy sulphides and arsenides into light oxides, which are washed away by a second washing, and the residue, called "black tin," is ready for the smelter. If the ore contains much copper it is first roasted, then heated with dilute sulphuric acid, and the dissolved copper precipitated by iron.

If wolfram, a tungstate of iron and manganese, be present it is necessary to separate it. This is always done by Oxland's process, as, owing to its high specific gravity, it cannot be separated by dressing and, owing to its stability, it is not decomposed by roasting. It is broken up by fusion with alkaline carbonates or sulphates into an alkaline tungstate and oxides of iron and manganese.

The charge of black tin is mixed with soda ash or salt cake in such proportion that the quantity of alkali is only slightly in excess of that required to combine with the tungsten, and is introduced on to the hearth of a reverberatory furnace, the bed of which is a cast-iron plate, and which is so arranged that the products of combustion, after passing over the bed, circulate under it before passing to the chimney. The charge is roasted for two or three hours with constant stirring, and it "frizzles" and becomes soft; it is then drawn and, while still hot, is transferred to a lixiviating tank and well washed. The solution is crystallised for the recovery of sodium tungstate, while the residue is washed to remove oxides of manganese and iron, and dried for use.

Smelting Black Tin.—The "black tin" is almost always treated in a reverberatory furnace, indeed it is in far too fine a state of division to be suitable for the blast furnace. A ton of "black tin" is mixed with about 4 cwts. of anthracite, and the mixture is charged on to the hearth of a reverberatory furnace. After a time the charge is well stirred, and in about six hours will be well fused; it is then tapped out into a casting pot. The products are crude tin, a residue which remains in the furnace, and a black and stony slag, consisting largely of silicate of iron, which, as it solidifies, is removed and thrown on one side. The tin is ladled into moulds for further treatment. The residue in the furnace is mainly carbonaceous matter retaining entangled in it metallic tin and some slag, it is raked out through the furnace door and put aside for further treatment.

Refining the Crude Tin.—Blocks of crude tin are piled on the hearth of the furnace and the temperature is very slowly raised. The tin melts and flows away; as the tin runs down more pigs are added till sufficient tin has liquated out, when it is tapped into a pot, which is heated by a separate fire. On raising the temperature the residue is also melted down, run into another pot, and cast into moulds.

The tin in the pot is now poled. This operation is much the same as the poling in the refining of copper previously described. A pole of green wood is forced below the surface of the metal, any scum which rises to the surface is skimmed off and thrown back into the furnace. For common tin, the poling is continued for an hour; for refined tin it is repeated several times. Tossing is sometimes used instead of poling; the workman lifts up the molten metal in a ladle and pours it back from a considerable height into the pot.

Treating the Residues.—The slags tapped from the smelting are remelted with lime, and any tin which separates is refined.

The slags or dross drawn from the furnace are stamped and washed to separate the metallic tin which they contain.

The refractory material left after the liquation of tin, and subsequently melted as described, is again liquated; the residue remaining on the hearth, called *hardhead*, is put aside for further treatment, and the tin obtained is refined.

Chemistry of the Process.—The chemistry of the process is very simple. The tin oxide is reduced by carbon—

$$SnO_2 + 2C = Sn + 2CO.$$

The other metals present are also reduced, and dissolve in the tin. These metals form more or less definite alloys, which remain in the tin, and from which it can be liquated. The chemical action of the poling is somewhat uncertain, it is certainly not a reducing action as in the case of copper poling, it may, to a certain extent, be an oxidising action owing to the amount of water vapour given off from the pole, but probably the action in this case and in tossing is mainly mechanical, the agitation favouring the separation of the impurities as a scum.

Smelting Tin in Blast Furnaces.—This has been attempted, but has not been a great success owing to the readiness with which tin passes into the slags. It is, however, carried on in some parts of the Continent and in the Malay Peninsula, whence large quantities of tin are exported to Europe.

Uses of Tin.—Tin comes into the market in several grades of purity. Grain tin, which is in columnar fragments, is prepared by heating the tin ingots to 200° C., and either dropping them from a height or breaking them with a hammer.

Refined tin is a similar quality of tin in cakes or ingots, and common tin is a less pure form.

Tin is used for packing purposes after being rolled into very thin sheets, "tin foil," but its principal use is for the manufacture of alloys.

CHAPTER XVI.

SILVER.

Silver.—Silver is one of the few metals that have been in use from prehistoric times, and which has always been highly valued. It has a silver-white colour, a high lustre, and is capable of taking a fine polish. It is both malleable and ductile, and is a good conductor both of heat and electricity. It melts at 954° C., and is sensibly volatile at higher temperatures. It does not tarnish on exposure to air or oxygen, either dry or wet, but blackens rapidly in air containing sulphur compounds. It does not combine with oxygen at high temperatures, so remains bright when fused in air; but in the molten condition it absorbs a large quantity of oxygen, which it gives out again on cooling, causing what is called *vegetation* or *spitting*.

It dissolves readily in nitric acid or in hot, strong sulphuric acid, but is not attacked by dilute sulphuric or by hydrochloric acid unless it is in an extremely fine state of division. It is attacked by chlorine either cold or in the melted condition, the chloride not being decomposed at a red heat.

Silver Ores.—The silver minerals have been already mentioned. Most silver ores consist of some of these minerals disseminated through a considerable quantity of gangue or rock, and the treatment necessary will vary with the form of combination in which the silver is present.

Ores in which the gangue is mainly quartz or other rock masses, and which are comparatively free from other metallic minerals, are called **milling** ores, since they can be treated by amalgamation in the mill. If the silver is present as metal or chloride, they are **free milling**, because they can be treated at once in the mill; but if the silver is present as sulphide or arsenide, they are refractory, and need roasting before amalgamation. When the silver is present as sulphide or arsenide, with other compounds of the heavy metals, they cannot conveniently be treated by amalgamation, and are called **smelting ores.** The methods of treating silver ores may be classified into :—

(1) *Amalgamation processes*, in which the metal is collected in mercury:—

α. The Mexican or Patio process.
β. The barrel process.
γ. The pan process.

(2) *Chemical or wet processes*, in which the silver is dissolved, and precipitated in various ways:—

α. Augustin process.
β. Von-Patera process.
γ. Ziervogel process.
δ. Claudet process.

(3) *Smelting processes*, in which the metal is collected in lead, copper, or copper regulus.

THE PATIO PROCESS.

This process, which is still in use in Mexico and South America, is one of the oldest methods of treating silver ores by amalgamation. The ores in the districts where it is used are complex, containing metallic silver, chloride, and sulphide, and often a small quantity of gold.

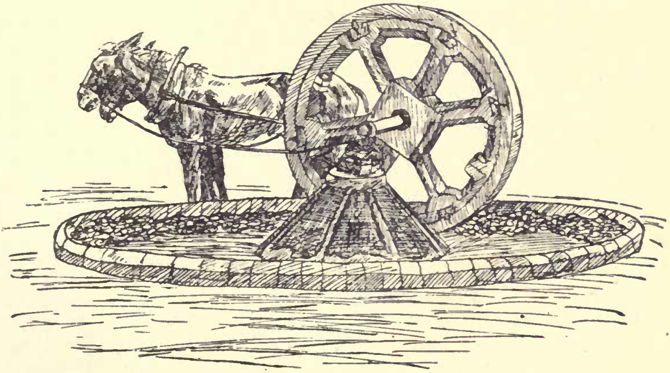


Fig. 52.—Chilian Mill.

The process consists of four stages:—(1) Crushing in the Chilian mill; (2) fine grinding in the arrastra; (3) treatment on the Patio; (4) retorting the amalgam.

(1) The ores are hand-picked and then crushed in the Chilian Mill. This consists of a long wooden arm, carried on a pivot in the centre of a circular paved floor; near one end is a vertical, circular stone or iron wheel free to rotate as it describes a circular path over the floor, and to the projecting end a mule is harnessed.

(2) The fine grinding is carried on in the arrastra, a primitive, but very efficient, machine. A circular floor of stones is very carefully laid and surrounded by a wall of stone; in the centre is a conical elevation to serve as a socket for a vertical shaft, the upper end of which is carried in a strong wooden frame-work. This shaft carries two or four arms, which project beyond the circular wall, so that mules can be attached. Blocks of stone, weighing from 100 to 200 pounds, are placed on the floor, and are attached by thongs of leather to the arms, so that as these rotate the stones are dragged round. A charge of about 1 ton of ore, mixed with water to a thick paste, is introduced, and the grinding action of the stone soon reduces it to a very fine state of division. If the ore contains gold, a little mercury is added. The stones wear out very rapidly, so that the weight of the ore is somewhat increased by the grinding. When the grinding is complete water is added, so as to make thin slimes, which are either dipped or run into slime pits. After about a month's working the amalgam is cleaned out, washed, separated, and distilled for the recovery of the gold and silver which it may contain.

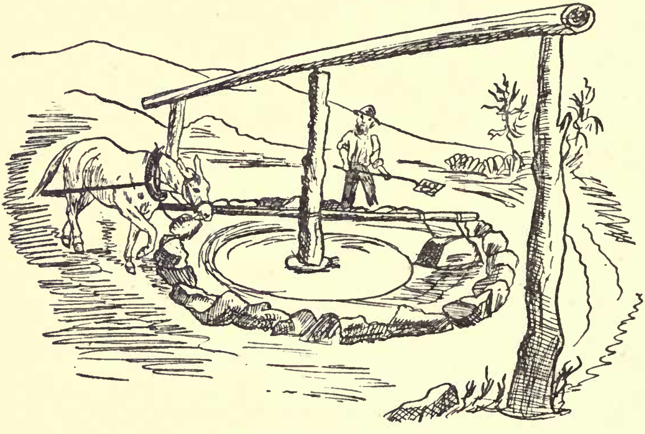


Fig. 53.—Arrastra.

The slimes are allowed to settle and the water is run off, and the material is ready for treatment on the patio.

(3) The patio is the court or yard in which the next operation is carried out; it is carefully paved with stone or wood so as, as far as possible, to prevent loss of mercury. The slimes are

brought to the patio in the form of a thick mud, which is spread into heaps or "tortas," each of which may contain from 30 to 130 tons; a wall of stone set in clay is made round the heap to prevent it spreading, and the mass is left to dry. When it is sufficiently dry the wall is taken down, and the mass is turned over into a regular pile. From 2 to 5 per cent. of salt is scattered over the pile. Mules are then turned on to it and kept walking round (but are allowed to stop for rest at intervals), and the heap is turned with a spade; for a large pile as many as twenty-five mules are required. The treading and spading are repeated for two days; then from ½ to 2 per cent. of magistral, a mixture of sulphates of copper and iron obtained by roasting copper pyrites in reverberatory furnaces, is scattered evenly over the surface of the pile, and the mass is well turned over with spades and trodden; this mixing and treading is repeated daily for three days. Mercury is now added; the full quantity is about 6 to 8 lbs. for each pound of silver present in the heap, two-thirds or three-fourths of this is added at first, the remainder at intervals during the process, and the heap is trodden for two hours. A hot solution of sulphate of copper is then added, and sometimes precipitated copper, and the treading is continued. A considerable amount of heat is evolved by the reactions which take place; if the temperature should become too high, lime or wood ashes or tails are added to cool it, or if the slimes are not too thin, cold water may be used. The treading is repeated daily until tests show the amalgamation to be complete, which may take from fifteen to fifty days according to circumstances.

The mass is taken to washing boxes, mixed with a large quantity of water, and the slimes are washed away, leaving the amalgam. The tails are subjected to an elaborate series of washing operations, for the separation of any amalgam which may have been carried over, and also of rich heavy sulphides which have been unacted on. The loss of mercury is very considerable.

Chemistry of the Process.—The reactions which take place are very complex, and have not been clearly made out. The copper sulphate of the magistral acted on by the salt is probably converted into chloride—

$$CuSO_4 + 2NaCl = CuCl_2 + Na_2SO_4.$$

It is then usually said that this copper chloride acts on silver sulphide, forming cuprous chloride, silver chloride, and sulphur—

$$Ag_2S + 2CuCl_2 = 2AgCl + Cu_2Cl_2 + S;$$

that cuprous chloride acting on silver sulphide forms cuprous sulphide and silver chloride—

$$Ag_2S + Cu_2Cl_2 = 2AgCl + Cu_2S;$$

and then that the mercury acting on this chloride decomposes it, liberating metallic silver which combines with the mercury, forming an amalgam—

$$2AgCl + Hg = 2Ag + Hg_2Cl_2.$$

The metallic copper which has been added re-reduces the mercurous chloride—

$$Hg_2Cl_2 + Cu = CuCl_2 + 2Hg.$$

These reactions may take place to a certain extent, but they certainly do not represent all, even if they do the principal reactions of the process. Silver chloride is not easily reduced by mercury, and it is denied by some authorities that silver chloride is produced at all.

Treatment of the Amalgam.—The amalgam is collected, mixed with fresh mercury, stirred up with water, and any scum which rises to the surface is removed; it is then dried and strained through cloth. The strained amalgam is pressed into mercury bottles, the bottoms of which have been cut off. These are then placed over tubes leading down into a vessel containing water, securely luted, and a fire is built up round them. The mercury distils out and is condensed in the water; the silver is left in the flask as a spongy mass, and is afterwards refined.

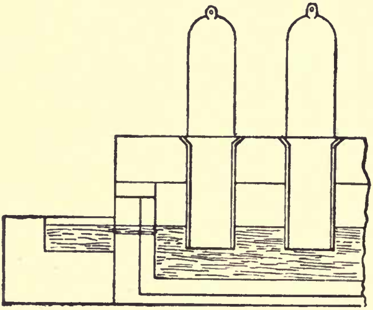


Fig. 54.—Mercury Retort.

BARREL AMALGAMATION.

This methodhas been in use for many years in Freiberg, and has been revived in a modified form in the United States. The ore, which contains sulphides and arsenides, is ground and roasted in a reverberatory furnace; it is then ready for amalgamation.

The barrels are of wood, 4 ft. 6 ins. diameter, and 4 ft. 6 ins.

long. There are two openings at opposite ends of a diameter; the bunghole is 5 ins. in diameter, and the mercury hole is 1 in. in diameter. The barrels are rotated by power, each one being driven separately. A charge of about 2,000 lbs. of ore, and 60 to 150 lbs. of scrap-iron is introduced, a little water is added, the bung is closed, and the barrel is rotated for four hours; it is then stopped and the mercury is introduced—the quantity being roughly about double that of the silver present—together with a little copper sulphate. The barrel is now rotated for about sixteen hours; it is then stopped and a sample is taken and examined; if the operation is not complete the rotation is resumed. If the amalgamation is complete the barrel is filled with water and slowly rotated for an hour to collect the mercury; it is then turned over with the small opening down, an iron tube is inserted, and as much mercury and amalgam as possible is drawn off; another 100 lbs. of mercury is added, the barrel is rotated for an hour, and this is drawn off and mixed with the amalgam. The bung is now removed, the barrel turned over, and the residues emptied out, flowing away down a trough provided with riffle bars, by which most of the amalgam or mercury present is retained, and then into a settling tub, where any mercury that remains is collected; the barrel is washed out and is ready for recharging.

The mercury is strained, and the amalgam is distilled in iron retorts.

The chemistry of the process is simple. The iron decomposes the silver salts present, except sulphides, and the liberated silver dissolves in the mercury. The iron also decomposes mercury salts, liberating mercury.

PAN AMALGAMATION.

This is the form of amalgamation now in most general use, as it is both cheaper and more efficient than barrel or any other form.

The ores are crushed, ground, and, if necessary, roasted. Stamps are generally used for the crushing, and the stamping is done wet, the slimes being run into vats and allowed to settle; if roasting be necessary, these are dried and roasted.

The pan is a grinding and amalgamating apparatus, and is an improvement on the Mexican arrastra. The pan consists essentially of two parts—the pan proper and the muller. The pan is a circular vessel about 5 feet in diameter, the bottom being of cast iron, with a steam chamber beneath, and the sides are of wood or of cast or wrought iron. In the centre of the bottom is cast a conical piece, through which the shaft, S, passes. On the bottom of the pan are arranged dies, D, of cast iron, separated by strips of hard wood. The shaft, S, carries the muller, M, by arms, A. The muller may be a complete ring or separate sections of metal, and to its under surface are attached the shoes, Sh, which are segments of cast iron, spaces being left between them. The muller can be raised or lowered by means of a screw and wheel,

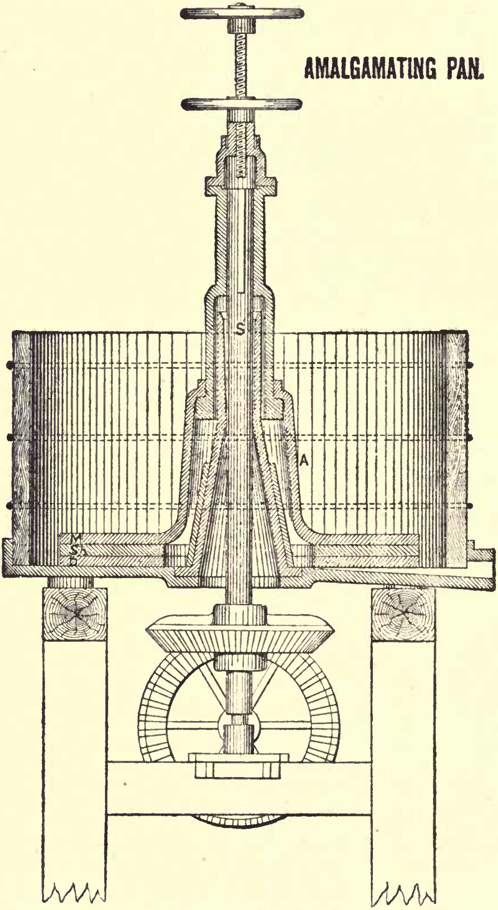


Fig. 55.—Amalgamating Pan (Messrs. Fraser & Chalmers).

and the shaft, S, is driven by gearing placed below the pan. Some water is put into the pan; then the pulp is introduced, and the muller is set in rotation. After a short time the muller is stopped, raised a little, the mercury is put in, and the rotation is resumed at from 60 to 80 revolutions a minute, the muller being lowered till the shoes bear on the dies. The amalgamation takes place rapidly at first, and then slackens, and steam is blown into the steam-chest to facilitate the action. The charge in the pan will vary from 800 to 2,000 lbs., and the operation will last five or six hours.

The action of the pan is simple; the rotation of the muller causes rapid centrifugal currents, which carry the ore and mercury under the shoes, and discharge them at the outer edge; the mixture is thrown up the side of the pan, and is deflected by copper plates arranged for the purpose, and so passes back under the shoes again. In this way the ore is reduced to an extremely fine state of division, and is brought into most intimate contact with the mercury.

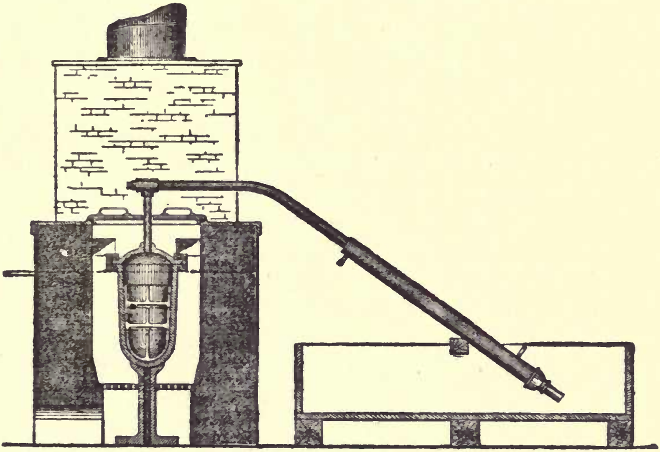


Fig. 56.—Mercury Retort.

Various chemicals are occasionally used to facilitate the action, such as salt, copper sulphate, lime water, potassium cyanide, &c.

Innumerable other forms of pan are used, but the principle of all is identically the same.

When the operation is over a considerable quantity of water is added, and the pulp is run into the settler. The **settler** is a tub, 8 or 9 feet in diameter, provided with a cast-iron bottom and stirring arms; and usually with discharge openings at different heights. The settler is filled and the arms are set in rotation and kept going for about three hours; the light ore is kept in suspension, and the mercury and amalgam settles. The top hole is then opened, and clean water is run in till it flows away nearly clear; the second and third holes are next opened in succession, and the process is repeated; lastly, the mercury plug is removed and the mercury at the bottom is run out. The plug is put in again, and the settler is restarted, a thorough cleaning-out taking place once a week only. The tailings are run through other settlers, to recover any mercury that may have been carried over. The mercury is strained through canvas bag filters, distilled in iron retorts, and the residual silver refined.

Treatment of Slimes, &c.—In many districts the slimes carry enough metal to pay for treatment by concentration.

WET METHODS OF TREATING SILVER ORES.

Of these many have been suggested; they all depend on getting the silver in solution, usually as chloride and then precipitating it. The only process now much used is the von-Patera process, which will be described.

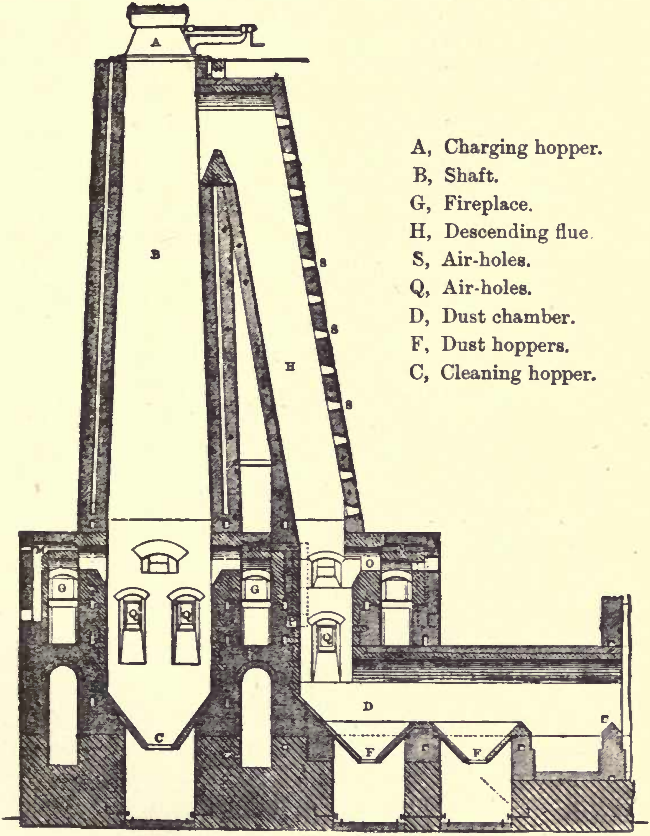
THE VON-PATERA PROCESS.

This process depends on the conversion of the silver into chloride by a chloridising roasting, solution of the chloride in sodium or calcium hyposulphite, and precipitation with sodium sulphide.

The ore is crushed and, if wet crushing has been used, is dried; it must not be crushed too fine (about 20 mesh), as very fine materials are difficult to leach.

Chloridising Roasting.—The ore is next roasted with salt, the quantity of salt used varies, but probably averages 5 to 10 per cent. Various kinds of furnaces may be used; long reverberatory furnaces and rotating cylindrical furnaces, such as the Bruckner calciner, are the forms generally adopted. A furnace very largely used is the Stetefeldt. This consists of a vertical shaft with fireplaces at the bottom, the finely powdered ore mixed with salt is delivered in thin streams at the top into the hot furnace where it meets a hot ascending current of air; the sulphur is thus burnt out and the ore is chloridised by the salt vapours. This furnace is mainly used for roasting sulphurous materials.

Base Metal Leaching.—The ore after roasting will contain the metals mostly as chlorides, but in many cases partly as sulphates, Assuming the ore to be a complex one, containing copper as well as silver, it will be necessary to leach with hot water to remove the base metal, in this case the copper. The quantity of salt must be kept low or the excess will dissolve some of the silver chloride. The chloridised ore is charged into the leaching tank, a wooden tub, 12 to 15 feet in diameter and 5 or 6 feet deep, provided with a perforated false bottom, on which is placed a



A, Charging hopper.
B, Shaft.
G, Fireplace.
H, Descending flue.
S, Air-holes.
Q, Air-holes.
D, Dust chamber.
F, Dust hoppers.
C, Cleaning hopper.

Fig. 57.—Stetefeldt Furnace.

filter bed of fine gravel and above this usually a layer of canvas. The charge of ore is usually sifted in, so that it may not lie too compact, and water, hot or cold, as the case may be, is added from the bottom till the ore is just covered. After a short time this is run off by a pipe which enters under the false bottom and more water is run in. The number of leachings required will depend on the nature of the metals to be leached out, and the solutions may either be preserved or run to waste according to circumstances.

Hyposulphite Leaching.—The residues in the vat are now treated with a 1 to $2\frac{1}{2}$ per cent. solution of sodium hyposulphite, which is allowed to stand some time, then drawn off and fresh solution added until the solution drawn off is free from silver, which may take 12 to 24 hours. The tails left in the tubs should not contain more than about 4 ozs. of silver per ton.

Precipitation of the Silver.—The liquor is run directly from the leaching into the precipitating vats; these are about 8 feet in diameter and 12 feet deep; a solution of calcium sulphide is added to precipitate the silver, stirred well, and the precipitate allowed to settle; the clear liquors are pumped off to be used over again for leaching; the precipitate is mixed with water and run off to filter presses, where the liquid is squeezed out and the sulphide of silver is left in semi-solid cakes.

Treating the Precipitate.—The silver sulphide is dried on the hearth of a reverberatory furnace and roasted till most of the sulphur is driven off. The residue is melted with a little lead and cupelled.

Treating the Base Metal Leachings.—If the base metal leachings contain copper, this may be precipitated by means of iron.

Chemistry of the Process.—This is very simple, the roasting converts the silver into chloride. This is converted by the hyposulphite in the leaching into silver-sodium hyposulphite, which is very soluble—$AgCl + Na_2S_2O_3 = AgNaS_2O_3 + NaCl$; and this, on treatment with sodium sulphide, is decomposed with the re-formation of sodium hyposulphite, or if calcium sulphide is used, the corresponding calcium salt.

$$2AgNaS_2O_3 + Na_2S = Na_2S_2O_3 + Ag_2S.$$

THE RUSSELL PROCESS.

This is a modification of the von-Patera process invented to overcome some difficulties in the application of the latter to certain ores. A considerable quantity of silver is always left in the tailings, especially if the chloridising has not been complete; and if any lead be present in the ore this will be converted into sulphate during roasting, dissolved by the hyposulphite, and precipitated by the sulphide, thus yielding a very poor bullion. The Russell process is meant to overcome these difficulties.

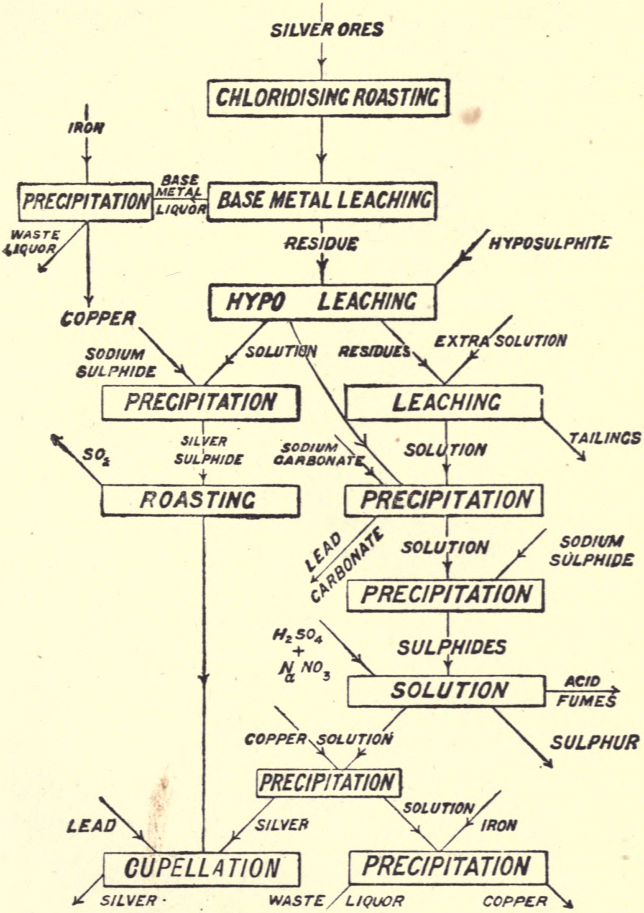


Fig. 58.—Plan of the von-Patera and Russell Process.

The grinding, roasting, and the leachings with water and with sodium hyposulphite are conducted exactly as in the Von-Patera process.

Precipitation of the Lead. — The solution from the hyposulphite leaching will contain the lead in the form of hyposulphite, to this is added a solution of sodium carbonate which precipitates the lead as carbonate. The precipitate is allowed to settle, the liquors are drawn off and the silver in them is precipitated with sodium sulphide; while the lead precipitate is dried and either sold or reduced. The lead carbonate is very pure, and is almost free from silver. When this method of separating the lead is adopted sodium hyposulphite must be used for the solution, not calcium hyposulphite, as any calcium in solution would be precipitated with the lead. This process can be used for treating the complex lead-zinc-silver ores which occur in some districts; the zinc is dissolved in the base metal leaching and may be precipitated as oxide by lime.

Treating with Extra Solution.—The extra solution is a solution of cuprous-sodium hyposulphite made by adding copper sulphate to sodium hyposulphite. This solution has a very powerful solvent action on most silver compounds, though its action on the chloride is less than that of simple sodium hyposulphite.

The material left in the vats after the hypo-leaching is, therefore, treated with extra solution (which is made just before it is to be used, as it does not keep well), till the solution is about 10 inches above the level of the ore; this is run off till it is only about 2 inches above it, the solution which has run off is returned, and so is kept circulating through the ore for some hours. The solution is then run off and the tailings are washed with water. To facilitate the action the leaching vat is best heated with steam pipes.

The solution of copper and silver is precipitated with sodium sulphide; the precipitate is treated with sulphuric acid to which sodium nitrate has been added; the escaping nitric acid is condensed in a tower, and the separated sulphur is used for the preparation of sodium sulphide. The solution of copper and silver is diluted; the silver is thrown down by copper, melted, and refined, and the copper liquor is precipitated by iron.

THE AUGUSTIN PROCESS.

In this process the ore is, if necessary, subjected to a chloridising roasting, and is then leached with a hot, strong solution of common salt. This dissolves the silver chloride and from the solution the silver is precipitated with copper.

THE CLAUDET PROCESS.

This has been described in connection with the treatment of copper ores, p. 153.

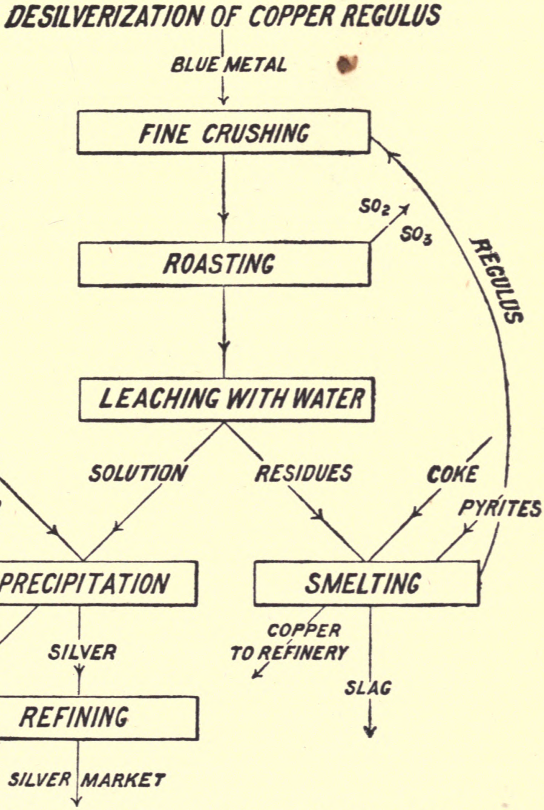


Fig. 59.—Plan of Ziervogel Process for Copper Matt.

THE ZIERVOGEL PROCESS.

This process is not generally applicable to silver ores, but is valuable for the desilverisation of argentiferous copper regulus (blue metal), which is produced in the smelting of argentiferous copper ores.

The matt is very finely ground, and is then cautiously roasted on the hearth of a reverberatory furnace. A complex set of reactions takes place, the copper sulphide is first converted into sulphate and oxide, and, ultimately, completely into oxide. The iron sulphide undergoes the same changes and the silver sulphide is converted into sulphate, which is not decomposed at the temperature of the furnace. When the roasting is complete the charge is drawn, allowed to cool, and leached with hot water, which dissolves the silver sulphate. The silver is precipitated from solution by copper, and the residues from the leaching vats are smelted for copper.

Separation of Silver from Copper.—In the old Freiberg method, which is now replaced by the Ziervogel process, the blue metal was reduced in the usual way, the silver passing into the copper. The copper was melted with lead, the mixture cast into cakes, and these were heated to above the melting point of lead in a special furnace. The lead "liquated" out, carrying with it most of the silver, and was afterwards cupelled.

CHAPTER XVII.

GOLD.

Gold.—Gold has been known from the most remote ages, and has always been very highly valued for purposes of ornament. It has a fine yellow colour, a high lustre, and is almost unalterable in air under ordinary conditions. It is the most malleable and ductile of the metals, and is a very good conductor of heat and electricity. It melts at about 1,045° C., and is somewhat volatile at higher temperatures.

It does not tarnish in air, either wet or dry, nor does it oxidise when heated in air or oxygen. It does not dissolve in any ordinary acid, but is rapidly attacked by a mixture of hydrochloric and nitric acids (aqua regia), or any other mixture which evolves chlorine, gold chloride ($AuCl_3$) being formed. Chlorine, and to a less extent bromine and iodine, attacks gold readily at ordinary temperatures; but as the chlorides, &c., are readily decomposed by heat, chlorine has no action at high temperatures. Gold, if in a fine state of division, is attacked by potassium cyanide in presence of air. Gold chloride in solution is readily decomposed by reducing agents, the gold being thrown down in the form of a black powder, which on ignition passes into the usual yellow form. The precipitant most commonly used is ferrous sulphate, which is oxidised to ferric sulphate and chloride—

$$2AuCl_3 + 6FeSO_4 = 2Fe_2(SO_4)_3 + Fe_2Cl_6 + 2Au\,;$$

basic ferric salts being also precipitated unless free acid be present. Oxalic acid, charcoal, and cuprous sulphide have been suggested; and also metals such as zinc, iron, or aluminium. From its solution in potassium cyanide gold is not thrown down by ordinary reducing agents; many metals, however, precipitate it readily, or it may be thrown down by an electric current.

Occurrence of Gold.—Gold almost always occurs in nature in the metallic condition. Native gold is never pure, but always contains silver or copper and sometimes other metals. It usually contains from 800 to 950 parts in 1,000 of gold, though the quantity may be very much lower, when it is as low as 500 the alloy is white and is called *electrum*.

Gold very frequently occurs disseminated through minerals, such as iron pyrites or arsenical pyrites, but whether in these cases it is present as metal or in some form of combination, is not certainly known, though the former seems the more probable. It also occasionally occurs in combination with tellurium in certain tellurides.

Gold occurs disseminated through sands or gravels, masses or veins of quartz or other rocks, or through other minerals; in any case the quantity of gold necessary to constitute a material an ore is very small.

1. *Shallow Placers or Modern Alluvial Deposits.*—These are sands, clays, and gravels of modern date, usually formed by existing rivers, containing gold. The gold is usually disseminated through the gravels in the form of fine dust or flakes, or in larger lumps called *nuggets.* Nuggets are occasionally of large size. One, for instance, "The Welcome," found in Victoria, weighed 183 lbs. Nuggets always have a rough or "nuggetty" appearance.(?) Alluvial gold is usually associated with heavy black sand (often containing magnetite, ilmenite, and sulphides), and in some localities also with diamonds. Alluvial gold deposits are common all over the world. The river gravels of Scotland, Ireland, Wales, and many parts of Europe have yielded gold, but the richest of such deposits were those of Australia and California, from which very large quantities of gold have been obtained.

2. *Deep Placers.*—These are older gravels (often left by extinct rivers) which have been protected from denudation by overlying deposits, generally a lava or other volcanic deposit. These are of much the same nature as the shallow deposits, but, not being at the surface, must be mined or treated by hydraulicing. They are largely worked in Australia and California.

3. *Vein Deposits.*—In these the gold is disseminated through a mass of non-auriferous matter, either constituting a lode or vein or a rock mass. If the gangue is quartz, the gold free, and sulphides absent, the ore will be free milling, so that the gold can be extracted by mercury; whilst, if the gangue is a metalliferous mineral, the gold cannot be so extracted, and the ore is refractory.

Shallow Alluvial Deposits.—The gold is obtained from these by the simple process of washing, the lighter earthy materials being washed away by running water, and the heavier metal left. Innumerable forms of apparatus have been used for the purpose.

The Pan.—This is the simplest form of apparatus, and has been used from prehistoric times. It simply consists of a shallow vessel of any form, the Australian pan being a flatbottomed iron dish. The pan is immersed in water, and filled with the "dirt" to be washed; it is then held near the surface of running water; the lumps are broken up by hand, and a rotary motion is imparted to it. The light sand and clay is carried away by the water, pebbles are picked out and thrown on one side, and the washing is continued till nothing is left but gold and heavy black sand. The powder is then dried, and the sand is carefully blown away; or the gold may be extracted with mercury. The batea, which is sometimes used, is a conical dish of wood or metal, but preferably the former.

Fig. 60.—Gold Washing with the Pan.

The pan in some form has been used in all countries; it is, however, a very laborious and wasteful method, and is now mainly used by the Chinese.

The Cradle or Rocker.—This was one of the first improvements on the pan introduced in Australia. It consists of a box mounted on rockers. At one end is fixed a hopper, at the bottom of which is a screen perforated with half-inch holes; below which is a baffle board, to throw the material running through to the top end of the box; and across the bottom are fixed riffle bars. The "dirt" is thrown into the hopper, water is poured on, and the cradle is rocked; the rocking breaks up the dirt, the fine material is carried through the holes and away over the low end by the stream of water, whilst the heavy gold and black sand collect on the bottom. The cradle is a better apparatus than the pan, though it requires much more water to work it. With it two men can wash about 4½ cubic yards of gravel a day.

Fig. 61.—The Cradle.

The Tom.—This is the simplest form of sluice. It consists of two boxes set at an incline of about 1 inch to the foot; the lower end of the upper sluice is cut off at about 45°, and is closed by a grating to keep back large stones. The lower box is provided with riffles, to which mercury is sometimes added. The dirt is thrown into the upper box, which is supplied with a stream of water; the fine material is swept through the grating, and the gold and heavy sands are caught by the riffles; the large stones are picked out and thrown on one side. Two men can wash about five times as much dirt with the tom as they can with the cradle.

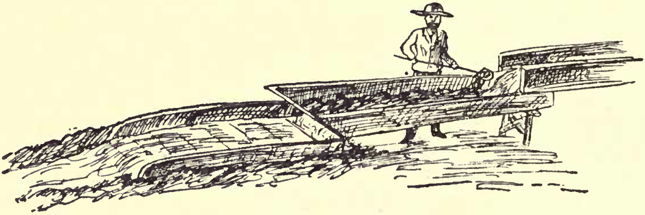


Fig. 62.—The Tom.

The Sluice.—This is the apparatus now almost invariably used for treating alluvial gold deposits. It consists of a set of inclined troughs made of rough boards, usually 16 to 18 inches wide and 18 inches to 2 feet deep. The sluice will be not less than 50 feet long, built up of sections about 12 feet long, each section being widened at the lower end, so as to receive the top end of the next. The whole is placed on trestles, so as to give a suitable slope. The bottom of the sluices are provided with riffle bars; these are placed longitudinally, and are kept in place by cross pieces placed at short distances apart, so that the bottom is divided up into a series of rectangular spaces 2 inches to 4 inches deep. The "dirt" is fed in at the top of the sluice, and is washed down by a continuous stream of water, the length of the sluice being varied according to the nature of the "dirt," ordinary "dirt" requiring 200 to 250 feet for its complete disintegration. When there is much gravel an under-current sluice is used. The end box is left open, but for some feet the bottom is replaced by a grating fine enough to keep back the gravel, and under this is a second sluice at right angles to the first. As the "dirt" is swept down by the water, the gravel passes over the grating and falls out, whilst the finer material passes through into the second sluice.

Mercury is usually introduced into the sluice so as to catch the gold, and frequently the tailings are made to flow over amalgamated copper plates.

About once a week the sluice is cleaned up, the supply of dirt is stopped, the riffles removed one by one, commencing at the top, the mercury is collected, squeezed through canvas filters, and the gold recovered from the amalgam by distillation. The amalgam is also scraped off the copper plates and distilled.

The sluice is a much better form of apparatus than the others, but it is often difficult to obtain a sufficient supply of water for satisfactory working.

Float Gold.—In all gold washings there is apt to be loss by what is called float gold—that is, gold in such a fine state of division that it is carried away on the surface of the running water. This is partially recovered by making the water flow in a very thin layer over blankets or cloths with a thick woolly surface in which the gold becomes entangled.

River Mining.—For obtaining the gold from the gravels in deep rivers many suggestions have been made, such as pumping up the gravel, dredging, &c., but these have not proved very successful. In some cases rivers have been diverted to allow of the gravels in their beds being washed.

Deep Alluvial Deposits.—Deposits of this class are common both in Australia and California. They are gravels deposited from old rivers, or from existing rivers when they ran in old channels, and which have been covered by deposits of lava, volcanic tuff, or other materials, whereby they were protected from denudation. When sufficiently rich these deposits may be reached by levels and shafts and mined in the usual way, but in California they are largely worked by a process called hydraulicing.

Hydraulicing.—This consists in washing down the material by means of water along sluices in which the gold can be recovered. For this purpose a large supply of water is necessary. This is brought by aqueducts, sometimes of very large size, from a considerable distance; indeed, some of the most important engineering works of the time have been the "flumes" for the conveyance of water for hydraulicing. These flumes have often to be carried over valleys or through hills, and the water must be delivered under sufficient pressure. The water is sold to the miners at so much an inch, a miner's inch being "the amount of water which will flow through an orifice 1 inch square in a board 1 inch thick, under a head of from 4 to 8 inches, according to the locality, the time of flow being indicated by the number of hours, thus—a 10-hour inch, 12-hour inch,

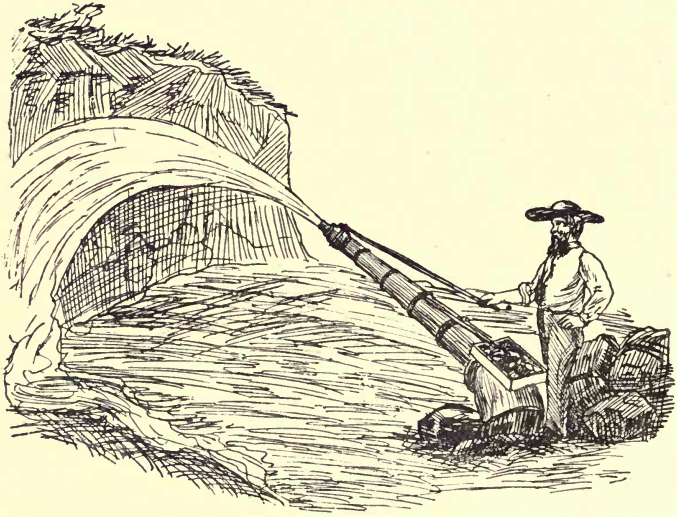


Fig. 63.—Hydraulicing.

&c. A 24-hour inch under a head of 7 inches is about 2,230 cubic feet.

A suitable face of the deposit having been found or made, usually the latter, the jet of water is directed against it by means of a movable nozzle called a *monitor* or *giant*. The nozzle of the monitor may be up to 11 inches in diameter, and the water may be delivered under a head of 200 feet. Such a jet of water acting on the face of the more or less incoherent rock has an enormous disintegrating power, which in some cases may be assisted by blasting.

The sluices are on exactly the same principle as those used for ordinary alluvial washing, but are much larger, and are frequently paved with stone; they are up to 6 feet wide and 3 feet deep, and are often provided with iron riffle bars. The length of the sluice varies according to circumstances, the average being probably about 1,200 feet, though some are a mile or more in length. Mercury is always added The enormous masses of tailings carried down often do a great deal of damage to the country adjoining the rivers into which they are carried, and hydraulicing is now somewhat limited by laws prohibiting the passing of tailings into streams. These tailings always carry small quantities of gold and are often re-washed by hand by the Chinese.

Hydraulicing is the cheapest of all methods of mining, the total cost being from 3 to 6 cents per cubic yard of gravel treated.

The cost of working gold in California by various methods is given in the report of the Californian State Mines for 1889, as—

(1) Auriferous vein deposits, . .	\$3 to \$10	per ton.
(2) Drift mining, . . .	·75 to 4	,,
(3) Miner's pan, . . .	5 to 8	,,
(4) Rocker or cradle, . . .	2 to 3	,,
(5) Sluicing, ...	·75 to 1	,,
(6) Hydraulicing, . . .	1½ to 8 cents	,,

Working Vein Deposits.—The working of vein deposits (or, as these are often called, gold quartz) consists, in addition to the mining, of two distinct operations—(1) Crushing the ore; (2) extracting the gold; though the two operations may be carried on at the same time. The ore may be divided into two classes, which will require quite different treatment. If the gold be free in a rock devoid of sulphides or arsenides, the gold will be readily attacked by mercury; such ore is said to be free-milling. If the ore contains sulphides or arsenides the gold is not attacked by mercury without some preliminary treatment, and the ore is said to be refractory or rebellious. In most of the so-called free-milling ores (though not in all) part of the gold is present in a form which will not amalgamate, and the tailings after the amalgamation of such ores are often worth treatment. It very frequently happens that the ores near the surface which have been exposed to oxidation are free-milling, whilst at greater depths they become refractory.

AMALGAMATION OF FREE-MILLING ORES.

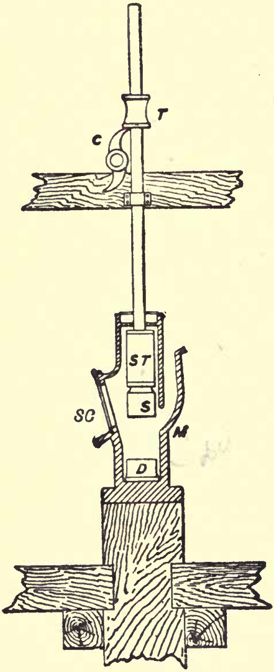


Fig. 64.—Stamp Battery.

T, Tappet.
C, Cam.
S T, Stamp.
S, ,, head.
D, Shoe.
M, Mortar.
S C, Screen.

Crushing.—The ores are first broken up, either by hand or in a stone breaker, and are then finely crushed (at the same time that they are being treated with mercury to recover a portion of the gold) in a stamp battery. The usual form of stamp has been described on p. 82. Each battery usually consists of five stamps, weighing from 650 to 950 lbs., and driven at from 30 to 100 blows a minute, with a drop of about 5 to 12 ins. The screen is of sheet-iron, perforated with holes or slits about $\frac{1}{24}$ to $\frac{1}{40}$ of an inch in diameter, for holes, or wide, for slots; though they may vary either above or below this in special cases. The stamps must not be allowed to fall in consecutive order, or the ore would all be driven to one end of the battery; various orders of fall are adopted, that generally used in California being—1, 4, 2, 5, 3.

Amalgamation in the Battery.—When amalgamation is to take place in the battery, a series of amalgamated copper plates are fixed in the mortar. These plates are of annealed copper, they are amalgamated with mercury and then rubbed with a little silver amalgam.

The pulp as it leaves the mortar is made to flow over similar amalgamated plates, placed in front of the screens, and often 12 to 14 ft. long. To aid in catching float gold, swinging amalgamated plates are used; these are convex, and are suspended so that their lower edges are just below the surface of the water, thus any float gold carried down must come in contact with them.

In addition to the use of amalgamated plates, mercury is usually fed into the mortar, and often a little potassium cyanide is added to facilitate the amalgamation.

The stamps are periodically stopped for a clean up, two batteries being usually cleaned at the same time; the amalgam is carefully collected, and that which is adherent to the plates is wiped off. The whole is mixed with fresh mercury in a barrel or pan or by hand, and washed to remove intermixed materials, such as pyrites, sand, &c. It is then squeezed through canvas, the liquid portion is used again, and the pasty amalgam is distilled in an iron retort. The residue from the retorts is melted in crucibles, and is afterwards refined.

There is considerable loss of mercury in battery amalgamation; the mercury "sickens" or becomes converted into fine black powder, and in this condition is carried away; this is produced by some chemical action, and is caused by the presence of certain minerals in the ore, the most injurious probably being arsenical pyrites and stibnite. There is also loss through flouring, which is simply the breaking up of the mercury by mechanical means into very fine particles. In California the loss is about ½ oz. to 1 oz. of mercury for each ton of ore treated.

The whole of the gold is never obtained; there will almost always be loss from the presence of unamalgamatable gold, which will pass into the tailings, and often also free gold, which for various reasons will not amalgamate but will be carried away; and also from loss of amalgam in the tailings. What is called "rusty gold" sometimes occurs; this seems to be gold which is covered by a very thin film of oxide of iron or other substance, which prevents the action of the mercury.

Amalgamation in Pans.—The finely-crushed ore or the tailings from the battery are sometimes amalgamated in pans. The pan used is exactly the same as that already described for the treatment of silver ore, and the method of working is much the same. The pan is not very largely used for the treatment of gold ores.

Many other systems of amalgamation have been suggested, and are in use for special classes of ore and in some localities.

CHLORINATION.

This method, which was first proposed by Plattner, consists in treating the ore with free chlorine, which converts the gold into gold chloride; this being soluble, can be dissolved out in water, and the gold can be precipitated from the solution by any of the usual means. It is only applicable to the direct treatment of free-milling ores, refractory ores requiring a preliminary roasting.

The Plattner Process.—The apparatus required consists of a chlorinating vat, a chlorine-generating apparatus, a settler, and a precipitating vat.

The chlorinating vat is a tub, about 7 feet in diameter and 3 feet deep. It is provided with a perforated bottom about 1 inch above the real bottom; on this is laid a filter bed of pebbles, 6 inches or so in thickness; and over this a thin wooden disc, fitting tightly and perforated with a large number of small holes, or a filter cloth stretched tightly over a well-fitting wooden frame. There are two pipes leading to the space beneath the false bottom, and the vat is provided with a tightly-fitting lid.

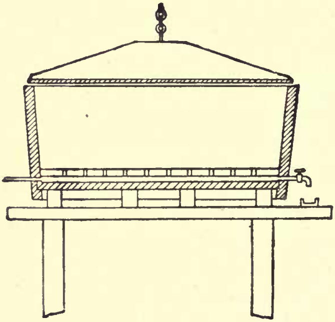


Fig. 65.—Chlorinating Vat.

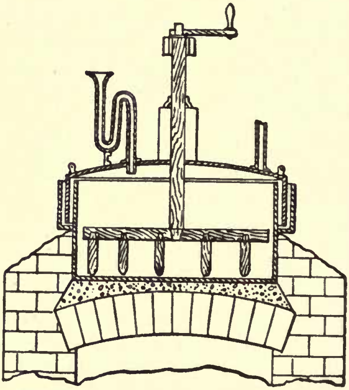


Fig. 66.—Chlorine Generator.

The chlorine generator is a lead vessel, made by "burning," no solder being used, the joint between the upper and lower portions being made tight by a water seal, and through the lid passes a vertical shaft carrying a stirrer—the whole being placed in a sand bath over a fireplace, so that it can be gently heated. A wash bottle is also provided to keep back any hydrochloric acid that may be carried over. To generate the chlorine about 20 lbs. of manganese dioxide and 20 to 24 lbs. of rock salt are introduced, and the cover is put in place; about 35 lbs. of sulphuric acid diluted with water is run in by the funnel and pipe; chlorine is evolved at once; when the action slackens heat is applied.

The settling and precipitating tanks are circular tubs, usually without covers; the latter is often provided with a hammer, by which it can be struck, so as to shake off any gold which may adhere to the sides. The tubs are carefully coated inside with a layer of pitch or other similar material.

The ore, which should be as free as possible from slimes, and should have been passed through a sieve with about sixty holes to the inch, is damped with water, and lightly charged into the chlorinating vat, the usual method of charging being to sift the damped ore in through a coarse sieve. When the vat is full to within 6 or 8 inches of the top, the cover is lowered and luted on with clay or sand, the plug-hole in it being left open. The chlorine is now slowly admitted by a pipe beneath the false bottom of the vat; and, as soon as all the air is expelled and chlorine is escaping freely from the plug-hole, this is closed, and the supply of chlorine is stopped. The whole is left at rest for from 24 to 48 hours, more chlorine being supplied should it be found, on opening the plug, that there is not a large excess present.

The excess of chlorine is then blown out; the lid is taken off, and water is added, either from above or below; and the liquor is run off through a filter bag into a settling tank, where any sand, &c., carried through settles. The last leachings are not mixed with the first, but are stored in separate tanks, to be used for the first leachings of the next charge. The liquid, having settled in the settler, is drawn off into the precipitating tank, a solution of ferrous sulphate is added in sufficient quantity to remove all the chlorine which is in solution and to precipitate the gold; and the whole is well mixed by means of wooden stirrers. The precipitate is allowed to settle, the waste liquor is siphoned off, and a fresh quantity of gold solution and ferrous sulphate is run in. When the tubs are to be cleaned up the liquors are siphoned off, the precipitated gold is collected, washed with acid to remove basic iron salts, dried, and fused in blacklead crucibles. "The bullion will be 920 to 990 fine, the alloying metals consisting chiefly of iron and lead"

The chemistry of the process is simple; the gold is converted into chloride, which is dissolved out, and is decomposed by ferrous sulphate; other metals are to a large extent attacked and dissolved. Reducing agents, such as ferrous sulphate, absorb a large quantity of chlorine, and often stop the chlorination completely; alkalies, such as lime, &c., also absorb chlorine very readily.

Modifications of the Chlorination Process.—Very many modifications have been suggested. In the Mears process the chlorination is conducted in barrels, the chlorine being pumped in, so that the action takes place under pressure. In the Thiess process a barrel is used, but the chlorine is liberated in the chlorinating barrel by the action of acid on bleaching powder ($CaOClCl + H_2SO_4 = CaSO_4 + H_2O + 2Cl$). In the Newbery-Vautin process pressure is obtained by pumping in air; in the Pollok process water pressure is used; and in Brown's process a vacuum is formed in the barrel before admission of the chlorine.

The Cyanide Process.—This is one of the most recent of the gold-extracting processes, and in its present form is known as the MacArthur-Forrest process. It consists in treating the ore with a ·4 to 1 per cent. solution of potassium cyanide. The ore is crushed, passed through a sieve of about 40 mesh (if tailings are being used, they will be sufficiently fine), and transferred to a large vat having both a false bottom and a filter bed. These vats are now often made very large—up to 40 feet in diameter. The ore is covered with the cyanide solution, allowed to stand twelve to twenty-four hours; the solution then drawn off, the residue washed with a weaker solution of cyanide and then with water; the weak solutions are used over again, and the strong solution is passed through boxes containing zinc turnings. The zinc is dissolved and the gold is precipitated on the residual zinc in the form of a black powder. The zinc trays are cleaned up about twice a month, and the residue is washed to separate the gold from the residual zinc, care being taken to remove as little zinc as possible. The precipitate is dried and fused in crucibles. A coarse bullion is thus obtained, and also slags; the latter always contain gold, and are treated by fusion with lead and cupellation.

This process is capable of treating ores containing gold in a refractory condition, and can, therefore, be applied to the tailings from battery amalgamation when these contain enough gold to be worth treatment. The process is not applicable to ores containing much copper, unless this be first removed by chloridising, roasting, and water leaching. If the ores have been roasted or exposed to the air, so as to allow the sulphur to oxidise, they may contain free acid, which would decompose some of the cyanide; in which case, therefore, lime is usually added.

TREATING REFRACTORY ORES.

Ores of this class cannot be treated directly, either by amalgamation or by chlorination, though many of them can be treated by cyanide, and all by smelting methods. Most refractory ores can be amalgamated or chlorinated after roasting, if the roasting be carried so far as to expel all the sulphur; but if the ores are very sulphurous, this is both expensive and tedious. If ores contain hydrated oxide of iron, it is always advisable to dehydrate them before chlorination, as chlorine readily attacks hydrated oxides. Chloridising roasting is not advisable if it can be avoided, as gold is apt to be volatilised.

Pyritic Smelting.—The treatment of very pyritous gold ores by roasting and chlorination or amalgamation is troublesome and expensive. If the pyrites be charged into a blast furnace, part of the sulphur will be volatilised, leaving mainly FeS; and, if air be plentifully supplied, a good deal of this may be oxidised into ferrous oxide, which will slag off; while a regulus of ferrous sulphide will remain, in which the gold will be concentrated. For very rich ores no coke whatever is required. The cost of the process is small, and a rich matt can be obtained, which can be treated by any of the ordinary processes.

Ordinary Smelting Methods.—Refractory ores are sometimes mixed with lead or copper ores and smelted in the usual way, the gold being concentrated either in lead or in copper regulus.

Concentration of Ores or Tailings.—Very poor ores or tailings are frequently concentrated, the concentrates being afterwards treated by any of the methods described. The principles of concentration have been described on p. 83.

REFINING AND PARTING.

The bullion obtained by any of the processes will need refining or parting. That obtained from the chlorination processes will be nearly free from silver, and will only need refining; whilst that from amalgamation or cyanide processes or native gold will probably contain silver and will need parting.

Refining.—The method of refining will depend much on the fineness of the bullion and the nature of the impurities present. Very poor bullion, when the alloying metal is other than silver, or when in addition to silver large quantities of base metals are present, is best cupelled with lead. Purer bullion may be refined by melting under borax, with or without the addition of nitre.

The refined gold may be "toughened" by the addition of ammonium chloride or treating with gaseous chlorine at a high temperature.

Parting.—When the gold is alloyed with silver the process of separation is called parting. The methods in most general use are those by means of nitric or sulphuric acids. If alloys of gold with silver, containing not more than 33 per cent. of gold, are treated with nitric acid (sp. gr. 1·2) or with hot sulphuric acid, the silver is completely dissolved out, and the gold is left in the form of a black mass which, on ignition, passes into the usual yellow form. If the alloy is richer than 33 per cent. it is necessary to reduce it to below this by melting it with silver. As the usual proportions are 3 parts silver to 1 of gold, the process is often called *inquartation.*

Parting with Nitric Acid.—The bullion is melted with the necessary amount of silver, and is granulated by pouring into water. The granulated metal is heated in glass or platinum vessels with nitric acid of about 1·2 specific gravity

$$(6Ag + 8HNO_3 = 6AgNO_3 + 4H_2O + 2NO);$$

the insoluble residue is allowed to settle, the hot liquid is drawn off, the gold is washed, thoroughly dried, and melted. The silver solution is diluted, and the silver precipitated as chloride by the addition of common salt

$$(AgNO_3 + NaCl = NaNO_3 + AgCl).$$

The chloride is washed and decomposed by metallic zinc

$$(2AgCl + Zn = 2Ag + ZnCl_2).$$

Parting with Sulphuric Acid.—The bullion is, if necessary, melted with silver and granulated as for the nitric acid process, and is then boiled with sulphuric acid (sp. gr. 1·76) in cast-iron pots; the silver dissolves and the gold is left

$$(2Ag + 2H_2SO_4 = Ag_2SO_4 + 2H_2O + SO_2).$$

The residual gold is washed, dried, and fused. The solution of silver sulphate is crystallised, and is then treated with a hot concentrated solution of ferrous sulphate, which precipitates the silver ($Ag_2SO_4 + 2FeSO_4 = Fe_2(SO_4)_3 + 2Ag$), any undecomposed silver sulphate being afterwards reduced by metallic copper; or the solution may be heated and the silver precipitated by means of copper, the copper being afterwards recovered by precipitation with iron.

Miller's Process.—This process consists in treating the molten alloy with chlorine gas, the gold is not attacked, gold chloride being decomposed at high temperatures, but silver and the base metals are converted into chlorides and form a slag which rises to the surface. The chlorine is generated by any of the ordinary processes and is passed through the gold, kept molten in a crucible, by means of a porcelain tube. The silver chloride retaining some gold, which must be separated by after treatment, rises to the surface and is removed. This process also toughens the gold by the removal of arsenic, antimony, and bismuth, traces of which may be present. The gold produced by this process has a fineness of about 995, the impurities being silver and copper.

GILDING.

(1) *Gilding with Gold Leaf.*—Gold is hammered out into extremely thin films, an adhesive size is spread on the surface of the article to be gilded, and the gold leaf is spread evenly on this; this is mainly used for non-metallic articles.

(2) *Fire Gilding.*—This is only used for metallic articles. An amalgam of gold is thinly spread over the article which is to be gilded, and this is heated to redness so as to volatilise the mercury. A similar method is used for gilding pottery and porcelain; the design to be gilded is painted on with a mixture containing gold chloride, the article is fired, the gold is left in a finely-divided condition and is afterwards burnished.

(3) *Electro-Gilding.*—See p. 228.

CHAPTER XVIII.

MERCURY.

Mercury or Quicksilver.—This is the only metal liquid at the ordinary temperature, its melting point being about – 39° C., it boils at about 360° C. Its specific gravity is about 13·6. It does not tarnish in air at ordinary temperatures; when heated to 250° C. it combines with oxygen forming red oxide, which is, however, completely decomposed on further heating to 270° C. It is dissolved readily by nitric acid or by hot sulphuric acid, while hydrochloric acid has but little action on it. Its compounds are readily decomposed by zinc, iron, and other metals, and by ordinary reducing agents, the metal being separated; they are all volatile, with or without decomposition, all the oxy-compounds being decomposed and yielding metal.

Mercury dissolves most metals readily, forming alloys called amalgams, these often remain disseminated through the liquid metal and can be separated by squeezing through canvas or other filtering substances; they are all decomposed by heating, the mercury being volatilised and the other metal left.

THE AIR-REDUCTION PROCESS.

The only important ore of mercury is Cinnabar, the sulphide, HgS (often mixed with the free metal); this is readily reduced by being heated with free access of air ($HgS + 2O = Hg + SO_2$), oxide of mercury not being able to exist at high temperatures. This method has been utilised in most mercury districts, the forms of plant used, however, varying very much. The plant consists essentially of two parts—(1) the furnace, in which the reduction takes place, and (2) the condensers, in which the volatilised metal is condensed. Of the older types used in Europe two will be described, and also two of the more modern forms of plant used in the United States.

The Idrian Process.—In this process the furnace consisted of a shaft furnace, with a fireplace (Fig. 67, A) at the bottom, the shaft being crossed by three perforated arches on which the ore being treated was put. Large fragments of ore were piled on the first arch, *a*, smaller fragments on the second, *b*, and on the third, *c*, mercury dust, &c., from the condensers, either contained in earthen vessels or moulded into blocks by means of clay or other similar substance. The heated products of combustion passing upwards mixed with a large excess of air effected the decomposition of the sulphide, and the mercury distilled out and was passed through a series of condensing chambers, C. The greater portion of the mercury was condensed in the two first chambers, in the others was collected a quantity of mercury in the form of dust mixed with other materials, which was redistilled. To ensure complete condensation a stream of cold water was made to flow over the inclined plates in the last chamber, D. This process is intermittent, the operation having to be stopped after each charge to allow of the furnace being emptied and recharged. Hahner's continuous process was invented to overcome this difficulty. The furnace is a cylindrical kiln fed at the top by a charging hopper, and provided at the bottom with a grate, the bars of which could be withdrawn so as to allow the spent charge to be removed. The furnace is charged with a mixture of the ore and charcoal. As with continuous working the condensing chambers are apt to get very hot, they are covered externally with iron plates, and are kept cool by a water spray.

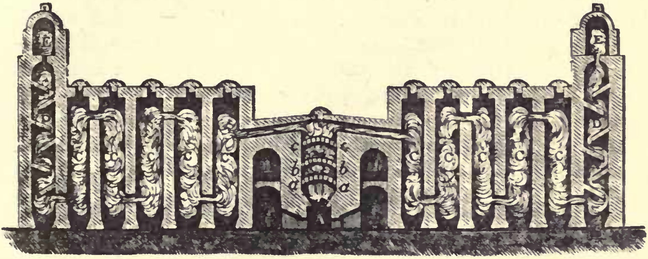


Fig. 67.—Idrian Furnace.

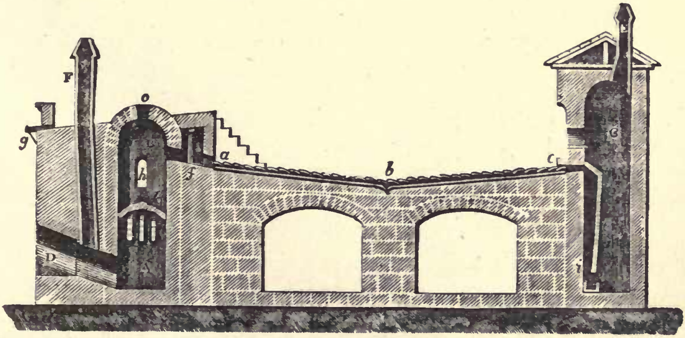


Fig. 68.—Aludel Furnace.

The Almaden Process.—Whilst the principle of this is exactly the same as that of the Idrian process, both the furnace and the condensing arrangements are different.

The furnace consists of a circular kiln crossed by a perforated brick arch (Fig. 68, *k*), on which the ore is piled, rich ore being placed at the bottom, usually on a layer of pebbles, and the poorer ore and slimes above. The fire, either of wood or coal, is made in the space, A, below the arch. The vapours pass away at the top of the kiln into the condensers, which consist of about eight rows of clay aludels, *a*, *b*, *c*. These aludels are pear-shaped vessels with openings at each end, the pointed end of the one is inserted in the larger end of the other, and the joints are luted as securely as possible with clay. The aludels are arranged on two inclined planes sloping towards the middle; those on the plane sloping from the furnace are provided at the lowest point with an opening stopped with sand by which condensed mercury runs out into a gutter and thence to a receiver.

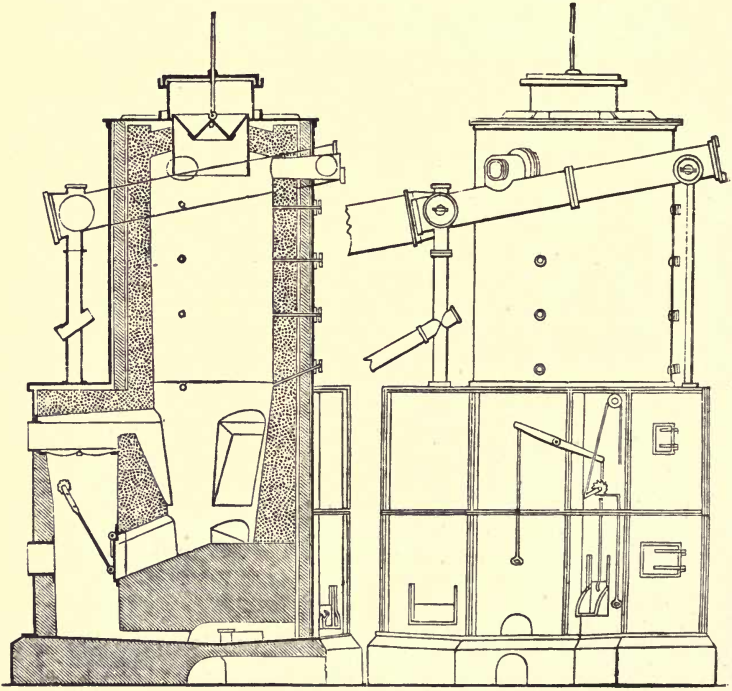


Fig. 69.—Californian Furnace.

The Continuous Californian Process.—As a type of the process now in general use in California for treating coarse ores that at New Almaden may be taken. The furnaces are of what is called the "Exeli" type, small shaft furnaces fired from outside. The upper part of the furnace is cylindrical, and is provided with a charging hopper for introducing the ore which is mixed with a small quantity of charcoal. The lower part of the furnace is conical, and it is provided with three fireplaces on which wood is burnt, between them are three openings for drawing the spent charge. The vapours are passed through brick condensers, then through large iron pipes surrounded by water, and finally away by a long flue.

The Granitza Furnace.—This is used at New Almaden for treating fine ores.

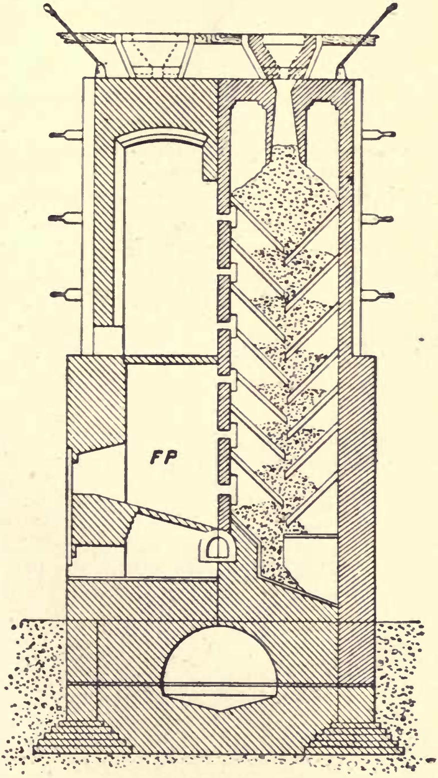


Fig. 70.—Granitza Furnace.

The furnace consists of a set of chambers provided with a series of sloping shelves over which the ore slides in its passage from a charging hopper at the top to the discharge opening at the bottom, the flame and products of combustion passing into the ore chamber by openings under each shelf, thence through a dust chamber, and away to the condensers. A furnace of this type will work about 36 tons of ore in 24 hours, 1 ton of spent ore being drawn each hour.

The Lime Process.—This process, at one time used in the Palatinate, depends on the decomposition of cinnabar by lime, the reaction being ($4HgS + 4CaO = CaSO_4 + 3CaS + 4Hg$); from the form of furnace used it is sometimes called the gallery process.

The ore mixed with lime is charged into earthen retorts which are arranged in rows in a furnace, receivers are attached, the fire is made up, and the mercury distils out.

Iron Process.—Attempts have been made to decompose cinnabar by heating it with iron ($HgS + Fe = FeS + Hg$), but the process has never come into practical use.

Purification of Mercury.—The mercury is purified by straining through fine canvas or chamois leather and sometimes by redistillation; on the small scale mercury is readily purified by warming with nitric acid.

Uses of Mercury.—Mercury is largely used in the extraction of gold and silver from their ores, in the silvering of mirrors, and for other purposes. Sulphide of mercury is prepared as a pigment known as vermilion, other compounds are used in medicine.

Amalgams.—Alloys of mercury with other metals are called amalgams.

CHAPTER XIX.

ALLOYS.

Alloys.—When metals are melted together they, in many cases, form bodies which differ in many respects from their constituent metals. Such substances are called alloys, and many of them are of great value in the arts. A great deal of study has been devoted to the subject of alloys, but many points concerning them are not yet fully established. There is no doubt, however, that solid alloys are not all of similar nature.

Some few are probably definite chemical compounds, in which the constituent metals are united in atomic proportions; many are probably solidified solutions of one metal or of one of the above-mentioned compounds in another; some are probably mere mechanical mixtures of two metals or of two definite alloys; whilst still another set are probably solutions or mixtures of allotropic modifications of the metals.

Properties of Alloys.—Owing to the varied nature of alloys, it might be expected that the relationship existing between the properties of the metals and those of their alloys would be very different in different cases; but a few are so general that they may be considered as being typical.

The melting-point of an alloy is almost invariably lower than the mean of those of its constituents, and is very frequently below that of the most fusible. The specific gravity of the alloy is very rarely the mean of that of its constituents, but it may be either higher or lower. The colour is often very different from either of its constituents; the tensile strength is usually greater than the mean of its constituents; and the alloy is very frequently harder than either of the metals which it contains. Nearly all the alloys have a metallic lustre, and in physical properties are decidedly metallic. They are frequently less oxidisable than their contituents, but when acted on by acids or other reagents they dissolve as mixtures of metals, except that an insoluble metal may protect a soluble one from solution. Most alloys are much worse conductors of electricity than their constituents.

Preparation of Alloys.—Alloys are usually made by melting together the constituent metals, but great difficulties are sometimes experienced when these have very different melting-points, or when one of them is volatile. Prof. Spring has proved that alloys can be prepared by the action of very great pressure on the finely-divided metals, and alloys can also be made by the simultaneous deposition of two metals by an electric current.

Liquation.—Many metals which mix readily in solution tend to separate on cooling. This is well shown with lead and zinc or lead and copper. In either of these cases the metals will mix in the liquid condition, but if the mixture is allowed to cool slowly the metals separate almost completely, the lighter one rising to the top and the heavier sinking to the bottom; the separation is, however, never quite complete, each metal retaining a small and definite quantity of the other in solution. If a mixture of the metals above mentioned be quickly cooled, an apparently homogeneous alloy may be obtained; but if this be heated to the melting-point of the metal which melts most easily, the most fusible metal will be melted or liquated out.

The same phenomenon is seen with other alloys, most of which, on slow cooling, tend to separate—either one of the metals or compounds or mixtures tending to separate. In bronze casting tin tends to separate and to rise to the top of the moulds, and many other similar cases are known. Alloys which contain more than two metals behave in a similar way, but the laws according to which separation takes place have not been fully investigated. As cooling goes on, definite alloys tend to separate, leaving a mixture of indefinite composition, which, therefore, solidifies last. When metals contain small quantities of other metals, the tendency is for the metal to reject these on solidifying, so that the last solidified portions are often the most impure.

The behaviour of each combination of metals must, however, be studied independently, for no absolutely general rules can be laid down.

COPPER ALLOYS.

Copper forms a large number of alloys, many of which are of great industrial value.

The Copper-Zinc Series.—These alloys, generally known as brass, are of great importance. The melting point of brass is below the mean of that of its constituents, it casts readily, and has a yellow colour, the shade varying with the proportions of the metals; some varieties are malleable and ductile. It is harder than copper, and therefore stands wear better. Copper and zinc seem to unite in all proportions and to show little tendency to liquate, but probably definite compounds, such as Cu_2Zn_3 and $CuZn_2$, exist. Brass is denser than the mean of its constituents.

Among the more important brasses may be mentioned :—

Copper, 90 per cent.; zinc, 10 per cent.—This is called "red brass" because after pickling in sulphuric acid the surface has a reddish colour.

Copper, 80 per cent.; zinc, 20 per cent.—This is used for making boiler and other tubes to be drawn cold, it has a yellow colour, breaks with a buff-coloured granular fracture, and has a tenacity of about 14 tons. It is malleable, and can be rolled or hammered into thin sheets = Dutch metal.

Copper, 70 per cent.; zinc, 30 per cent.—An alloy very frequently used for boiler tubes, it is very similar to the above, its tenacity is a little less, but it is suitable for working cold, rolling into leaf, and many other purposes.

Copper, 66·6 per cent.; zinc, 33·4 per cent.—English standard brass used for similar purposes; the fracture is yellower and finely crystalline, the tenacity is less, but it will work cold and can be forged or drawn into wire.

Copper, 60 per cent.; zinc, 40 per cent.—Muntz metal has a yellow colour, breaks with a granular fracture, can be rolled hot, and is largely used for ship sheathing.

Copper, 50 per cent.; zinc, 50 per cent.—Common yellow brass. It has full yellow colour, breaks with a crystalline or granular fracture, cannot be rolled or hammered, but makes good castings. Makes a good strong solder.

Copper, 35 per cent.; zinc, 65 per cent.—White brass. It has a silver white colour, breaks with a conchoidal fracture, is very brittle, and is used as a solder.

The addition of a little iron to brass makes it harder and stronger. Delta metal is made by dissolving iron in zinc, and using this to make the brass. The metal has a yellow colour, is hard, resists corrosion very powerfully, and has a tensile strength of 30 to 35 tons, according to its composition.

Brass Solders are used for soldering brass and copper; they are simply brasses containing from 34 to 50 per cent. of copper. The metal is cast into cakes, these are heated and then powdered in an iron mortar; the alloys being brittle at high temperatures.

Preparation of Brass.—Brass is now always prepared by the direct process. Copper is melted in a crucible, zinc added and stirred in in small pieces at a time, and when all is added, the metal is stirred and poured into the moulds. Blacklead crucibles are now almost always used, and are made up to 200 lbs. capacity. As there is always a loss of zinc by volatilisation, this must be allowed for, the allowance in the case of 70 and 30 brass being about 2 per cent. of the weight of the brass to be made.

For brass which is to be used for cold working, only the purest metals must be used; the copper must be free from arsenic, antimony, and bismuth, and the spelter from iron and lead.

Brass was known before the discovery of zinc, and was made by melting together copper, charcoal, and calamine; such brass was called calamine brass.

The Copper-Tin Series.—These alloys, known as bronzes, are of great importance. The usual alloys made are the following:—

Copper, 90 per cent.; tin, 10 per cent., commonly called gun-metal, as it was used for casting ordinance when bronze guns were used. It has a grayish-yellow colour, breaks with a granular fracture, has a tenacity of about 12 tons, and is hard and elastic, on cooling it shows marked liquation into two alloys, the one poorer in copper tending to rise to the surface.

Copper, 80 per cent.; tin, 20 per cent.—Bell metal. Alloys of this type, but often containing more copper, are used for casting bells; they are hard, fine grained, strong, and sonorous. When heated to redness and suddenly cooled the metal becomes soft, but is hardened by slow cooling. It is malleable at high temperatures.

Copper, 66·6 per cent.; tin, 33·4 per cent.—Speculum metal. This alloy, to which a little arsenic is often added, is silver white, brittle, breaks with a conchoidal fracture, and takes a fine polish; it was used for making mirrors or specula for telescopes and other instruments.

Two definite compounds of copper and tin seem to exist; most of such alloys consisting of mixtures of these with one another, or with one of the metals.

$SnCu_3$. Copper, 61·79; zinc, 38·21. It has a bluish colour, has the highest density of any member of the series, is very brittle, so that it can be pounded in a mortar, and has the lowest electric conductivity.

$SnCu_4$ is very similar.

Preparation of Bronze.—Bronze is always made by melting together the constituent metals, either in crucibles or on the hearth of a reverberatory furnace. Tin is not volatile, so that there is no fear of loss if the metals be protected from oxidation.

Phosphor Bronze.—This is bronze containing a small quantity of phosphorus. This alloy is very similar to bronze in most respects; it has a high tensile strength (36 tons), is very durable, and resists corrosion.

The phosphorus is usually first alloyed with tin. The phosphor-tin thus alloyed may contain up to 20 per cent. of phosphorus; it is silver-white and very brittle, and a portion of this is added to the copper in place of tin.

Among other copper alloys may be mentioned **German silver** an alloy of copper and nickel, containing 20 to 30 per cent. of the latter; and **aluminium bronze** or aluminium gold, containing about 90 per cent. of copper and 10 per cent of aluminium.

Copper and Antimony.—An alloy of 50 per cent. of copper and 50 per cent. of antimony has a fine violet colour, and is known as regulus of Venus.

TIN AND LEAD ALLOYS.

Alloys of tin and lead are harder than lead; they are malleable and ductile. Pewter contains 80 per cent. of tin and 20 per cent. of lead. Soft solders are similar alloys, containing from two parts of tin and one part of lead to two parts of lead and one part of tin.

Britannia metal, which is largely used for the manufacture of teapots and other articles, is an alloy of tin and antimony, containing about 90 per cent. of the former and 10 per cent. of the latter. **Type metal**, used for casting printers' type, contains about four parts of lead to one part of antimony, or for small types three parts of lead to one part of antimony. An alloy containing 50 per cent. of lead and 50 per cent. of bismuth melts at 160° C.; whilst one of 20 per cent. of bismuth, 40 per cent. of lead, and 40 per cent. of tin melts below 100° C. These alloys are called "fusible metal."

GOLD ALLOYS.

The most important of these are the alloys used for coinage. A gold alloy is not usually said to contain so much per cent. of gold, but to be so many carats fine, a carat being one-twenty-fourth, so that fine gold would be 24 carat; whilst a 12-carat gold would contain 50 per cent. of gold. The standard gold of this country is 22 carat—that is, it contains $\frac{22}{24}$, or 91·66 per cent., of gold, the alloying metal being copper. The Australian coinage is of the same standard as the British, but that coined at Sydney, being alloyed with silver, is much paler in colour. The standard used in most European countries is 90 per cent. of gold, whilst the Spanish standard is 87·5.

22-carat gold is the richest used in jewellery; 18-carat—$\frac{18}{24}$ or 75 per cent. of gold—is very largely used; 15-carat—$\frac{15}{24}$—and 12-carat—$\frac{12}{24}$ or 50 per cent.—are also largely used. These are called bright golds, because they show the fine colour of the alloy. 9-carat gold—$\frac{9}{24}$ or 37·5 per cent.—is largely used for the manufacture of cheap jewellery; it is the poorest which will stand the nitric acid test, or which can be hall-marked. These alloys are frequently coloured by being heated and plunged into a pickle of acid, so as to dissolve out some of the base metal and leave the surface richer than the body of the article.

The solders used for gold are alloys of gold, silver, and copper.

SILVER ALLOYS.

Silver alloys with most metals, the only important alloys being those used for coinage.

The British coinage contains 92·5 per cent of silver, or has a fineness of 925. The coinage of most European countries has a fineness of 900, and some countries use even a much poorer standard. Silver of 925 fineness is called **standard silver**, and it is the only alloy which can be hall-marked.

Silver solders are usually alloys of silver, copper, and zinc.

COATING ONE METAL WITH ANOTHER.

On the property of alloying depends the power of coating one metal with another. If a plate of clean copper gold or other metal be dipped into mercury, combination at once takes place, and the plate becomes covered with mercury or amalgamated. The coating metal is not pure mercury, but an amalgam of mercury with the metal which adheres very tenaciously to the plate. If in place of mercury a mercury salt be used, the result is the same, the salt is decomposed and the mercury alloys with the metal. Similarly gold, silver, &c., may be deposited on the base metals by immersing them in a solution of a gold or silver salt.

Iron and some other metals are readily coated with tin, zinc, or other metal by immersion in a molten bath of the latter. In order to ensure an adherent coat, the metal must be absolutely free from scale, or dirt of any kind.

Tin-plate.—This is thin sheet-iron coated with tin by dipping. At one time wrought iron made in open fires was universally used, but now Siemens or Bessemer steel has superseded it.

The plates are pickled in warm dilute sulphuric acid, washed, thoroughly dried, annealed, and rolled cold between very smooth rolls.

They are then pickled to remove every trace of scale, washed, and dipped into a bath of molten grease, from which they are transferred to the bath of molten tin, the dipping being repeated if necessary, the plates are then put into a second grease pot kept at a temperature above the melting point of tin; superfluous tin is allowed to drain away, the plates are cooled, scoured with bran and woollen cloths, and packed for sale.

Galvanised Iron.—This is iron covered with a thin coating of zinc. The iron article is thoroughly cleansed by pickling in acid and washing, and is then immersed in molten zinc. Large articles are dipped, washed, and dried. Wire is galvanised by drawing it first through "acid pickle," and then through the molten zinc.

CHAPTER XX.

APPLICATIONS OF ELECTRICITY TO METALLURGY.

Electricity.—When plates of two different metals, say copper and zinc, are plunged into dilute acid and connected by a wire, a series of changes takes place, and an electric current is said to flow through the wire from the copper to the zinc. Under these conditions the current would be very feeble, but by suitably selecting the metals and liquids, and arranging a sufficient number of pairs, currents of greater power can be obtained. For practical purposes in metallurgy the battery is usually replaced by a dynamo, which is an apparatus by which a long coil of wire is very rapidly rotated in the magnetic field produced by powerful magnets, thus producing a current.

Of the effects produced by the current, two are used in metallurgy, electrolysis, and the production of heat.

Electrolysis.—If, instead of allowing the current to flow all its course along a wire, the wire be divided, plates of platinum or other metal be attached to the severed ends, and these put into acidified water, and the current passed, the water will be decomposed, and its constituents, hydrogen and oxygen, will be liberated. The plate attached to the copper, or that by which the current enters the solution, is called the anode; whilst that by which it leaves, and which is attached to the zinc plates, is called the cathode. The oxygen is liberated at the anode, and the hydrogen at the cathode. If in place of water the current be passed through a solution of a metallic salt, say copper sulphate, copper will be deposited at the cathode, and oxygen will be liberated at the anode, the SO_4 separated by the current yielding sulphuric acid and oxygen by contact with water.

The metal is always liberated at the cathode, and the non-metal at the anode, the amount of metal liberated being proportional to the amount of current which flows. If the anode, instead of being of platinum, be of the metal which is being deposited at the cathode, an amount of metal will be dissolved from it equal to that which is precipitated at the anode.

Electrolysis is used for three purposes—(1) Electro-deposition; (2) electro-refining; (3) electro-reduction.

Electro-Deposition.—The most important of the metals deposited are copper, gold, silver, and iron; certain alloys are also deposited. The deposition of copper is called electrotyping, the amount deposited being usually sufficient not merely to form a covering, but to make a reproduction of the article. A mould is made in wax, guttapercha, plaster-of-paris, or fusible metal—the last-mentioned is a conductor, the others are non-conductors; if, therefore, they are used the moulds are rubbed over with graphite. The mould is then attached to the wire, so as to form the cathode, and the anode is made of a sheet of copper. These are immersed in a 15 per cent. solution of copper sulphate, acidified with sulphuric acid, and a feeble current is passed. Copper is rapidly precipitated on the mould and is also dissolved from the anode, and when enough has been thrown down the layer which has been deposited is removed from the mould, every mark on it, however minute, being reproduced.

A coherent film of copper cannot be obtained on an iron article by electro-deposition in an acid sulphate solution, so for such articles a thin layer is first deposited in a hot solution of copper cyanide.

The electro-deposition of silver, called electro-plating, is commonly used for covering articles of white metal with a thin layer of silver in the manufacture of electro-plated goods. The article is made the cathode, as before; the anode is a plate of silver, and they are immersed in a solution of silver cyanide in potassium cyanide. The silver is deposited of a fine white colour, but is dull, and if a bright surface is required is burnished.

The deposition of gold, or electro-gilding, is used for coating articles with gold. The cathode is the article to be coated; the anode is a plate of gold. These are immersed for a few minutes in a hot solution of gold cyanide in cyanide of potassium; the deposition is very rapid, only a thin layer being required. The gold comes down bright, and does not require burnishing.

Iron is deposited for some purposes; the article is made the cathode, the anode is a plate of iron, and the solution is a 15 per cent. solution of ammonium-ferrous sulphate. The metal deposited is of a gray-white colour, is extremely hard and brittle, and contains about 240 times its volume of hydrogen, which is expelled by annealing. Owing to its great hardness a layer of iron is often deposited on the face of engraved copper plates to be used for printing. The operation is called *steeling the plate.*

Tin, lead, and other metals may be desposited, if required.

Brass or bronze may be deposited by using solutions containing both copper and zinc and copper and tin.

Electro-Refining.—Refining by electrolysis depends on exactly the same principle as electro-deposition. The cathode is usually a very thin plate of refined metal, and the anode is a bar or cake of the unrefined metal; these are immersed in a solution of a salt of the metal being refined. As the current passes the impure anode is dissolved and pure metal is deposited at the cathode.

Electro-Refining of Copper.—The cathode is a thin sheet of electrotype copper; the anode is a bar of blister copper, and the liquid is an acidified solution of copper sulphate. The blister copper is attacked; the copper, iron, zinc, &c., are dissolved, but the copper only is precipitated from acid solutions, so that the "bath" gradually becomes charged with impurities, till at last, when it has become too impure to work, the liquid is drawn off and the copper is thrown down by scrap iron. The silver, gold, sulphur, &c., contained in the blister copper do not dissolve, but fall to the bottom of the vat, whence they are collected and treated for the recovery of the precious metals; hence electrolytic refining allows of the recovery of gold and silver in the copper. The electrotype copper deposited is very pure, but is always highly crystalline in structure.

If the copper be deposited on a mandrel kept rotating to ensure a uniform deposit, a seamless tube could be made; this however, would be too porous to be of any use, but it may be made compact by pressing with a burnisher of agate. This process is called the Elmore process, for making seamless copper tubes.

Electro-refining of Lead.—Base-bullion may be refined by an exactly similar method. The bullion is cast into thin cakes which are enclosed in muslin bags and made the anode, the cathode being a thin plate of lead, and the solution one of lead sulphate in sodium acetate. Lead is deposited on the cathode, the silver, gold, copper, &c., remain in the bag in the form of a sponge, and are afterwards treated.

Electro-extraction of Metals.—This depends on exactly similar principles.

In the electro treatment of copper, bars of copper regulus (or the ores themselves coarsely crushed and moulded into slabs under pressure) are used as anodes, and the liquid employed is acidified solution of copper sulphate, as before. These processes have never come largely into use.

Electric Smelting.—The other principle of which use is made is the evolution of heat by the passage of an electric current through a poor conductor. If the current be sufficiently strong, a very high temperature can be obtained, far higher, indeed, than can be obtained by combustion. One example of the use of this principle has been given under "electric welding." For smelting it is used in the reduction of aluminium, magnesium, and other very refractory metals. One form of apparatus is that known as Siemens electric furnace, which has been largely used for experimental purposes. The passage of the electric current through the carbons produces a very high temperature, wrought iron and platinum being rapidly melted. If in such a furnace an electrolyte be used, it will be melted and then decomposed. In this way aluminium is now made from alumina.

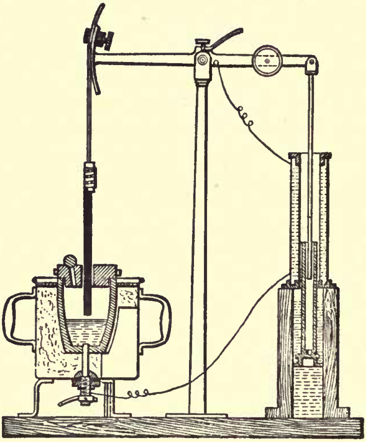


Fig. 71.—Electric Furnace.

PART II.—PRACTICAL COURSE.

I. REDUCTION.

Reduction is the separation of metals in the free state, from substances in which they exist in combination with other elements. The most important of the compounds with which the metallurgist has to deal are oxides, or can be converted into oxides by simple means, so that reduction is usually understood to be the removal of oxygen.

Reducing agents are substances which have sufficient affinity for oxygen to remove it from its compounds with the metals. The most important are—Carbon (C), Hydrogen (H), and Carbon Monoxide (CO).

The metals may be classed into groups according to the ease with which their oxides may be reduced.

1. Metals, the oxides of which are reduced by heat alone—

   Gold, Au; silver, Ag; platinum, Pt; mercury, Hg.

2. Metals, the oxides of which are reduced by C or H at a red heat—

   a. The metal, being volatile, is lost.

   Zinc, Zn; cadmium, Cd.

   b. The metal fuses

   Lead, Pb; tin, Sn; antimony, Sb; copper, Cu, &c.

   c. The metal does not fuse

   Iron, Fe; nickel, Ni, &c.

3. Metals, the oxides of which are not reduced by C or H at a red heat—

   Aluminium, Al; magnesium, Mg, &c.

Exercise 1.—Reduction of Lead Oxide.

Weigh out 10 grms. of litharge (PbO), mix it intimately with ·5 grm. of powdered charcoal, on a sheet of glazed paper. Put the mixture into a crucible, cover the crucible with a lid, and put it into the furnace. Heat it to redness, and keep at a red heat for about ten minutes. Remove the crucible from the fire and pour the lead obtained into a mould. If there be any charcoal left, examine it carefully, and pick out any shots of lead that may be mixed with it, and add them to the principal button. When quite cool weigh the lead, and calculate the percentage of lead contained in the litharge.

$$2PbO + C = 2Pb + CO_2.$$

Exercise 2.—Reduction of Copper Oxide, CuO.

Weigh 10 grms. of black oxide of copper, mix it with 2 grms. of charcoal; put the mixture into a crucible, and proceed exactly as in Exercise 1, except that, as copper is much more difficult to fuse than lead, a much higher temperature will be required. The crucible must be kept at a bright red heat for about ten minutes. Cool, pour, and weigh as before.

$$CuO + C = Cu + CO.$$

Exercise 3.—Reduction of Oxide of Iron, Fe_2O_3.

Weigh a splinter of red hæmatite, put it into a crucible, and surround it with charcoal broken into small pieces. Cover the crucible, and lute on the cover with clay. Heat to redness for about an hour, let cool, turn out the contents, and examine the piece of hæmatite. It should be unchanged in form, but should have become black, soft, malleable, and magnetic—*i.e.*, no longer hæmatite, but iron. Weigh it.

$$Fe_2O_3 + 3C = 2Fe + 3CO.$$

The temperature must not be too high, or the reduced iron will combine with the carbon and form cast iron, which will melt.

FLUXES.

It is only in the case of quite pure oxides that charcoal alone can be used as a reducing agent. If the material contains earthy impurities, these will remain unfused, and will prevent the reduced metal collecting into a button. A similar result is obtained if too much carbon is used, as can be easily tested by repeating Exercises 1 or 2 with three or four times the weight of charcoal there given. To enable the metal to collect, the whole mass must become liquid, and to ensure this some substances are usually added, which either are themselves very fusible, or which form fusible compounds with the earthy or other substances present. These are called fluxes. The primary object of a flux is to produce a fluid slag; some fluxes have also reducing or oxidising properties.

The following are the most important fluxes used in the laboratory :—

1. *Sodium Carbonate*, Na_2CO_3.—The dry powder, or the bicarbonate, $NaHCO_3$, but not the crystals which contain water. It fuses at a bright red heat to a somewhat viscid liquid, which can retain a considerable amount of solid matter in suspension. It decomposes all silicates forming fusible silicate of sodium. Hence it is very useful for dissolving siliceous gangue. A more fusible flux may be made by mixing 10 parts of sodium carbonate and 13 parts of potassium carbonate.

2. *Borax*, $Na_2B_4O_7$.—The fused salt (the crystals contain 47 per cent. of water). Borax fuses to a clear liquid; it forms a fusible compound with silica, and dissolves earthy and other oxides; forming borates. It can also retain considerable quantities of solid matter in suspension. Hence it is a very valuable flux.

3. *Common Salt*, NaCl.—This salt fuses at a red heat to a very fluid liquid, and is valuable as a covering flux to protect metals from the action of the air or to form very fluid slags. It volatilises at light temperatures, and is apt to carry off metals as chlorides.

4. *Silica*, SiO_2, combines with many earthy oxides to form fusible silicates. It is best used in the form of white sand.

REDUCING AGENTS.

5. *Carbon.*—Pure wood charcoal finely powdered is the best form in which this can be used.

6. *Argol, Bitartrate of Potash*, $KHC_4H_4O_6$.—This is a very valuable reducing agent, on heating it leaves an intimate mixture of carbon and potassium carbonate. The large amount of carbon left (about 15 per cent.) renders the mass refractory; hence it is always used in conjunction with other fluxes.

7. *Flour* or *Starch* are powerful and valuable reducing agents.

8. *Potassium Cyanide*, KCN.—This is a very valuable reducing agent under some circumstances; it absorbs O very readily, forming cyanate—*e.g.*, $PbO + KCN = Pb + KCNO$—it is very fusible, and the commercial salt always contains enough potassium carbonate to flux any silica that may be present. Potassium cyanide is extremely poisonous.

OXIDISING AGENTS.

9. *Nitre, Potassium Nitrate,* KNO_3.—This is the only oxidising agent used in the laboratory; it parts with its oxygen very readily, and is a very effective desulphurising agent.

MIXED FLUXES.

10. *White Flux.*—Equal quantities of argol and nitre mixed and ignited.

11. *Black Flux.*—A mixture of 2 parts argol and one of nitre.

12. *Black Flux Substitute.*—10 parts bicarbonate of soda and 3 parts flour, well mixed.

REDUCTION OF OXIDES.

(*With Fluxes.*)

EXERCISE 4.—DETERMINATION OF LEAD IN IMPURE LITHARGE.

Weigh out 5 grms. of litharge, mix it intimately on a sheet of paper with 1 grm. of flour and 10 grms. of sodium carbonate, put the mixture into a crucible, and cover with a layer of salt. Heat gently in the furnace till the whole is fused, then heat strongly for a few minutes, remove the crucible from the fire, tap gently to collect the metal into a button, and pour into a mould, let cool, break away the slag, powder it, pick out any shots of lead it may contain, add them to the principal button and weigh. Calculate the percentage of lead.

The temperature must not be too high, as lead is sensibly volatile.

EXERCISE 5.—DETERMINATION OF COPPER IN IMPURE COPPER OXIDE.

Weigh 5 grms. of the oxide, 10 grms. of sodium carbonate, and 1·5 grms. of flour; mix intimately, put into a crucible, cover with a layer of borax, and heat, keeping the temperature above the fusing point of copper for ten minutes, then tap and pour as in Exercise 4. Break away the slag, and weigh the copper obtained. Calculate the percentage.

EXERCISE 6.—REDUCTION OF TIN OXIDE.

Weigh 20 grms. of the oxide, mix it intimately with 5 grms. argol, 35 grms. of sodium carbonate, and 5 grms. of lime; put he mixture into a crucible, and heat gradually. Keep at a dull red heat for twenty minutes, then raise the temperature till the whole is quite fluid. Remove the crucible from the fire, tap gently, and let cool. When *quite* cold break the crucible, and clean and weigh the button of tin.

This is not a very satisfactory method, and the tin oxide must be free from silica.

EXERCISE 7.—DETERMINATION OF TIN IN TIN OXIDE.

Potassium cyanide, KCN, is the best reducing agent for tin oxide.

Put a layer of potassium cyanide at the bottom of a crucible. Mix 10 grms. of the oxide with 20 grms. of (80 per cent.) KCN, put the mixture into the crucible and cover with a layer of cyanide. Heat in the furnace, gently at first, till the mass is well fused, and keep fused for ten minutes. Remove the crucible from the fire, and tap gently; when it is cold, break it, and clean and weigh the button of tin.

Dissolve the slag in water; should any shots of tin be left, they must be collected and added to the principal quantity.

EXERCISE 8.—REDUCTION OF HÆMATITE, IRON OXIDE, Fe_2O_3.

A. Weigh 10 grms. of powdered hæmatite, mix it with 10 grms. powdered glass, 1 grm. lime, and 2 grms. charcoal. Put the mixture into a crucible, lute on the cover, and heat for an hour to the highest attainable temperature. Let cool, break the crucible and examine the residue. It should be a spongy infusible mass of malleable iron.

B. Make exactly the same mixture, except use 4 grms. of charcoal instead of 2, and proceed exactly as in A. A well-fused bead of cast iron should be obtained.

EXERCISE 9.—REDUCTION OF COPPER OXIDE BY HYDROGEN.

Fit up an apparatus for evolving hydrogen, a drying tube, and a piece of glass combustion tube about 12 inches long, arranged so that it can be heated over a bunsen, fitted with perforated corks. Select also a porcelain boat of such size that it will slip into the tube.

Weigh the boat, put into it about 1 grm. of copper oxide and weigh again, the increase of weight will be the amount of copper oxide taken. Slip the boat into the tube and start the hydrogen. As soon as all the air is expelled from the apparatus, put a burner under the part of the tube where the boat is, and heat for about half-an-hour. Let the boat cool in the current of hydrogen, then withdraw it from the tube and weigh it.

$$CuO + 2H = H_2O + Cu.$$

Exercise 10.—Comparison of the Reducing Power of the Various Reducing Agents.

Mix ·5 grm. of powdered charcoal intimately with 40 grms. of litharge, put the mixture into a crucible, cover with a layer of litharge, lute a cover on the crucible, put it into the furnace, heat to redness, and keep at a red heat for ten minutes. Remove from the fire, tap gently, pour into a mould, and when cool break away the slag and weigh the lead obtained.

Repeat the experiment, using 1 grm. of each of the following reducing agents in place of the charcoal:—

(1) Starch.
(2) Argol.
(3) Flour.
(4) Resin.

Make a table of the comparative reducing power of these substances, taking that of charcoal as 100.

II. PREPARATION OF OXIDES BY ROASTING.

Many metals can be converted into oxides by roasting in air.

Exercise 11.—Preparation of Tin Oxide (Putty Powder), SnO_2.

Put about 10 grms. of tin into a scorifier, and put it into a muffle. Close the doors of the muffle till the tin is melted, then open them so as to allow free access of air. The tin will soon become covered with a film of oxide, push this back so as to expose a fresh surface of the metal, and continue doing this till all the tin seems to be converted into oxide. Remove the scorifier from the muffle, and when cool grind the resulting oxide in a mortar, and pass it through a sieve with 60 holes to the inch. Return any unoxidised tin which remains on the sieve to the scorifier, and roast again; cool and powder as before, and so on till all the metal is oxidised. Weigh the resulting oxide, and from its weight and that of the tin used calculate the composition of the oxide.

$$Sn + 2O = SnO_2.$$

The oxidation of the last portions is often troublesome; it is therefore convenient, when the quantity of residual tin is small, to weigh it, and deduct the weight from that of the tin used.

EXERCISE 12.—PREPARATION OF LITHARGE, PbO.

Weigh about 20 grms. of lead, put it on a scorifier, and proceed exactly as in Exercise 11.

Great care must be taken in regulating the temperature; if it be too low the scum will consist mainly of lead, if it be too high the litharge will melt and will attack the scorifier.

$$Pb + O = PbO.$$

By continued roasting the litharge may be converted into red lead.

EXERCISE 13.—PREPARATION OF BLACK OXIDE OF COPPER, CuO.

Put some strips of copper on a roasting dish into the muffle, and heat them to redness for about ten minutes, remove them from the muffle and let them cool. They will be found to be covered with a film of oxide which can be detached. This oxide is a mixture of cuprous and cupric oxides. By detaching the scale and reheating the copper several times a quantity of scale may be collected, as, however, this is somewhat tedious, commercial copper scale, which is of similar composition, may be used for the rest of the experiment.

Powder about 10 grms. of the scale, put it on a roasting dish and roast till the colour becomes black, let cool, and weigh; return to the dish and roast again. Repeat this till there is no further gain in weight.

From the gain in weight calculate the percentage of red oxide, Cu_2O, contained in the scale. The reaction is

$$x\mathrm{CuO} + y\mathrm{Cu_2O} + y\mathrm{O} = (x + y)\,\mathrm{CuO}.$$

71 grms. of Cu_2O give 79 grms. of CuO, hence an increase in weight of 8 grms. would be equivalent to 71 grms. of Cu_2O in the mixture.

Suppose the black oxide obtained weighs 11 grms., then the gain in weight will be 1 grm., then

$$8 : 71 : 1 : x = \frac{71}{8} = 8{\cdot}36,$$

so that the 10 grms. of scale contained 8·36 or 83·6 per cent. of red oxide.

EXERCISE 14.—PREPARATION OF RED OXIDE OF COPPER, Cu_2O.

Weigh 10 grms. of powdered black oxide of copper, in as fine a state of division as possible; mix it intimately with 8 grms. of copper, also finely divided; put the mixture into a crucible, lute on a cover, heat slowly to full redness, and keep at that temperature for about five minutes, then remove from the fire, take off the lid, and pour the oxide into a mould, or let the crucible cool and break it. The oxide should have a dark red colour, which, when powdered, should become crimson, $CuO + Cu = Cu_2O$.

EXERCISE 15.—PREPARATION OF OXIDE OF ZINC, ZnO.

Weigh 10 grms. of zinc into a roasting dish, and proceed exactly as in Exercise 12. The temperature must not be too high, as zinc is volatile.

III. PREPARATION OF SULPHIDES.

Most metals combine very readily with sulphur when the two are heated together. The sulphides are, as a rule, much more fusible than the metals which they contain.

EXERCISE 16.—PREPARATION OF IRON SULPHIDE, FeS.

Weigh 20 grms. of iron filings and 20 grms. of stick sulphur; mix them intimately, and put the mixture into a long scoop. Put a crucible into the furnace, and when it is well red hot project the mixture into it. Cover the crucible and heat till the whole is well fused, then remove the crucible from the fire, and pour the contents into a mould. When cool weigh it, and from the weight obtained calculate the composition of the sulphide.

Example.—Suppose 31·4 grms. of sulphide were obtained from 20 grms. of iron, then the compound will contain 20 grms. of iron and 11·4 grms. of sulphur,

$$\begin{aligned} \text{So that } 31{\cdot}4 : 20 :: 100 : x &= 63{\cdot}7 \text{ per cent.,} \\ \text{and } 31{\cdot}4 : 11{\cdot}4 :: 100 : x &= 36{\cdot}3 \quad ,, \\ & \overline{100} \quad ,, \end{aligned}$$

EXERCISE 17.—PREPARATION OF SULPHIDE OF LEAD.

Weigh out 20 grms. of lead, preferably in thin strips, put it into a crucible, cover it with flowers of sulphur, put on cover and put into the furnace. When the whole is well fused, and the evolution of sulphur vapour ceases, remove the crucible from the fire, and pour its contents into a mould. Calculate the composition of the sulphide.

EXERCISE 18.—PREPARATION OF COPPER SULPHIDE, Cu_2S.

Weigh 20 grms. of copper in cuttings or turnings; mix it with its own weight of sulphur, and project it into a red-hot crucible (as in 15). Cover the crucible and heat strongly. When the mass is well fused pour it into a mould, and when cool weigh it.

EXERCISE 19.—PREPARATION OF TIN SULPHIDE, SnS.

Weigh 20 grms. of tin, mix it with its own weight of flowers of sulphur, and project the mixture into a red-hot crucible. When the mass is well fused pour it into a mould. Powder the sulphide, and pass it through a 60 sieve. Mix any residue of tin that may remain with more sulphur, and again fuse, cool, and powder. When all the tin has been converted, return the powdered sulphide to the crucible, cover it with a layer of salt, heat strongly for ten minutes, pour into a mould, and when cool weigh.

Many sulphides can be prepared by heating an oxide of the metal with sulphur.

EXERCISE 20.—PREPARATION OF LEAD SULPHIDE FROM LITHARGE.

Weigh out 20 grms. of litharge, mix it intimately with 20 grms. of sulphur, and heat gradually to redness, then pour into a mould.

EXERCISE 21.—PREPARATION OF ZINC SULPHIDE.

Zinc and sulphur do not combine when heated, the temperature of combination being above the boiling point of zinc. Weigh 10 grms. of zinc oxide and mix it intimately with 10 grms. of flowers of sulphur, project the mixture into a hot crucible, cover, and heat strongly for fifteen minutes, then remove from the fire, and pour into a mould.

$$2ZnO + 3S = 2ZnS + SO_2.$$

IV. REDUCTION OF SULPHIDES.

Sulphides are not completely reduced by the ordinary reducing agents, such as carbon. They may be converted into oxides by roasting, and these may be reduced, or they may be reduced directly by various desulphurising agents, of which iron is the most suitable.

EXERCISE 22.—REDUCTION OF GALENA, PbS.

Mix 20 grms. of pure lead sulphide with 5·5 grms. of iron filings, project the mixture into a hot crucible, and heat to bright redness for about ten minutes. When the mass is well fused, tap gently, let cool, break the crucible, clean the lead from the slag, which is iron sulphide, FeS, and weigh.

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

As a rule, it is better to add sodium carbonate as well as the iron.

EXERCISE 23.—REDUCTION OF GALENA, PbS.

Mix 20 grms. of powdered galena with 20 grms. of sodium carbonate, and 2 grms. of argol; put the mixture into a large crucible, stick in two or three large nails, and cover with a layer of borax. Put the crucible into the furnace, heat it to redness, and keep it at that temperature for about fifteen minutes. Remove the nails one by one with the tongs, tapping them so that any adherent shots of metal may fall into the crucible; then remove the crucible, tap gently, and pour into a mould. Break away the slag, and weigh the lead obtained.

EXERCISE 24.—REDUCTION OF ANTIMONITE, Sb_2S_3.

Weigh 20 grms. of finely-powdered antimonite, mix it with 9 grms. of iron filings, and about 10 grms. of sodium carbonate; put the mixture into a crucible, heat gently at first, then strongly, and keep the mass in a state of fusion for at least fifteen minutes; then remove from the fire, tap gently, and let cool. Break the crucible, separate the slag very carefully, since antimony is very brittle, and weigh the metal bead.

$$Sb_2S_3 + 3Fe = 3FeS + 2Sb.$$

The temperature must not be too high, as antimony is readily volatile.

In Exercises 23 and 24 instead of iron a mixture of ferric oxide and carbon may be used.

Potassium cyanide also acts as a powerful desulphurising agent. Potassium sulphocyanate, KCNS, being formed.

EXERCISE 25.—REDUCTION OF ANTIMONITE, Sb_2S_3.

Weigh 10 grms. of powdered antimonite, and mix it with 30 grms. of powdered potassium cyanide; put the mixture into a crucible, at the bottom of which about 10 grms. of cyanide have been put, cover the crucible, and heat in the furnace. As soon as the mass is well fused, remove the crucible from the fire, and let cool with the cover on. Break the crucible and weigh the antimony.

V. ROASTING.

EXERCISE 26.—ROASTING IRON PYRITES, FeS_2.

Weigh 10 grms. of finely-powdered pyrites, put it into a scorifier or roasting dish, and heat in the muffle, leaving the door open so as to allow free access of air. Stir with an iron rod, and continue the roasting till the mass on removal from the muffle has no smell of burning sulphur (SO_2), then cool and weigh the oxide, which should have a red colour.

$$2FeS_2 + 11O = Fe_2O_3 + 4SO_2.$$

EXERCISE 27.—ROASTING COPPER SULPHIDE, Cu_2S.

α. For Sulphate.—Weigh 10 grms. of copper sulphide, put it on a scorifier and roast at a low red heat. When the roasting is complete remove the dish, let the powder cool somewhat, and brush it into a beaker containing about 100 c.c. of water. Boil for a few minutes and filter; to the filtrate add ammonia in excess. The solution should be coloured blue, showing that some copper was dissolved by the water, part of the copper sulphide having been converted into sulphate.

$$x + yCu_2S + (5x + 4y)O = xCuSO_4 + (x + 2y)CuO + ySO_2.$$

β. For Oxide.—Roast 10 grms. of copper sulphide exactly as in α, but raise the temperature as high as possible towards the end of the roasting. Cool and weigh.

On boiling the residue with water, filtering and adding ammonia, there should be no blue colour; the copper sulphate first formed being decomposed, $CuSO_4 = CuO + SO_3$, so that the ultimate reaction is $Cu_2S + 4O = 2CuO + SO_2$.

The oxide obtained could be reduced exactly as in Exercise 5.

EXERCISE 28.—ROASTING ANTIMONITE.

Weigh 10 grms. of finely-powdered antimonite, put it into a scorifier and roast till it no longer smells of SO_2. Cool and weigh the resulting oxide.

This operation requires very great care, as the sulphide is very easily fusible; should it clot, the experiment must be begun over again.

$$Sb_2S_3 + 9O = Sb_2O_3 + 3SO_2.$$

Mix the oxide with 20 grms. carbonate of soda and 10 grms. of argol, and fuse. Let cool, break the crucible, and weigh the residual antimony.

EXERCISE 29.—ROASTING ZINC BLENDE, ZnS.

Weigh 10 grms. of powdered blende and roast it on a roasting dish, at first at a low temperature, but finally at as high a heat as possible. Cool and weigh

$$ZnS + 3O = ZnO + SO_2.$$

EXERCISE 30.—ROASTING GALENA.

Weigh 10 grms. of finely-powdered galena, put it on a scorifier, cover with an inverted scorifier, and put in the muffle and close the doors when the scorifiers are at a dull red heat, open the doors, remove the upper scorifier, and roast till the mass no longer smells of SO_2. Let cool and weigh.

The residue consists of a mixture of PbO and $PbSO_4$, $(x+y)PbS + (4x+3y)O = xPbSO_4 + yPbO + ySO_2$. Calculate from the weights the proportions in which the sulphide and sulphate are present in the mixture—*i.e.*, the value of x and y. Great care must be taken not to raise the temperature too high, especially at the early part of the roasting, as PbS and PbO are fusible.

VI. MORE COMPLEX REDUCTIONS.

EXERCISE 31.—REDUCTION OF LEAD SULPHIDE BY LEAD OXIDE.

Weigh 10 grms. of powdered galena, mix it intimately with 20 grms. of litharge, put the mixture into a crucible, cover with a layer of sodium carbonate, cover the crucible, and heat till the mass is well fused. Pour into a mould and weigh the lead obtained.

$$PbS + 2PbO = 3Pb + SO_2.$$

EXERCISE 32.—REDUCTION OF LEAD SULPHIDE BY LEAD SULPHATE.

Weigh 10 grms. of finely-powdered galena, and mix it with 12·5 grms. of lead sulphate, put into a crucible, cover, put into the furnace and heat to fusion, put into mould, and weigh the lead produced.

$$PbS + PbSO_4 = 2Pb + 2SO_2.$$

EXERCISE 33.—REDUCTION OF COPPER OXIDE BY COPPER SULPHIDE.

Weigh 10 grms. of powdered copper sulphide, and roast it to oxide (or take 10 grms. of copper oxide) and mix it with 10 grms. of copper sulphide. Put the mixture into a crucible, cover with a layer of sodium carbonate, and heat up to the fusing point of copper. Pour into a mould, and weigh the copper obtained.

$$2CuO + Cu_2S = 4Cu + SO_2.$$

EXERCISE 34.—REDUCTION OF COPPER SULPHIDE BY COPPER SULPHATE.

Weigh 10 grms. of copper sulphide, mix it intimately with 10 grms. of anhydrous copper sulphate; put the mixture into a crucible, cover with sodium carbonate, and heat to the melting point of copper. Pour into a mould, separate, and weigh the resulting copper.

VII. PREPARATION OF ALLOYS.

Exercise 35.—Preparation of Pewter.

Weigh 40 grms. of tin and 10 grms. of lead, put them into a crucible, cover with charcoal, and heat. When the metals are melted stir with an iron rod, and pour into a mould. Weigh the alloy obtained, and test its malleability by flattening it on an anvil.

Other alloys—such as solder, 30 grms. tin, 10 grms. lead; type metal, 20 grms. tin, 20 grms. antimony, 40 grms. lead—may be made in exactly the same way.

Exercise 36.—Preparation of Fusible Metal.

Weigh 20 grms. of bismuth, 10 grms. tin, 10 grms. lead, melt them at a low temperature under charcoal, and pour into a mould. Weigh the alloy. This alloy will melt in boiling water.

Exercise 37.—Preparation of Yellow Brass (50 and 50).

Brass is essentially an alloy of copper and zinc, the proportions varying according to the purposes for which it is required. As zinc is volatile, a certain amount is always lost, and this must be allowed for by addition of excess of zinc.

Weigh 50 grms. of copper and melt it under charcoal; when the copper is melted add the zinc. This is best done by taking the zinc in the tongs and stirring it into the molten copper till it is dissolved. Stir with an iron rod, pour into a mould, and weigh the brass obtained. Calculate its composition, and break the ingot and examine its fracture.

Other copper alloys—such as best brass, 80 grms. copper, 22 grms. zinc; muntz metal, 30 grms. copper, 22 grms. zinc, &c.—may be made in the same way.

Exercise 38.—Preparation of White Brass (30 to 70).

Proceed exactly as in Exercise 37, using 30 grms. of copper and 75 grms. of zinc.

Bronze is essentially an alloy of copper and tin, though other metals are frequently added.

EXERCISE 39.—PREPARATION OF GUN METAL.

Melt 45 grms. of copper under charcoal. When it is melted add 5 grms. of tin, stir, and pour into a mould. Weigh the alloy obtained.

EXERCISE 40.—PREPARATION OF SPECULUM METAL.

Melt 35 grms. of copper under charcoal, add 14·5 grms. of tin, and ·5 grm. of arsenic. Stir, pour into a mould, and weigh the alloy.

The alloy should be hard and brittle, and should take a high polish.

EXERCISE 41.—PREPARATION OF REGULUS OF VENUS.

Melt 25 grms. of copper under charcoal, add 25 grms. of antimony, stir, and pour into a mould.

The alloy should be brittle, and the fractured surface should have a violet colour.

EXERCISE 42.—PREPARATION OF GERMAN SILVER.

Owing to the extremely different melting points of the metals used, this is somewhat difficult to make.

Weigh 30 grms. of copper and alloy it with 30 grms. of zinc exactly as in making brass.

Melt together under charcoal 25 grms. of copper and 20 grms. of nickel. When well fused add the copper zinc alloy previously heated to redness, stir well, and pour into a mould. The temperature must be maintained as high as possible during the fusion.

EXERCISE 43.—ALUMINIUM BRONZE.

Fuse 45 grms. of copper in a crucible without any flux, then add 5 grms. of sheet aluminium, stir well, keep fused for about five minutes, and then pour into a mould.

EXERCISE 44.—PHOSPHOR BRONZE.

This is bronze to which a little phosphorus is added, either as phosphor bronze or phosphor tin.

Phosphor Copper.—Weigh out 2 grms. of copper phosphate, mix it with ·5 grm. of charcoal, cover with a layer of sodium carbonate, and heat to fusion. Pour into a mould, separate from the slag, and weigh the alloy.

Phosphor Bronze.—Prepare bronze exactly as in Exercise 39, then add 1 grm. of the phosphor copper, stir and pour.

EXERCISE 45.—PREPARATION OF CALAMINE BRASS

Mix 50 grms. of black oxide of copper, 50 grms. of oxide of zinc, and 5 grms. of powdered charcoal; cover the crucible and heat to fusion. Pour the resulting alloy into a mould; when cool weigh, then break it and examine the fracture.

EXERCISE 46.—COPPER AND LEAD.

Melt 50 grms. of copper and 50 grms. of lead; stir and let cool slowly. The metals will separate into two distinct layers, the lead below, the copper above.

Similarly, zinc and lead melted together separate on cooling.

EXERCISE 47.—EFFECT OF ARSENIC ON TIN.

Melt 50 grms. of tin under charcoal, add 5 grms. of arsenic, stir, and pour into a mould. When cool, break the ingot and examine its fracture.

VIII. LIQUATION.

EXERCISE 48.—LIQUATION OF LEAD AND COPPER.

Remelt the copper and lead from Exercise 45, stir, and without allowing time for the metals to separate pour into a mould. Hammer the button of metal into a flat disc, and put it into a crucible of such size that it rests about half-way down in an inclined position. Put some charcoal on the top, cover the crucible, heat for half-an-hour to dull redness. The lead will melt and will run down. Weigh the residual copper and the lead obtained.

Care must be taken not to melt the copper.

IX. PREPARATION OF SILICATES.

EXERCISE 49.—PREPARATION OF LEAD SILICATE.

Weigh 44 grms. of litharge, mix it intimately with 12 grms. of silica, put the mixture into a crucible, cover, heat gently for half-an-hour, then raise the temperature to full redness. When fusion is complete pour into a mould, and when cool weigh the glass obtained.

EXERCISE 50.—PREPARATION OF CUPROUS SILICATE.

Mix 20 grms. of cuprous oxide and 8·5 grms. silica, put into a crucible, lute on a cover; heat gently at first, keeping at a dull red heat for half-an-hour, then heat as rapidly as possible to fusion; remove the cover and pour into a mould.

EXERCISE 51.—PREPARATION OF BLAST FURNACE SLAG

This is a double silicate of lime and alumina. Mix together 40 grms. China clay, 20 grms. of white sand, and 50 grms. of lime, all finely powdered; put into a crucible, lute on the lid, heat gently at first, then to the highest temperature, and when fused pour into a mould.

EXERCISE 52.—PREPARE FERROUS SILICATE.

Mix 32 grms. ferric oxide, 25 grms. charcoal, and 12 grms. sand, and put into a crucible. Heat gently for half-an-hour, then to the highest temperature till fused, and pour into a mould.

X. CUPELLATION.

EXERCISE 53.—PREPARATION OF CUPELS.

Mix some powdered bone ash with water till it just coheres when strongly pressed. Fill the cupel mould with this, place the die above and strike it down with a mallet. Trim the edges if necessary with a sharp knife, push the cupel out of the mould. Make at least a dozen, and put them in a warm place to dry.

EXERCISE 54.—CUPELLATION.

Place four cupels in the muffle and heat them to full redness; when they are hot put into (1) a piece of lead; (2) a piece of bismuth; (3) a piece of silver; (4) a piece of copper. Close the doors till the silver is melted, then open them so as to allow free access of air. Heat for some time, and note the result:—

1. Lead—absorbed completely.
2. Bismuth ,, ,,
3. Silver—unchanged.
4. Copper—oxidised, but not absorbed.

EXERCISE 55.—CUPELLATION OF LEAD AND OTHER METALS.

Take two cupels; put into one ·2 grm. of silver, and into the other ·2 grm. copper; then into each 5 grms. of lead. Put the cupels into the muffle, close the doors till the lead is melted, then open them so as to allow free access of air, and heat till the lead disappears; then remove from the muffle, cool, examine the cupels, and weigh the residual silver. Note the vegetation of the silver.

EXERCISE 56.—ESTIMATION OF SILVER IN LEAD.

Weigh 10 grms. of argentiferous lead, put it into a cupel; cupel exactly as in Exercise 54, and weigh the residual silver. A cupel will absorb about its own weight of lead as litharge.

EXERCISE 57.—ESTIMATION OF SILVER IN AN ALLOY.

Weigh out about ·5 grm. of the alloy, wrap in about 3 grms. of sheet lead, and cupel as in Exercise 55. Weigh the residual silver.

XI. REFINING AND PURIFYING METALS.

EXERCISE 58.—SCORIFICATION.

When an alloy of lead and silver is roasted with free access of air the lead oxidises and passes off as oxide, whilst the silver is concentrated in the residual lead. Other base metals are to a large extent carried off by the litharge.

Weigh 10 grms. of lead containing silver, put into a scorifier and heat in a muffle to a full red heat, till the surface is covered with fused litharge, pour into a mould, break away the slag, and cupel the residual lead.

EXERCISE 59.—SCORIFICATION OF SILVER ORES.

Silver ores are decomposed by scorification with lead, the silver passing into the lead.

Weigh out about 5 grms. of a rich silver ore, mix it with about 20 grms. of lead, put into a scorifier, and heat to light redness, put a little borax on the surface, and keep in the muffle till the surface is covered with fused slag, then pour into a mould, break away the slag, and cupel the residual lead.

EXERCISE 60.—VARIETIES OF COPPER.

Melt 50 grms. of copper, add about 1 grm. of oxide of copper, or keep the melted mass exposed to the air for some time; stir, and pour into a mould. When the copper is cold, nick it with a chisel, and break it in the vice; it should be very brittle, and break with a dull red fracture. This is *dry copper;* the dryness is due to the presence of dissolved cuprous oxide.

Return the copper to the crucible, melt it under charcoal, stir with a stick, and pour into the mould. Test by nicking, fixing in the vice, and hammering; it should double over a long way before breaking, and should break with a silky fracture. This is *tough copper.* If it be not tough, return it to the crucible, and again melt it under charcoal. When tough copper is obtained, return it to the crucible, keep it melted under charcoal for some time, stirring occasionally with a stick; then pour and test. The copper should be *overpoled.* It should be brittle, and have a pale granular fracture.

EXERCISE 61.—REFINING COPPER.

Weigh 10 grms. of crude copper, such as Chili bar or blister copper. Heat a crucible to bright redness, and drop into it the weighed copper. Cover the crucible. When the copper is melted, remove the cover so as to allow free access of air, close the top of the furnace, leaving just sufficient opening to allow the interior of the crucible to be seen. The surface of the molten copper will be at first dull, but as oxidation goes on it will brighten for a moment; immediately this happens shoot in about 20 grms. of refinery flux,* cover the crucible, heat strongly for a few minutes, then pour into a mould. Break away the slag and examine the bead. It should be of a copper colour, and should have a slight dent in the top. Break it; it should be tough.

If it be not sufficiently refined it must be returned to the crucible and melted with more refinery flux. If it be dry it must be remelted under carbonate of soda and argol.

When the copper is obtained tough, return the slags to the crucible with some argol; fuse, pour. Should a minute shot of copper be left, add it to the main portion before weighing.

EXERCISE 62.—PREPARE AND SOFTEN HARD LEAD.

Melt 20 grms. of lead in a crucible, add 1 grm. of antimony, stir, and pour into a mould. Hammer the metal, it is hard and

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* *Refinery Flux.*—Nitre, 3 parts; argol, 2 parts; salt, 1 part.

brittle. Put the hard lead into a scorifier and melt it in the muffle, stir it from time to time so as to expose a fresh surface to the action of the air. After roasting for some time pour the lead into a mould, and when cold test it as to its softness and malleability. If it be not soft return it to the muffle and roast again. This must be repeated till the lead can be hammered thin without cracking at the edges.

EXERCISE 63.—PREPARATION OF SILVER FROM AN ALLOY.

Dissolve a silver coin in dilute nitric acid in a beaker, and heat, then add hydrochloric acid; boil, filter, wash, and dry the precipitated silver chloride. Mix the dry precipitate with about six times its own weight of dry sodium carbonate in a small crucible and fuse at a high temperature. Let cool, break the crucible, and weigh the separated silver.

$$2AgCl + Na_2CO_3 = 2Ag + 2NaCl + CO_2 + O.$$

XII. WET PRECIPITATIONS.

EXERCISE 64.—PRECIPITATION OF COPPER.

Dissolve about 2 grms. of copper sulphate in water, add dilute sulphuric acid, and some small nails or pieces of iron wire. Let stand for some time in a warm place till the blue colour has disappeared. The iron will dissolve and the copper will be precipitated. Pour off the solution, wash once or twice by decantation, remove the nails, rising off any adherent copper, pour off the water, rinse the copper twice with alcohol, dry in the water bath, and weigh.

EXERCISE 65.—PRECIPITATION OF SILVER.

Dissolve about 2 grms. of silver in hot, strong sulphuric acid; dilute considerably, and add some thin strip copper. Let stand for some hours in a warm place with occasional stirring. Remove the strips of copper, rinse off any adherent silver; filter and wash.

Melt the silver in a small crucible under sodium carbonate.

Precipitate the copper from the filtrate by means of iron.

XIII. REFRACTORY MATERIALS.

Exercise 66.—Preparation of Crucible Covers.

Mix some powdered Stourbridge clay with about half its weight of powdered burnt clay and enough water to just make the mass plastic. Roll it out into a thin sheet on a board, and cut it into discs of the required size by the circular tin cutters; attach a small piece of clay for a handle, dry the covers, then heat them to redness in the muffle; cool and keep for use.

Exercise 67.—Preparation of Clay Crucibles.

Make a mixture of burnt and unburnt clay as above, knead it thoroughly, put some of the clay into the cylinder of the crucible mould, and press in the core, giving it a rotary motion. Remove the core, drop a small plug of clay into the hole at the bottom left by the point of the plug, smooth off with the smoothing core. Trim the top with a sharp knife if necessary, push out of the mould and set aside to dry. When dry fire in the muffle.

The mould and cores should be slightly greased to prevent adhesion of the clay.

Exercise 68.—Brasquing Crucibles.

Use the crucibles prepared as above. Mix some treacle with its own bulk of boiling water, add just enough of this in some charcoal powder to make it cohere when pressed. Mix very thoroughly, fill the crucibles with the mixture and press it well in. By means of a sharp knife cut a cavity in the centre of the mass, and smooth the inside with a glass rod rounded at the end. Stand aside in a warm place to dry, then put a cover on each and heat to redness in the muffle. The brasque should form a firm coherent lining to the crucible.

Exercise 69.—Testing Fire-Clay.

Mix the fire-clay with enough water to make it cohere under pressure. Knead it well, and roll it out into a thin sheet. Cut out some triangles and dry them. When dry put them into a black-lead crucible, and heat to the highest possible temperature; let cool, and examine the pieces. Note particularly whether the edges or corners are in the least rounded.

XIV. FUEL.

Exercise 70.—Determination of Coke Yielded by a Coal.

Weight 10 grms. of finely-powdered coal, put it into a porcelain crucible provided with a lid. Put this into a larger clay crucible containing powdered coke, cover the small crucible with pieces of charcoal and the larger one with a lid. Put into the furnace, heat strongly for twenty minutes, then let cool. When quite cold remove the inner crucible, weigh the coke, and examine its properties.

Exercise 71.—Determination of the Ash of a Coal.

Weigh 2 grms. of the finely-powdered coal, put it into a scorifier and burn in a muffle, stirring occasionally with a platinum wire till all the carbon is consumed. Let cool, brush the ash into a watch-glass, and weigh it.

XV. MISCELLANEOUS EXAMPLES.

Exercise 72.—Amalgamation.

Weigh about ·5 grm. of finely-divided silver, put it into a mortar, add 10 grms. of mercury, and grind with a pestle till the silver has disappeared. Squeeze the mercury through a piece of chamois leather, transfer any residue to a small gold assay crucible, cover it, and heat it above the melting point of silver. When all the mercury has volatilised, cool and weigh the residual silver.

Exercise 73.—Amalgamation of Silver Ore.

Weigh ·5 grm. of finely-divided silver, and mix it with 9·5 grms. of sand. It will now represent an ore containing 5 per cent. of silver in the metallic condition. Put the mixture into a mortar, add 10 grms. of mercury, grind thoroughly. Wash away the sand with a gentle stream of water. Squeeze the mercury through chamois leather, and expel the mercury from the residual amalgam exactly as in Exercise 72.

EXERCISE 74.—CHLORIDISING ROASTING.

Take 10 grms. of copper pyrites, 10 grms. of salt, and 1 grm. of silver sulphide. Roast at a red heat in the muffle for about an hour. Let cool. Boil the powder with water, acidified with a drop or two of hydrochloric acid, and filter. To the filtrate add potassium iodide (KI) solution drop by drop, with constant stirring, till the solution is coloured brown by iodine, then filter. Dry the precipitate, put it into a cupel containing about 5 grms. of melted lead, cupel, and weigh the silver obtained. $KI + AgCl = AgI + KCl$. This is Claudet's process.

EXERCISE 75.—VON-PATERA PROCESS.

Mix pyrites, salt, and silver sulphide, and roast exactly as above. Warm the residual mass with a 10 per cent. solution of sodium thiosulphate ($Na_2S_2O_3$). Filter and wash. To the filtrate add ammonium sulphide, let settle, filter, dry the precipitate, cupel it with lead, and weigh the silver.

EXERCISE 76.—PRECIPITATION OF GOLD.

Weigh ·5 grm. of gold, dissolve it in aqua regia, and evaporate to dryness; redissolve in boiling water, and dilute to about 500 c.c., add a solution of ferrous sulphate, and warm. The gold is precipitated. Let settle, pour off the water, transfer the gold to a small crucible, heat it to redness in a muffle, and weigh.

EXERCISE 77.—CHLORINATION OF GOLD ORE.

Mix ·5 grm. of gold with about 10 grms. of sand. Put the mixture into a flask and cover it with water. Saturate the water with chlorine, cork the flask, and stand it aside. If the green colour of the chlorine disappears, pass in more, so as to keep the flask full of the gas. Filter, wash. Boil till all chlorine is expelled, then precipitate the gold with ferrous sulphate, as in Exercise 76.

EXERCISE 78.—CEMENTATION.

When wrought iron is heated in carbon combination takes place. Put a few large nails or pieces of hoop iron in a large crucible, and fill up with charcoal: lute on a cover and heat strongly for two hours, let cool, remove the iron and break it; the fracture will be found to be bright and crystalline, or steely.

PROPERTIES OF THE ELEMENTS.

	Symbol.	At. Wt.	Sp. Gr.	M. P.
Hydrogen, . .	H	1	...	Gas.
Oxygen, . . .	O	16	...	Gas.
Chlorine, . . .	Cl	35·5	...	Gas.
Sulphur, . . .	S	32	2·05	114° C.
Carbon, . . .	C	12	2·4	Infusible.
Metals.				
Aluminium, . .	Al	27	2·67	700°
Antimony, . .	Sb	120	6·7	425°
Arsenic, . . .	As	75	5·6	...
Bismuth, . . .	Bi	208	9·83	270°
Copper, . . .	Cu	63·5	8·96	1,090°
Gold, . . .	Au	196	19·5	1,200°
Iron, . . .	Fe	56	7·79	2,000° (?)
Lead, . . .	Pb	206	11·45	334°
Mercury, . . .	Hg	200	13·59	39°
Nickel, . . .	Ni	88·5	8·80	1,600° (?)
Silver, . . .	Ag	108	10·50	1,000°
Tin, ...	Sn	118	7·29	235°
Zinc, . . .	Zn	65	7·10	433°

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