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PAPERS

READ BEFORE THE

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OF THE

SCHOOL OF PRACTICAL SCIENCE

TORONTO.

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PREFACE.

The Papers read before the Engineering Society of the School of Practical Science for the Session '97-'98, are printed in the present volume.

The practical nature of the papers presented will make them especially valuable, and the Society feels grateful to those who have contributed.

The paper by Mr. Elliott on "Electric Power" is one of the best we have yet had, and represents a vast amount of work in the collection and compilation of data.

The paper on the "Production of Steel by the Basic Process," will be of general interest on account of the growing favor with which this process is being received, and the superior quality of steel produced by it.

The work of getting the cuts made has been done by Mr. J. Keele, B.A.Sc., to whose kind assistance the Society is much indebted.

There are fifteen hundred copies of this volume.

TORONTO, April 15th, 1898.

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ENGINEERING SOCIETY

—OF—

The School of Practical Science

TORONTO.

PRESIDENT'S ADDRESS.

GENTLEMEN,—It is my first sad duty to refer to the loss which the School and the Engineering Society have sustained in the death of Mr. W. E. Boustead, B.A. Sc., who previous to this sad occurrence held the position of Demonstrator in Metallurgy and Assaying, and was also a life member of our Society. Mr. Boustead's long illness made it impossible for him to carry on his duties at the School last year, and for this reason he may not have been known to many in the lower years; but to those of us who were fortunate enough to come under his instruction he had endeared himself as a painstaking and conscientious instructor with whom it was indeed a pleasure to pursue one's studies.

You have elected me to the responsible position of President of our Society. Let me assure you that I fully appreciate the great honor you have conferred upon me by so doing. I thank you most sincerely, and hope that by entire devotion to the interests of the Society I may in some measure merit the confidence you have reposed in me.

The members of the General Committee have already shown themselves to be energetic and capable officers. On their behalf I bespeak your hearty co-operation.

Your presence here this afternoon encourages me to believe that it is your intention to take the fullest advantage of the opportunities offered by closely identifying yourselves with the Society. For the benefit of those who are with us to-day for the first time a short account of the history, aims, and internal economy of our organization may not be out of place, and will, we hope, stimulate in many that active interest in the Society which can alone maintain the high standard which has been set for us by our predecessors.

The Engineering Society of the School of Practical Science was founded in the spring of 1885. Professor Galbraith was elected President, and remained at the head of the Society during the three following years. Under the careful nursing of our worthy Principal the infant Society quickly developed into a robust organization, quite capable, Professor Galbraith thought, of taking care of itself. At his suggestion, therefore, the undergraduates, in the spring of 1888, elected one of their number to the Presidency, and have continued to do so from that time. That the students were worthy of this confidence is shown by the continued advances made by the Society under the management of the able men, who, backed as they were by energetic committees and enthusiastic members, have elevated our Society to the proud position it now occupies.

This growth in influence and usefulness is all the more substantial because of the fact that it has often been in the face of difficulty. Two years ago it was only the clear-headed energy of Mr. Campbell and his colleagues which saved the Society from becoming bankrupt.

Last year Mr. King grappled with the question of a revision of our constitution, which, in keeping pace with the requirements of the growing Society, had been so many times amended and changed that it had become too unwieldy and obscure. A competent committee was appointed, of which Mr. Duff was chairman. As a result of their labors we now have our constitution and by-laws collected together under one cover in a concise and comprehensive form.

As will be seen by reference to our constitution the objects of the Society are : (*a*) The encouragement of original research in the science of Engineering ; (*b*) the preservation of the results of such research ; (*c*) the dissemination of these results among its members ; and (*d*) the cultivation of a spirit of mutual assistance among the members of the Society in the practice of the profession of Engineering.

But this is I think a very modest enumeration of the advantages which the Society offers to its members. Assisted by liberal contributions from the Faculty, the Society maintains a library, the shelves of which are well stocked with the class of literature which is of most value to Engineering students. Here also may be found the leading scientific magazines and papers of the day. New books are being constantly added, last year's committee having set aside $25 for the purpose of making additions to our already excellent collection.

Of no small importance, but overlooked by some, is the fact that the Society offers an excellent opportunity to the undergraduate to accustom himself to public speaking, and to methods of procedure at public meetings; two accomplishments which may some day be of great service to us as Engineers. The meetings of the Society are held every second Wednesday throughout the academic year, at three o'clock, at which hour all other departments of the school are closed. The Council have kindly granted this time from the regular school hours, and by this act recognize our meetings as part of the regular work of the school.

The main feature of our gatherings is the reading of papers on engineering subjects by the students and graduates. As to the nature of these papers, do not think that because you may not be able to write something which will be a valuable addition to current Engineering literature, that your paper will not be acceptable to the Society. The majority of us have as yet had very little experience in the actual practice of engineering. On the other hand, very few go through the school who have not spent some of their vacations in an office, workshop, or in the field. Your experiences there cannot fail to be of interest to your fellow members.

After the reading of each paper all are invited to take part in the discussion which follows. It is by this means that much of the real value of the paper may be brought to the surface. If you do not understand any point ask about it. Probably the writer of the paper has, in the course of its preparation, had to ask himself, and has himself had to answer this very question, and will therefore be in a position to appreciate your difficulty.

As an additional inducement to the students to write papers the Council of the School allow marks for each paper read by an undergraduate. The maximum number of marks obtainable is 100, and all over 50 are considered in granting honors.

By a recent change in our constitution the editor of our Pamphlet is now appointed by the Council of the School. Since the adoption of this change we have had every reason to be satisfied with its operation. We congratulate ourselves on the appointment of Mr. Angus to this most responsible position.

I wish to refer to the recent appointments to the staff of our school, namely: J. W. Bain, B.A. Sc., Fellow in Metallurgy and Assaying; F. N. Speller, B.A. Sc., Fellow in Analytical and Applied Chemistry; and R. W. Angus, B.A. Sc., Fellow in Electrical Engineering. On behalf of the Society I congratulate these gentlemen on their appointments, and promise them the loyal support which has always characterized the attitude of an S. P. S. man towards his instructors.

And now, gentlemen, in closing let me exhort you to be loyal to our school, loyal to our staff, loyal to our society, and loyal to ourselves and the profession of which we are looking forward to becoming reputable members.

H. S. CARPENTER.

TORONTO, Oct. 13, 1897.

THE MODERN BASIC OPEN HEARTH PROCESS.

By F. N. Speller, B.A. Sc.

A subject dealing with such an important material as structural steel requires little in the way of introduction before a society such as this, neither will it be necessary to take up the time in proving the value of a study in detail of the manufacture of this material for all engineers and architects, or the mistake of leaving the subject wholly to the province of the metallurgical engineer. However, to the latter class, and to those specially interested in the advancement of this branch of practical science, it affords an unparalleled example in its development, to the present state, of the achievements made possible by chemistry and mechanical engineering applied together.

The varieties of steel are so numerous, and the effects of extremely small quantities of certain impurities so fatal, that the question of specifications which will ensure absolute safety for any special case becomes one of vital importance. Two classes of tests must be specified—separate and yet intimately related—dealing with first the chemical composition, and secondly, the physical properties of the metal.

It is not intended to discuss specifications in this paper, although much useful discussion might be provoked thereby; however, the short-sighted policy of indifference to the chemical side of the question so long as certain physical tests are fulfilled, for sometime manifested by some is fortunately becoming a thing of the past, perhaps when the engineer did not take the trouble to inform himself on all points of the treatment of steel from the ore to the finishing mill, this wise indifference was best.

Steel may be made in the crucible, Bessemer converter, or in the open hearth furnace, of exactly the same chemical composition in C, P, Mn, Si, S, etc., and yet possess radical different properties and uses. It is for chemistry to settle the cause of this phenomena.[*]

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[*] Oxygen, hydrogen, and nitrogen have been found in various proportions in steel, and their presence in varying proportions in Bessemer, Open Hearth, and Crucible steel, has led to the conclusion that they have some essential influence over the

Again, a steel may be dangerously high in phosphorus and sulphur and yet pass the ordinary physical tests. Cases of failure have been recently studied by polished cross-sections which showed numerous micro-flaws around the normal crystals of the steel—these being invariably accompanied by extra high phosphorus or sulphur. In the testing machine these steels might be manipulated so as to pass muster, but when in position vibration gradually enlarges these flaws until under some sudden strain for which the factor of safety was supposed to provide, fracture occurs, mysterious breaking of rails, shafts, axles, are often examples of this. Nothing seems to be proven clearer by long experience than that carbon is the only hardener to be tolerated in steel for general use, and that everything else must be reduced to the lowest possible limit, especially phosphorus.*

Bessemer inaugurated the reign of the age of steel in 1856 by announcing his process. He early experienced the deleterious effect of phosphorus which the operation in the converter would not

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properties of the metal; this hypothesis being deduced from the fact that when all other foreign elements are present in the same proportion in each, there still remains a decided difference in properties depending upon the mode of manufacture.

The Bessemer converter affords steel the best chance to take up these gases, as they are all three forced through the molten metal in large quantities; in the Open Hearth the metal is more or less protected from the gases passing through the furnace, and we would expect to find less present in the finished metal, while in the crucible where the steel is entirely isolated from such contamination, the least quantity of these gases should be found in the metal.

The determination of nitrogen, oxygen and hydrogen in the laboratory is accompanied by peculiar difficulties which have never been satisfactorily surmounted, however, careful examination by the best known methods has shown the foregoing hypothesis to be relatively correct, and that variation in physical properties follows variation in content of oxygen, hydrogen and nitrogen, just as the effect of manganese, sulphur and phosphorus manifests itself.

* Prof. Roberts-Austen has pointed out the marked relation which exists between the atomic volumes of those elements, which when present in very small quantities in metallic gold increases or decreases, the tenacity and extensibility of the metal, i.e., the elements which increase this property of gold have the same or a lower atomic volume, while the elements which have the opposite effect, have invariably a larger atomic volume.

Similarly in steel the foreign elements can be classified distinctly into two groups, according to their atomic volume:

I.		II.	
C (as diamond)	3·6	W	9·6
Mn	6·7	Si	11·2
Ni	6·9	As	13·2
Cu	7·1	P	13·5
Cr	7·7	S	15·7
Fe	7·2	O	17·0
		Al	10·7
		Sn	16·1
		Sb	18·2
		Bi	21·1

remove, but fortunately happened on Swedish pig iron, low in phosphorus and high in manganese. Although Bessemer steel at this time cost $210 per ton, it was rapidly coming into use.

The demand for steel thus created acted as a stimulant to inventors, and numerous new processes were brought on the field, but the only one which has achieved commercial importance is that of Sir Wm. Siemens.

In the year 1861 Wm. Siemens made the regenerative principle applicable for the production of high temperatures, using gaseous fuel. In 1868 he succeeded in making steel by decarbonizing cast iron with iron ore.

Martin had a few years previous to this produced steel by dissolving wrought iron scrap in a bath of cast iron on the open hearth of a Siemens furnace; a combination of these two operations gave us the Siemens-Martin process and its modifications.

The next turning point in this short history of progress was the discovery in 1878 of the basic principle by Gilchrist and Thomas, the essential condition of which was, to provide a basic cinder containing in the neighborhood of 50 per cent. CaO, which takes up the phosphorus when it has been oxidized to P_2O_5, forming a phosphate of lime which is stable, and is retained in the cinder, and thus permanently removed from the metal.

This was of great importance in England, allowing as it did the use of native high phosphorus iron in a basic-lined converter.

We are referring to the effect of small quantities of these elements and their compounds in relatively large masses of iron.

Those in Group I, confer useful properties to iron, or have no effect when present in moderate quantity; thus copper when present up to as much as 0·6% has little effect on iron, either hot or cold, provided the phosphorus and sulphur are low. There are large quantities of steel rails laid in the eastern States, and doing good service to-day with this amount of copper in them.

The elements in Group II. either by themselves or in combination are injurious to the useful properties of steel, rendering it hot or red short, thus causing trouble or failure in rolling, or through cold-shortness destroying its power to withstand vibratory strain.

Exceptions must be made in the case of very small amounts of silicon and aluminum introduced purposely to facilitate pouring, they have a quieting effect on the molten metal, it is supposed by removing the excess of oxygen and other gases, or rendering them more soluble.

Antimony and bismuth when present in copper in the proportion of 1 part in 1,000 have a very deleterious effect on the metal, rendering it useless for many purposes. It will be noticed that the atomic volume of these metals is nearly three times that of copper, it seems as though the presence of a comparatively few such atoms so disturbs the equilibrium of the molecular structure as to completely alter the physical properties.

The same principle was soon applied to the open hearth furnace, simply by building in a basic or neutral lining, capable of resisting the action of a highly basic slag.

This lining is made of burnt dolomite or magnesite, mixed with tar and rammed into place. It must be understood that the lining is essentially neutral or passive, and takes no part in the removal of phosphorus. It is the function of the lining simply to provide a strong support for the charge, and to be indifferent to the chemical action being carried on on the hearth, thus being free from corrosion by the metal or slag. These desirable conditions are only partially fulfilled owing to necessary restrictions in choice and cost of material.

Bessemer deservedly receives great credit for the complete form in which he delivered his process to the world, very few changes, and those only in minor mechanical details, have been made over his original ideas. Sir Wm. Siemens, known by reputation in so many departments of science by his genius and inventions, has by indefatigable sound scientific work given the open hearth process complete to us to-day, and further improvements in America are thought by practical experts to be in the line of his later patents.

In considering the construction of the open hearth furnace we will start with its most essential feature, the regenerators. These consist of four chambers filled with checker work, placed below the level of the hearth, connecting the furnace with the chimney, two are for the use of the incoming gas and air, and the others for the exit of the products of combustion to the chimney. In passing through the checkered brick-work the waste gases give up their excess of heat to the bricks and enter the bottom of the chimney shaft at about 300° C.

On reversing the air and gas currents, these acquire the temperature of the heated checkers before entering the hearth, which increment of heat is added to the hearth at every reversal of the valve. After some time the temperature of the checkers reaches about 1000° C., at which it is kept by regulating the supply of air so that now the excess of heat produced above that used up in the furnace is made up by radiation.*

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* On working out the calorific equation for the O. H. furnace we arrive at the results below from the following data:

Natural gas used per ton of iron produced, 6000 cubic feet
Temp. of chimney gas, 350° C above ord. temp. of air.

Specific heat of these gases........................	.268 (average)
" " " air	.237
Latent heat of fusion of pig metal..................	33 cal.
" " " " steel......................	43 "
Average spec. heat of iron up to...............1500° C=	.180 (approx.)

The gas is reversed every 15 minutes and a temperature can easily be reached at which the purest silica sand will melt, the melter has to constantly keep his eye on the roof and walls and check the gas when the heat shows signs of melting the brickwork. Where natural gas is used it is not regenerated owing to its high calorific power.

The foundations of the furnace structure must be deep and strong, the furnace hearth being entirely separated from the checker chambers and built in solidly underneath to prevent loss of steel in case of the metal working through the bottom.

The walls and roof are subject to great strain and wear, a general thickness of nine inches silica brick gives the best service, various arch shapes are given to the roof, the exact shape between certain limits is non-essential. It was for some time supposed that the space above the hearth should be as small as possible, this soon caused great havoc with the low roof. Now the tendency is to have a high roof, protected from the direct flame, and utilize more completely the heat by radiation, not allowing the flame to impinge on the bath. This principle was proved by Mr. Fred. Siemens.

The furnace is built into a framework of steel sheet, channels, and bracing bars, so constructed as to give the best support against expansion and to preserve the shape of the parts.

The box through which the gas enters (when natural gas is used) the doors, and often the ports which provide entrance and exit for the gases are water-cooled, thus greatly increasing their endurance and that of the men in charge.

The ports through which the gas and air enter the melting chamber constitute the most important detail of construction, as

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TOTAL HEAT PRODUCED.

Heat from combustion of gas 168.5 CM x 8480=........	1,428,880 cal.
" " " of C, Mn, P, Si & Fe.............	143,000
	1,571,880

LOSSES.

From waste gases..	214,690
Allowing 10% excess of air	14,000
Absorbed in melting and heating............................	290,000
Radiation (by diff.)	1,053,190

The above calculation can only be considered approximately correct, owing to our uncertainty as to the specific heats at high temps; however, the fact that an empty furnace requires nearly as much gas to keep the temperature at working heat as when the charge is in, proves the general truth of the above figures, which show that the radiation at this temperature must amount to about 67% of the available calorific power of the fuel.

on their stability depends the life of the furnace. Having to stand the full force of the gas as it burns and sweeps across the hearth, carrying with it particles of oxide of iron and slag, they must be made of the most refractory material obtainable, generally chrome ore (with 45 % $Cr_2 O_3$) or magnesite brick. The shape of the ports determines the direction of the flame across the hearth; faulty ports often allow the flame to shoot upwards toward the roof with disastrous results.

The silica brick of the walls and roof and other parts must be protected by a passive joint of chromite wherever it comes in contact with the basic part of the furnace, and if this joint is subject to pressure it must be protected from undue heating, the reason of this being of course the certainty of a fusible silicate being formed by the acid and basic materials present.

The operation on the hearth. When starting a new furnace, heat is applied slowly until the required temperature is reached, two and a half or three days after lighting—charging now begins.

The success of the modern open hearth process will depend largely on the ease and quickness with which the large amount of material is handled. Charging boxes are placed three on a buggy and are loaded in the stock yards by hand; in modern mills this is the only place where hand labor need be employed. The buggies with their boxes are run down a track in front of the furnaces lying between the charging machine track and the furnace front. Many experiments have been made to do away with the costly and arduous labor of hand charging, resulting finally in an electrical machine which works with such despatch and ease as to be really all that can be wished for. This machine can be run from one furnace to another and has four separate motions controlled by electric motors. The end of a long ram can be automatically attached to one end of the charging box, which is lifted with its load, run into the furnace, turned over, brought back and placed empty in its place on the buggy. In this way a furnace can be charged with fifty tons of material in less than thirty minutes. (See plate 1.)

The limestone in pieces of ten to twenty pounds, is charged first with a little ore: on this is dumped the pig metal, with some regard to distribution. For a fifty-ton furnace about which we are speaking the charge would consist of: limestone, 6,000 pounds; ore, 500 to 1000 pounds; metal, 45,000 pounds.

Gas is now turned on strong, and the furnace allowed to heat up to melting temperature, this taking about forty-five minutes. It is

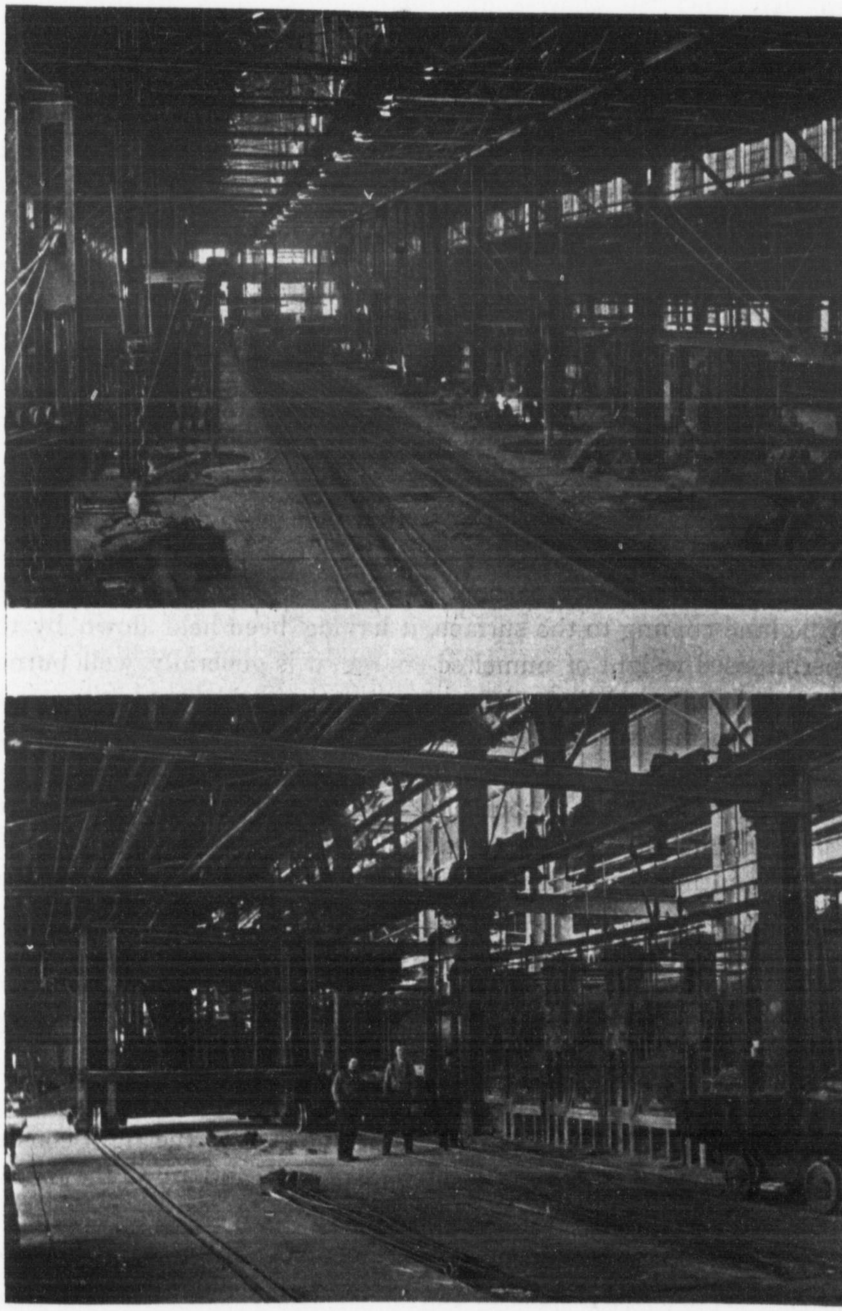


FIG. 1.

now ready for the charge of scrap iron, which will amount to about 47,000 to 50,000 pounds, and consist of old rails, crop ends from the mills, Bessemer waste, tin plate scrap, nails, broken or worn out machinery, in fact any iron low in carbon and valueless for any other purpose will be gladly received.

When the furnace is charged the operation of melting begins, with a high temperature, the flame being so regulated as to secure complete combustion over the centre of the bath ; the limit to which the temperature may be raised depends on the fusibility of the brick and the general correctness of the lines of the furnace. As stated above, the ports have more to do with the direction of the flame than any other factor, and as they wear away, and the checker work of the regenerators becomes clogged and the passages obstructed the efficiency of the system is considerably impaired, until the furnace has to be let out for repairs.

During melting the impurities are gradually oxidized, starting with silicon, which combines with iron and oxygen, forming a light foamy slag. A slight ebullition around the side of the hearth at this point is caused by oxidization of carbon. The end of this stage is marked by the lime coming to the surface, it having been held down by the superimposed weight of unmelted charge, it is generally well burned by this time, although dark pieces of undecomposed limestone occasionally show themselves bobbing to the surface, and floating around in such a manner as to receive the designation of black ducks, which is another example of the resourceful imagination of the iron smelter in connection with the barn yard. The liberated CO_2 from the limestone, although absorbing some heat, acts as an oxidizer, giving up the whole or part of its oxygen to the impurities of the charge.

The third stage begins with this boiling on the lime, the partially formed slag and molten metal being thrown violently upward for a foot to two feet, facilitating rapid oxidization. The melter prepares to get his "slag in shape" as soon as this boiling subsides, showing that the lime is all up. This is an operation involving great skill and experience, it simply means getting the slag of the right consistency and composition to retain the phosphorus, and aid in the oxidization of other impurities.

The charge now is engaged in a quiet boil due to oxidization of carbon and the escape of the products of this reaction. A test is taken, *i.e.*, a small quantity of metal is removed in a hand ladle, poured into a mould and broken, by the peculiar grain of the fracture the melter estimates the contents in carbon and phosphorus. It is now five or six hours since charging commenced, the phosphorus is usually still too high, although fifty to seventy per cent. has been removed.

If the carbon is still high a few thousand pounds of hæmatite iron ore of pure quality is added, it gives up its oxygen to the carbon, and other impurities which are removed, causing a rapid boil and flickering blue flames to play over the surface due to the burning of CO. Tests are taken at intervals and the reduction regulated, when the carbon is down to 0.18% the addition of ore is stopped and the charge allowed to slowly bring itself down to .08 to .10% C, when it is ready for tapping. While ore is being added the gas is partially turned off, as a good deal of heat is generated by the oxidation going on.

If, however, on taking the first test the carbon is low and the phosphorous high, the heat is said to melt low, pig metal is added to supply the bath with carbon, thus preventing the oxidization of iron while the excess of phosphorus is being reduced. The phosphorus has been found as high as .090% after melting, when the carbon was as low as .07%.*

If the carbon is now down to .08% and the phosphorus below .030%, the heat is ready for tapping, providing the slag is of the proper consistency to run easily from the furnace and the heat at the proper temperature, which is usually estimated by the eye or by the rate of solution of an iron rod placed in the bath.

A thin slag can be thickened by an addition of burnt dolomite or limestone, also if the slag be too thick an addition of fluorspar will confer fluidity.

The sulphur is not removed to more than forty per cent. of that originally present ordinarily, but if the heat can be held in the furnace for two or three hours, the sulphur can be reduced to a very low point by treatment with calcium chloride and calcium fluoride.

The removal of phosphorus is controlled primarily by the amount of SiO_2 in the slag, and secondly by the per cent. of CaO present. The SiO_2 may vary considerably from 14 to 20% perhaps, but the lime usually runs about 45 to 50%; only 1.25 to 1.50% silicon can be per-

* The increasing demand for very soft low carbon steel (0.1% and less) involves an element of danger if carried to an extreme, as when the bath gets so low in carbon there is a considerable risk of the iron itself being burned and the steel containing more oxygen than is desirable or safe. Sufficient ductility can be secured by reducing the other foreign elements to a minimum and allowing an extra point or two of carbon to protect the iron.

mitted in the pig metal used supposing it to constitute half the charge, by procuring pure limestone, however, as free from SiO_2 as possible the silicon, of the metal may be raised to a limit dependent on the amount of slag which is required. The advantage of deriving all the silicon required from the pig metal is apparent, the higher the silicon the lower the sulphur, which cannot be cheaply removed in the open hearth or in any process yet devised, except in the blast furnace.

A certain amount of iron (about 12 %) and considerable manganese is oxidized and enters the slag, increasing its fluidity.

Just before tapping 80 pounds of ferro-manganese (carrying 80 % Mn) is added to the bath, to take up and remove excess of oxygen in solution, which would render the heat wild in tapping, — the manganese originally present in the charge has been reduced to about .15 to .10 %.

The tapping hole is carefully pricked and the metal and cinder run down a trough into the ladle (see plate 2), which holds the metal and the covering of about eight inches of cinder, the rest of the cinder runs over and settles in the pit below. Fifty pounds of ferro-silicon is added to the steel as it runs into the ladle along with enough ferro-manganese and anthracite coal or coke dust to give the carbon and manganese ordered for that particular steel. The ladle in which the steel lies is conveyed by a capacious truck to the end of the building (see plate 3) where it is lifted up by a massive hydraulic crane swung around over moulds arranged on buggies into which the steel is run by raising the stopper. Very often, especially where large ingots are required, the steel is poured into moulds arranged in a pit surrounding the ladle crane carrying the ladle (see plate), these large ingots are generally bottom cast, i.e., the metal flows down a fire-clay pipe enclosed by a strong iron casting and proceeds by two horizontal branches filling the moulds from the bottom, this facilitates greater soundness and freedom from blow holes in the ingots.

The total loss of iron in making steel by this processs amounts to 4 or 5 %.

The acid and basic open hearth processes have an entirely different relation to each other than the acid and basic Bessemer process. In the latter process the metal used must be either low or high in phosphorous to admit of successful working, while in the acid and basic open hearth the same stock can be used for both if the phosphorous and silicon be low. It is possible to remove any quantity

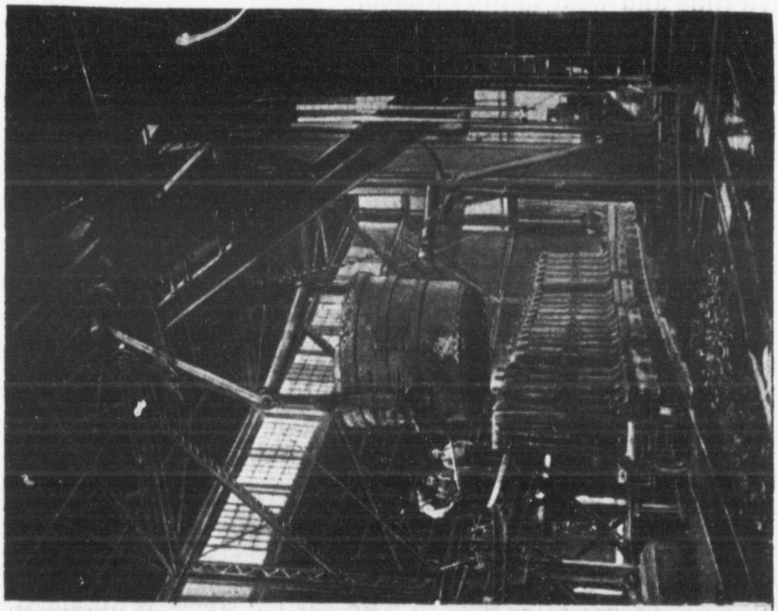


FIG. 3.

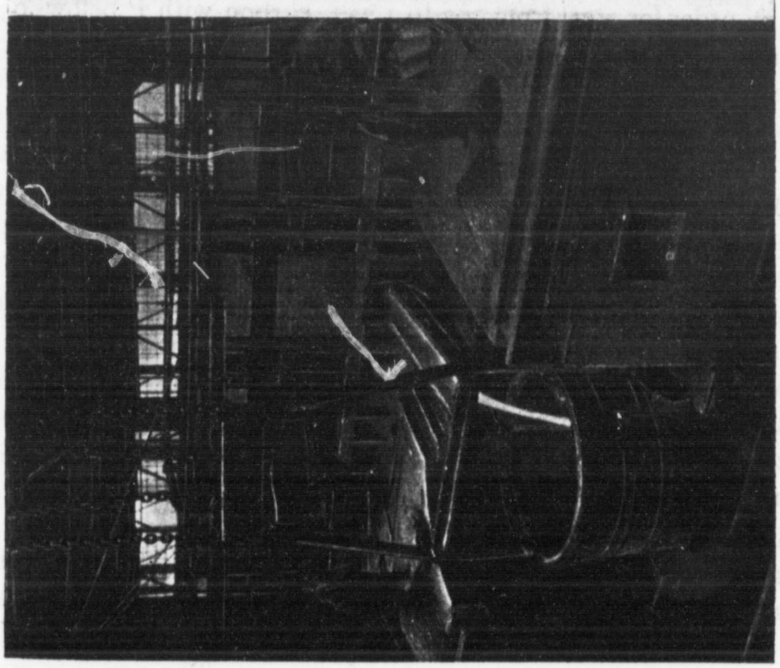


FIG. 2

of phosphorus usually met with in pig iron by the basic open hearth, but it is generally more economical to use a good quality of metal, and save time and wear in the furnace.

Open hearth steel can hardly compete with Bessemer in price, even under the most favorable conditions, although in the great mills where special facilities exist for handling material, economy of manufacture has brought the two processes very close together in this respect. The difference in cost of manufacture is almost wholly made up in the increased length of time the open hearth charge is retained in the furnace, the extra cost of machinery in the Bessemer plant balancing the fuel required for the open hearth furnace. The superiority of open hearth steel for special structural parts, axles, tires, springs, boiler plate, armor plate, etc., has been proved by experience, and by the same process we have learned to distrust Bessemer steel for such service. This must not be forgotten, even though the chemical and physical tests as far as they go show no difference.

Engineers are gradually recognizing the advantage of this steel where a large quantity is required of a certain composition within narrow limits; a high degree of uniformity is ensured, and further, greater homogeneity, especially in high carbon steels, is obtained.

Any order for steel from .07 to 1.25% carbon with P below .030% and sulphur below the same limit, and any manganese required, can be filled in the open hearth department, as the heat is under perfect control from the start and can be manipulated at the will of the melter. In making high carbon spring steels up to 1.25C, special quality pig iron is used and the heat melts high; when laboratory tests show the carbon to be down to .90 or 1.00%, the heat is tapped and recarburized thirty or thirty-five points to the required temper.

It will not be possible within the limits of this article to go into those modifications of the open hearth process which show more or less promise of success, such as the Bertrand-Thiel process and others, but the direct process in connection with the open hearth furnace should be mentioned, as it has been tried on the large scale and promises great success.

A mixer, which is simply a large brick-lined vessel revolving on rollers, is filled with about 250 tons of pig metal brought while molten at intervals from the blast furnaces, which may be several miles away. A uniform supply is thus kept at hand and drawn off as wanted for charging the regenerative furnaces. The action is much more violent when using this hot metal than when all the stock is cold, and a heat of fifty tons, which, by the independent method would take eight hours to work, can be tapped five hours after charging.

One department of the works should not be passed without a reference, viz., the most essential chemical laboratory. It can never be supplanted where a reliable check is to be kept on the operations and is maintained in operation night and day. The great amount of information and data collected on the chemistry of iron and steel has done much to improve the quality and lower the cost of reliable material.

The growth of the open hearth industry in the last few years is best illustrated by a few figures collected from official returns:

	Bessemer ingots.	O. H. ingots.
GREAT BRITAIN.		
1890	2,047,080 tons.	1,589,227 tons.
1896	1,844,896 "	2,354,636 "
UNITED STATES.		
1890	3,688,871 tons.	513,232 tons.
1896	3,919,906 "	1,298,700 "

As the production of Bessemer steel has been practically stationary for the past six years, the future advance of the open hearth seems assured, due especially to the yearly decrease in the price of production, and to the further advancement of analytical chemistry.

A large amount of scrap is required for the above process. As much as seventy-five per cent. can be used in the furnace charge. On the other hand, such is the independence of the process, that good steel can be made of pig iron and ore alone; however, in many manufacturing districts scrap iron and steel is a drug on the market, and if a small quantity of pig iron were available, and suitable fuel for producer gas, or better, natural gas at hand, a small open hearth plant might be run with profit. The state of the iron and steel industry is one of the surest gauges of the general commercial prosperity of the country. Is there no such field of industry awaiting development in Canada?

ROAD CONSTRUCTION.

By A. W. Campbell, C.E., Provincial Road Commissioner.

I had the pleasure of addressing your society about a year ago on the subject which at present is occupying my time. If I remember rightly, my talk was quite as much on the management of councils, as on the management of roads. I recall, too, that your Principal, Prof. Galbraith, suggested afterwards that he had expected me to discuss rather the practical construction of roads. It is said that in a certain Ontario town, the fathers of the community were discussing the ways and means of laying a suitable roadway on their main business thoroughfare. The financial difficulties seemed insurmountable. One tenacious member of the council finally expressed his belief that if they would "put their heads together" a plan would be devised. His suggestion, it is said, was acted upon, and the result was wooden block pavement. I rather fancy that were Prof. Galbraith to undertake the work of a city engineer, he would see very clearly the connection between the matter of my address and the practical designing of streets. It might have been described under the head of paving material.

It is, however, a material so seldom commented upon, in works on paving, and as a practical discussion was desired, I really could conceive of no department of a more practical nature. It may be that familiarity with the real work of road construction has caused me to underestimate the difficulties, but I am more strongly convinced to-day than I was perhaps a year ago, that good business ability, combined with the ability to deal with councils, must be the most important part of a municipal engineer's qualifications. However, this afternoon, I will try to be as practical as possible and will discuss a few points in the actual work of constructing a broken stone roadway for a town street.

Staking Out Work

To lay out the work; that is, to place the necessary stakes for the guidance of the foreman and workmen, is a matter to which I had better refer. It may save someone from falling into an error similar to one which came under my notice several years ago when I was employed on a drainage dispute. I found that the township engineer who had been called upon to report on the cleaning of the drain had placed the grade pins along the bottom of the ditch. How he expected to have the ditch deepened without at the same time disturbing his stakes, I did not inquire.

I have found the most convenient method to be, having decided upon the width of roadway to be paved, to place stakes at the street intersections on the curb line. An offset is then made, back of the boulevard from these stakes a convenient distance, and a line of pickets is planted, along which the foreman can sight in guiding the excavation to the curb line. With a shorter offset of about three feet from the curb lines as indicated by the stakes at street intersection, grade pins should be planted on one side of the roadway, from the top of which pins the depth of excavation should be noted, so as to form a perfect cross-section.

In taking levels from these grade pins, the rodman first places the rod on the grade pin, then, as nearly as he can judge, on the line of the curb; then in the centre of the roadway; then on the more distant curb line. From these levels, the grade and cross-section of the roadway and the necessary cuts and fills can be established. Sufficient earthwork has to be done to bring the finished surface of the roadway to the required elevation, and in connection with this, provision must be made for deeper excavation at the side than at the centre, so as to allow the earth sub-grade to conform in part to the crown of the roadway. The first set of grade pins will answer for rough work, when the main part of the earth is being removed by plows, scrapers, etc. A second line of grade pins may be placed in the centre of the roadbed when the excavation is nearly completed, in order to bring the earth sub-grades to the exact elevation; or if a greater degree of refinement is necessary, grade pins may be planted close to the curb line as well, making three lines of stakes in the roadway. The surface of the earth sub-grade, it has been said, should conform in part to the surface of the finished roadway—that is, it should be rounded up to the centre, or crowned with a rise slightly less than that of the finished roadway—less, in order that a greater thickness of broken stone may be placed on the centre of the roadway than on the sides.

If there are established grades, the first thing for the engineer to do in laying out the street, is to take levels on the street and estimate the cuts and fills necessary in order to reduce the street to this grade. There will, of course, be bench-marks, usually a water hydrant, a door step or other stable monument from which he may deduce the grade. If it is necessary for the engineer to himself choose a proper grade, he will have to do so, not on the arbitrary lines of what should be, but as circumstances will permit. That is, he must not injure property by excessive cuts and fills. At the same time, in making the necessary excavation to receive the roadmetal and reduce the grades to an easy slope, he may find an almost alarming amount of earthwork necessary, but by terracing a boulevard, this may sometimes be safely done, especially if the street is a residential one.

GRADES

The first point to consider is the grade of the street. If the town is a properly conducted town, (as very few in Ontario are, unfortunately), there will be in the engineer's office plans showing the established grades of the streets. These established grades are those which have been planned by the engineer from levels taken of all the streets, which plan has been accepted or established by the council. They are the grades to which the streets will be reduced when such improvement is deemed necessary by the council for paving or other purposes. By their establishment in this way they constitute a warning to citizens in erecting houses on the street and gas and water companies in laying pipe. For damage is frequently done to property by cutting away a hill or filling up a hollow in order to properly level the street, especially if houses, stores or offices have been erected. And this warning is the means of saving the municipality from the liability to pay for damages which might have been avoided by the citizen if he had recognized this established grade in subsequent improvements on his property.

DRAINAGE

The next point to consider is the laying of underdrains. These are of ordinary field tile, laid in the usual manner. The necessity of drainage is a matter which cannot be too forcibly pointed out. Without it no pavement, whether asphalt, brick, stone, block or broken stone can be a success. A soil, like a sponge, always retains in its texture a certain amount of moisture. Water falling in the form of rain, sinks to the first impenetrable strata, and from there rises higher and higher until it finds a lateral outlet. Underdrains supply such an outlet at a sufficient depth below the surface of the road, to preserve a firm strata of the natural soil. The natural soil has in every case to support the weight of traffic. The paving material merely forms a covering to shed water from the roadway and prevent it sinking into and softening the natural foundation beneath; that is its more important duty, that of secondary importance being to form a wearing surface. Drainage is a fundamental feature of all good pavements. If sewers exist with capacity for storm water—and I am not so enthusiastically in favor of a separate system as are some—they will materially aid in providing outlets for the underdrains, as well as surface gutters.

A system of separate sewers has its advantages, and is the only one which can be used in some instances, but the advantages derived from more perfect street drainage will in many cases more than compensate for the disadvantages of the combined system. If sewers do not exist, drains should be emptied as often as convenient into natural water courses. One secret of successful surface drainage is to dispose of water in small quanties before it gains force and headway. With underdrains, a certain amount of flushing in this way is not objectionable. The tile should ordinarily be four inches in diameter, and placed below frost line for the best effect. This is seldom done, however, two and one-half or three feet being the usual depth. The fall should average one foot in 100; the minimum never less than one in 1,000 feet. It is well to fill over the tile with gravel or some other porous material, especially if the subsoil is a stiff clay. In quick or water sand, sawdust is the best material to place around the tile to prevent the silt passing into and choking the bore. Care must be taken to see that the tile is placed with a constant and uniform grade.

ROLLER AND ROLLING SUB-GRADE.

The sub-grade and under drainage completed, the roadway will be ready for its first rolling. A heavy steam roller weighing from ten to twenty tons is indispensable in making a good broken stone or other pavement. A greater weight than twelve tons is apt to be too heavy on loose stone or earth, unless consolidation has been first commenced by a lighter horse roller of six or eight tons. The natural earth should be compressed until perfectly smooth and solid, so that when the broken stone is placed on it and rolled, the stones will not be forced down into the soil. It is important to preserve a smooth, hard sub-soil, as drainage is thereby greatly assisted.

CURBING

The subsoil and drainage completed, the curbing may be put in place. Curbing defines the roadway, forms the gutters, protects boulevards and sidewalks, and keeps the road metal in place. The best material for curbing is flagstone four to eight inches in thickness, each stone to be not less than three feet in length and about eighteen inches in depth; greater dimensions are preferable, as there is then less liability to disturbance. The curb should not be set so high that water cannot flow readily over it from the sidewalk and boulevard into the gutter. A good substitute is cedar plank. It should be spiked to six inches in diameter cedar posts, two and one-half feet in length, which are placed at intervals of eight feet and bevelled at the top, with an incline from the roadway so that the curb will slope toward the boulevard. While wood is extensively used for curbing in most cities of the Dominion on the less important streets, yet where stone is plentiful and easily obtained, its greater durability and better appearance will recommend its general use.

PLACING THE STONE

The curb provided, the broken stone may be placed in the roadway. It should be spread in layers of not more than four inches thickness, and each layer thoroughly rolled. The stone should have been separated into grades according to size. The largest stones should be such as will pass through a $2\frac{1}{2}$-inch ring, and the smallest, consisting of the screenings, should be used as a top dressing and binder. One of the most serviceable of modern road-making implements is the stone crusher; this, with the aid of a screen attachment will break and separate the stones into the required grades ready to be placed in the roadbed. The value of having the stones of uniform size in each layer is not sufficiently appreciated. If of irregular sizes, the smaller ones at the surface wear more quickly than the larger, and the rough surface is the result. The large stones, moreover, have a less uniform bearing, and a horse stepping on the edge of one will loosen and throw it out of place, thus leaving it to roll under the feet of horses and wheels of the carriage.

THE BINDER

A matter in which there is considerable diversity of practice, is in the use of a binder. A binder, I may explain, is some fine material, generally sand or gravel, which is spread over a layer of stone on the roadbed, and is then flushed or harrowed into the interstices so as to fill the vacuum, and for a bond between the stones when the layer is rolled. It is, I have said, generally sand, but in some cases road-makers have used clay, or even ordinary dirt. I was recently shown a binder used largely on the streets of St. John, New Brunswick, which was merely a red clay. The majority of engineers, however, who recommend this kind of a binder, specify " clean, sharp sand."

A layer of stones such as will pass through a 2-inch ring, loosely spread on the roadway, will have spaces between the individual stones amounting to about one-half the entire mass. It is the province of a roller to compress the stones until a smooth, hard roadway is formed. The heavy weight of the roller will, in consolidating the layer of stone and in wedging one against the other, wear off sharp angled edges. After such a layer of clean broken stone is thoroughly consolidated the vacuum remaining is reduced to about 33% of the mass, but varies according to the toughness or softness of the stone. A hard trap rock will necessarily pulverize and yield less under the pressure of a roller, than will a limestone rock. So that the vacuum remaining with the former will be greater than with the latter.

Personally, I have always preferred to use for a binder the stone chips and dust, known as screenings, which is created in the crushing of the broken stone, and the less foreign material incorporated in the roadbed the better. The value of any stone used in a roadway is largely dependent upon the degree with which its dust will cement and re-cement in the roadway. For this reason the utility of limestone and trap rock is not by any means proportional to the difference of hardness of the two stones, since pulverized limestone will form a cement more readily than will the dust of trap. Sand or any foreign material mixed with broken stone prevents this cementing process. It prevents, moreover, the firmer mechanical clasp which one stone takes upon another as the result of rolling. It helps to retain water in the texture of the roadway, rendering it more liable to injury from frost. The result is that in the spring or in wet seasons, the roadway becomes spongy, the sand oozes to the surface and has merely to be carted away. Its only advantage is that, with less rolling the road material can be consolidated into a hard (but not so durable) pavement. Clean material should always be used in the construction of a broken stone roadway; and this implies that sand, gravel or clay should not be used as a binder, dependence being placed upon the screenings and the roller.

TELFORD FOUNDATIONS.

The roadway which I have thus briefly described is one based on the macadam system. A slightly different plan, and one used in the more durable classes of construction, is a modification of the Telford system, in which the first layer of stones partakes more of the nature of a foundation. Large stones are placed on edge by hand, in regular rows across the roadway, the stones in the centre being larger than those at the side so as to follow the ultimate surface of the road. The projecting points of these stones are then chipped off with a stone hammer, and are wedged into the interstices. On this foundation the broken stone is placed in layers after the manner first outlined. The Telford system is of most service in laying a roadway over a damp subsoil difficult of drainage.

HISTORY AND CHEMISTRY OF THE CYANIDE PROCESS.

By W. W. Stull, Graduate S. P. S.

HISTORY OF THE PROCESS

It has been stated that jewelers and alchemists of the middle ages were aware of the fact that gold, in a finely divided state, was soluble in a solution of potassium cyanide; and that they made use of it in the gilding of metals. In the year 1806 Hagen stated the same fact, and an English patent (No. 8447) issued in 1840 by Elkington, shows that he made use of cyanide in his galvano-plastic operations. Prince Bagration in 1843 gave notice that the gold was more soluble at the surface of the solution, or where there was a free access of air. In the year 1846 Elsner gave the equation for the reaction as follows:

$$Au_2 + 2KCy + H_2O + O = 2KAuCy_2 + 2KOH.$$

For the next few years many chemists and metallurgists such as Faraday, Glassford and Napier experimented with it, and have given us their results. In 1867 a patent was issued to Julia H. Rae, of Syracuse, N. Y., (Feb. 5, 1867. No. 61866 U. S. A.), which made use of potassium cyanide in connection with an electric current. This, however, did not prove to be economical in practice, and hence never got beyond the experimental stage. Jerome W. Simpson, Newark, N. J., who was a very energetic metallurgist, obtained a patent in 1885 (July 28, 1885, No. 323222 U. S. A.), making use of potassium cyanide, ammonium carbonate, and sodium chloride for the extraction of gold and silver from their ores. He evidently used the sodium chloride to chloridize a portion of the silver and thus render it more susceptible to the action of the cyanide. He stated that the cyanide solution was to be very weak, and that zinc might be used in the subsequent precipitation of the gold.

From this time till the present a great number of patents have been issued, making use of potassium cyanide for the extraction of gold and silver; many also for the precipitation of the same metals from the solution. Among these the most important is that issued to Messrs. J. S. McArthur and William Forest (No. 14174 Eng., Oct. 19, 1887). After getting their patent in England they applied for patents in all countries where gold was found and which issued patents. Owing to misunderstanding regarding their claims they were unable to obtain an American patent till 1889 (May 14, 1889, No. 403302 U.S.A.) Their claim was worded as follows:

" Having fully described our invention, what we desire to claim and secure by letters patent is:

" The process of separating precious metal from ore containing base metal, which process consists in subjecting the powdered ore to the action of a cyanide solution, containing cyanogen in the proportion not exceeding eight parts of cyanogen to 1000 parts of water."

They state in their patent, also, that the weak solution is used so as to have a selective action on the metals. In other words if a solution containing a considerable quantity of the cyanide were used, it would act upon other substances which might be present in the ore.

Some time after this patent was issued to McArthur & Forest they obtained patents on the use of zinc, in the precipitation of the gold from the solution, also for the use of caustic alkalies for the neutralization of acid tailings. Zinc had been used for this purpose before, as may be seen by reference to Mr. A. P. Price's patent, April 24, 1884, No. 5125 Eng., or to Mr. Simpson's, mentioned above. Regarding the use of caustic alkalies to neutralize the acid tailings, it might be stated that it was by no means a new discovery, but rather an application of a fact which had been considered too widely known to be patented. The issuing of these patents has caused considerable contention among metallurgists, however. McArthur and Forest must be given due credit for the way in which they introduced their process, which has since proven its value in nearly all countries in the world.

The first country in which the cyanide process was used to any extent was South Africa, in the Witwatersrand gold fields. In 1891 the Robinson mine erected works for the treatment of tailings, and in that year alone nearly $890,000 worth of gold was extracted. Since then many plants have been erected and millions of dollars' worth of gold extracted from the tailings, which had hitherto been considered worthless.

As soon as the great advantages of the cyanide process over all others in treating tailings in South Africa was established new details were from time to time introduced and patented. It is not the intention of the writer to enter into a detailed description and history of each, but rather to point the salient features in the development of the process.

Mr. Von Gernet stated that electrical precipitation of gold extracted from ores by cyanide was in use in Europe and Asia before the year 1888. Dr. Siemens found while electro-plating in Berlin that gold anodes lost weight while standing in a cyanide solution, without any electric current passing. This induced him to try the precipitation by the electrolytic method, and in 1888 he commenced operations on a large scale which were successful. In May, 1894, a plant treating 3,000 tons of tailings per month (at the Worcester mine, South Africa,) was completely successful while using this method. Many more have now adopted the same method, which has become a formidable rival to the zinc process. I might also add that since the introduction of lead peroxide as anodes many former objections have been done away with.

Another very important improvement has been introduced by Messrs. Sulman and Teed, who in 1894 patented their process (No. 18592 Eng.) This consists of introducing a small quantity of bromide of cyanogen (or any haloid cyanide) into the potassium cyanide solution. This has been done to give a greater activity to the solvent, thereby saving time and hence loss of cyanogen by decomposition. They have succeeded in introducing their treatment in at least two places. I might here state, in regard to the history of the process in Canada, that we have a plant at Deloro, Ont., where mispickel ore is now being treated successfully by the Sulman and Teed method. This ore could not be worked successfully by amalgamation or chlorination.

The Regina Mine in Western Ontario has been using the cyanide process for treatment of tailings, but I have not been able to get any definite information as to its success. I believe it is meeting with success, however, as it has been in use for some time and no complaints are heard.

The recent introduction of zinc fume as a precipitant is also worthy of notice. This is a decidedly cheaper and an easier way of treating the solution. Many improvements might be also given in regard to the treatment of the precipitates, but some of them will be spoken of later on.

Regarding the amount of gold extracted by this method alone, it is only necessary to say that in the first six years from the time of its introduction, $17,000,000 in gold has been won from ores previously regarded as valueless.

SCOPE OF THE PROCESS

This process can be advantageously applied to many gold and silver ores, and is often particularly adapted to those which are found to be rebellious to other treatments. Those ores in which the gold exists either in a very finely divided state, or in which it is rusty or coated with some foreign film, as sulphite or iron oxides, and which cannot be amalgamated, are easily treated by this process. The base metals which usually accompany these refractory ores are iron, zinc, lead, copper, antimony and arsenic, usually as sulphides or arsenides. These are dissolved to a certain extent, but by properly reducing the strength of the cyanide solution their action may be reduced to such an extent as to be practically not worthy of notice. Of course much depends upon the state of chemical combination in which they exist – which will be treated further on under Chemistry.

The principal exceptions are those ores which contain hydrated copper oxides, copper carbonate and antimony. If the copper compounds are soft and porous or spongy, they are particularly dangerous; while if physically hard, they are not acted upon so readily. In the case of the antimony ore it is not because the solution acts upon it, but rather that the gold is held so closely in contact with it as to be almost incapable of extraction.

The greatest success of the process has been, and is yet, in the treatment of low grade ores and tailings at a very low cost and in large quantities. It is on this ground that the cyanide process cannot be compared with any other, and yet takes its place as a very powerful one in metallurgy. Where the loss of cyanide is great in the treatment of an ore it at once precludes a commercial success, although a satisfactory percentage of extraction may be achieved.

CHEMISTRY OF THE PROCESS

As stated before, the solubility of gold in potassium cyanide was discovered early in this century, and the equation was given by Elsner in 1846, as:

$$2Au + 4KCy + O + H_2O = 2KAuCy_2 + 2KOH.$$

That is, a double cyanide of gold and potassium is formed. This salt may be obtained from the solution and on analysis answers to the above formula. According to the above equation oxygen is necessary for the operation. For a number of years this was a question on which a great many discussions took place. McArthur and Forest claimed till 1890 that oxygen was not necessary, but have since found out their mistake. Mr. Louis Janin, jr., claimed, in 1892, that Elsner's equation was not correct, and gave the following:

$$Au + 2KCy + H_2O = KAuCy_2 + KOH + H.$$

For which he states: "It certainly seems more logical that there should be a simple interchange between the gold and the potassium, forming metallic potassium, which is decomposed in the nascent state by the water of the solution, than that there should be another element necessary to the reaction. The theory of the equation is upheld by experiments made by myself, which showed that strongly concentrated solutions of potassium cyanide have less solvent energy upon metallic silver than weaker ones. This, I suppose (for I did not make any attempt to prove the reaction), was due to the polarizing action of the hydrogen." This statement in my opinion is extremely weak.

It is now known beyond a doubt that oxygen is essential to the success of the process, also that hydrogen is not given off. In a series of experiments on this subject by F. H. Mason, F.R.C., Halifax, he found:

1. That air passing through solutions of potassium cyanide considerably increases the rate of solution of the gold.

2. That air coming into direct contact with the gold increases its rate of solubility in potassium cyanide solutions.

3. That amalgam, on the surface of gold, protects it to an enormous extent from the solvent action of potassium cyanide.

The solvent power of a cyanide solution in practice depends upon the oxygen supplied from air (1) previously dissolved in the liquors, (2) entangled between the grains of ore or tailings, and (3) dissolved by the surface layers of liquid diffused through the mass during leaching. This accounts for the great activity exhibited by the solution at the commencement of the leach, and also shows that the supply of oxygen is limited.

Many chemists and metallurgists have been experimenting with several substances which might be used to take the place of the oxygen. It has been found that any of the haloid group of elements will do this quite successfully. Upon this fact is based the Sulman and Teed process in which they make use of bromide of cyanogen in small quantities in the ordinary solution. In the above reactions it was seen that there was one atom of potassium to be gotten rid of, and as there was insufficient heat generated to decompose water, the potassium combined with the oxygen to form K_2O, which immediately dissolved in the water to give KOH. Now when the bromide of cyanogen is present we get the following reaction.

$3KCy+BrCy+Au_2=2KAuCy_2+KBr$, or where bromine is present in a free state.

$$2KCy+Au+Br=KAuCy_2+KBr.$$

These reactions do not in my opinion represent the actual occurrence but rather the ultimate result of a series of actions. C. A. Mulholland gives (in the Engineering and Mining Journal, June 1, 1895), the following reaction,

$$4Au+8KCy+2Br+O_7+H_2O=4KAuCy_2+2KBrO_3+2KOH.$$

Now it is well known that the adding of free bromine greatly increases the rate of solubility, and as the amount of oxygen is much greater than if the bromine were not used it seems quite evident that this is wrong. Mr. Warwick thinks that the equation as given by Messrs. Sulman and Teed, the originators of the bromo cyanide process, is not a representative one, as it shows that it is independent of oxygen. He claims that when the potassium cyanide acts upon gold it forms auro-potassium cyanide, etc., as in Elsner's equation, and then the bromo-cyanide acts upon the double cyanide to form potassium-bromo-auro-cyanide as

$$2KAuCy_2+2BrCy+4KOH=KBr_2Au_2Cy+5KCy+2H_2O+O_2$$

This would give us a supply of nascent oxygen which would no doubt aid greatly in the dissolving of gold. He also claims that it is an oxygen carrier and not an oxidizer. I am of the opinion that no definite reactions can be given for its action, as they may pass through several different ways and yet arrive at the same final result, viz., the formation of $KAuCy_2$ and KBr.

Bromo-cyanide has no decomposing effect on potassium cyanide, and we can say that it simply acts as a storehouse of cyanogen and contains an element to combine with the potassium as long as there is gold to be dissolved.

Sulman and Teed claim that it requires less than one-half cubic inch of a solution of 0.1 per cent. KCy and 0.025 per cent. BrCy to dissolve 1 gr. of gold. Or a solution weighing $\frac{1}{3}$ the weight of the ore would extract as a maximum 80 oz. of gold per ton, while with oxygen only (i.e., by Elsner's Eqn.), 18 cubic inches of KCy solution of same strength are required to dissolve 1 gr. of gold, or as a maximum 2 oz. 2 dw. 9 grs. per ton of ore.

A weak solution of cyanide is more active than a strong one. The maximum solubility occurs with a solution of 0.25 per cent. This has been accounted for by Mr. McLaurin of New Zealand, by supposing that the rate of dissolution of gold is partly dependent upon the number of cyanide molecules in unit volumes, and partly on the number of oxygen molecules in the same unit volume and that the solubility of oxygen in cyanide solution decreases with the concentration. T. K. Rose, B.Sc., states that the rate of solubility of nearly pure gold is a function of time and temperature, while the concentration is of less importance.

Regarding the temperature, McArthur and Forest found that, "up to a temperature of about 100°, 110° F (the tropics) the solution from ores is unaffected, but beyond that the consumption of cyanide is seriously increased and the selective action retarded till at the boiling point it ceases and is to a certain extent reversed, for precipitation of the gold then takes place."

After a great number of experiments conducted with great care, Mr. Warwick came to the following conclusions regarding the rate of solubility of gold, some of which I have verified myself by experiment this year:

1. Temperature plays an important part in the cyanide process.

2. Presence of metallic cyanides has a marked influence upon the rate of solubility.

3. Gold-silver alloys are less soluble than pure gold.

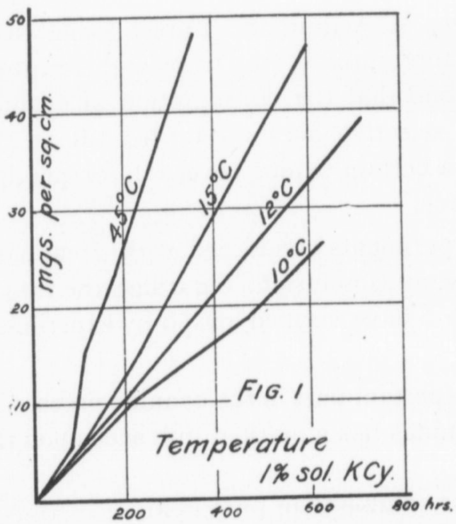
4. Rate of solution is uniform and dissolving is proportional to time.

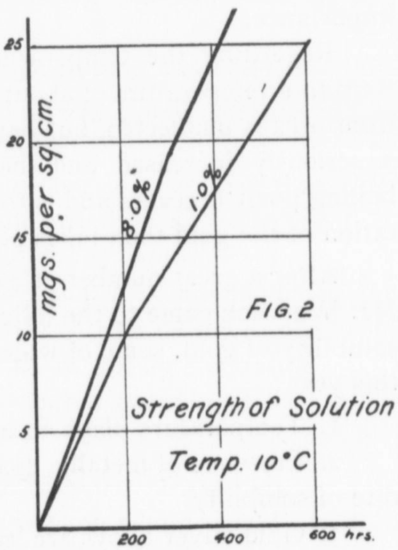
5. Halogen cyanides hasten the action of potassium cyanide.

To make the above conclusions more plain I give several curves representing them. These diagrams were constructed by Mr. Warwick from his experiments. There is one point about them to which I wish to draw your attention. They do not represent exactly what takes place in an ore for the reason that his plates do not expose the same amount of surface in proportion to weight as would be exposed in an ore. However they give us in less space what actually takes place than I could by writing.

Before going further into the actions which take place in the direct solution or precipitation of the gold, I intend to give an account of the reactions which take place with other substances which cause a direct loss of cyanogen, and hence, affect the economical side of the process.

Upon examination of the best authorities on chemistry, it is found that zinc, iron, nickel and copper are dissolved by potassium cyanide with the evolution of hydrogen, while cadmium and silver are soluble in presence of oxygen. Tin, mercury and platinum are not soluble at all. But as these metals do not occur native in ores it is necessary to look to the solubility and action of their compounds which actually do occur and which are injurious.





The most common mineral associated with gold is, perhaps, iron pyrites, and according to William Skey, analyst for New Zealand Government, it is almost, if not entirely, unaffected by cyanide solutions of any strength. However, pyrites is easily weathered, and the products thus formed have a most injurious effect. With air and water pyrites is changed as:

$$FeS_2 + H_2O + 7O = FeSO_4 + H_2SO_4.$$

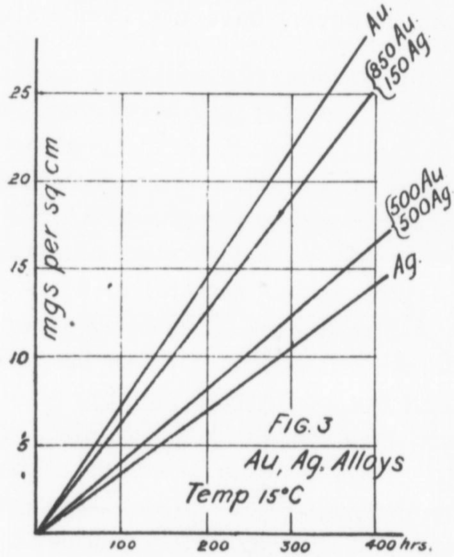
It is found that the ferrous sulphate thus formed reacts with cyanide as:

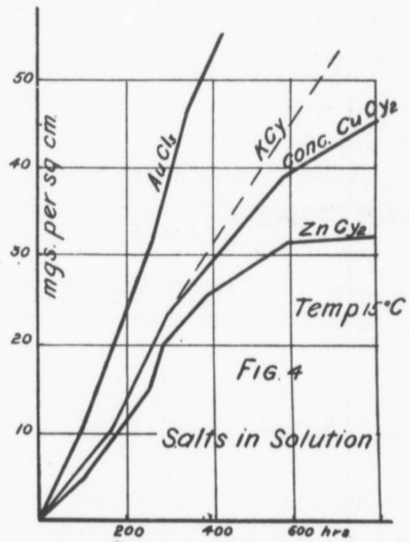
$$FeSO_4 + 2KCy = FeCy_2 + K_2SO_4.$$

The sulphuric acid also reacts similarly with the liberation of hydrocyanic acid. The ferrous cyanide here formed is attacked by the excess of cyanide and ferrocyanide of potassium is formed as

$$FeCy_2+4KCy=K_4FeCy_3.$$

This causes a great loss of cyanide, as will be seen. One molecule of $FeSO_4$ decomposes or destroys in practice 51 lbs. ($25) of cyanide per ton for each one per cent. of ferrous sulphate in the ore.





The ferrocyanide may be further acted upon if sufficient acid be present to form prussian blue, which has now become a by-product in the process as

$$12K+3FeCy_6+6FeSO_4+3O=Fe_2O_3+6K_2SO_4+Fe_7Cy_{18}.$$

This reaction is indicated by a blue color on the sides and surface of the tailings in the vat, also it is seen in the electrolytic process on the iron plates.

Ferric salts when unmixed with ferrous decompose the cyanide with the formation of prussic acid and ferric hydrate as

$$Fe_2(SO_4)_3+4KCy=FeCy_6+3K_2SO_4.$$

With further decomposition we get

$$FeCy_6+6H_2O=Fe_2(OH)_6+6HCy.$$

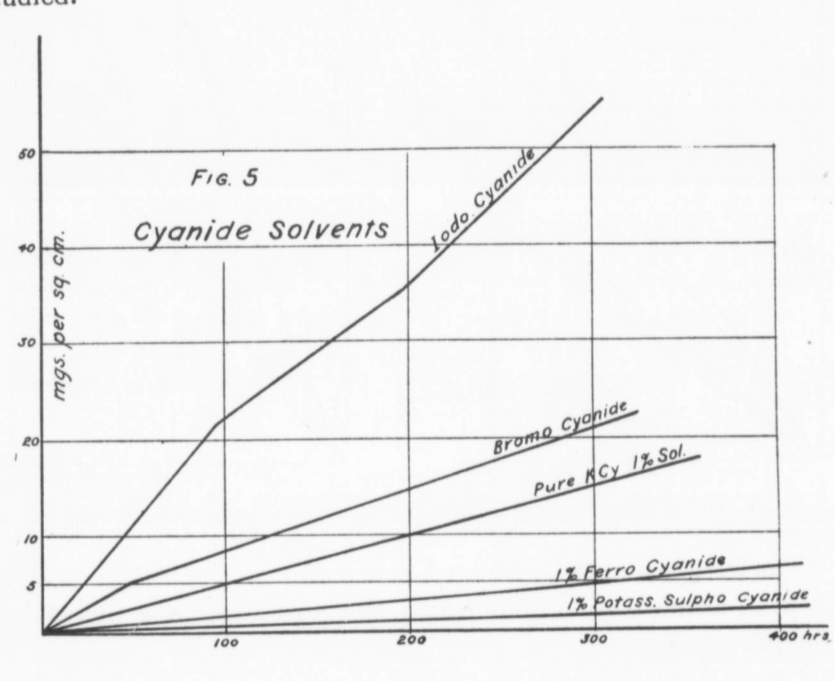
But as these salts are not at all likely to occur separately we may have very complicated equations arising. If the ferrous sulphate is in excess we get

$$12KCy + 3FeSO_4 + Fe_2(SO_4)_3 = Fe_3(FeCy_6)_2 + 6K_2SO_4$$

i.e., the ferrous-ferro-cyanide is produced. When the ferric sulphate is in excess we get ferric-ferri-cyanide as

$$18KCy + 3FeSO_4 + 2Fe_2(SO_4)_2 = Fe_4(FeCy_6)_2 + 9K_2SO_4.$$

In the case of an earthy pyritic ore we are likely to get sulphates of magnesia, lime or alumina which react similarly to the above, usually forming prussic acid. Their actions, however, have not been fully studied.



Another common mineral associated with gold is copper. The sulphides, oxides and carbonate all have an injurious effect on the solution. The sulphide is soluble in weak solutions and sulphur and oxidized compounds of sulphur are formed. It is with copper that the so-called "selective action" of cyanide takes place. It is noticed that when a strong solution (say 1.25 per cent.) of cyanide when treating an ore with as low as 0.25 per cent. copper gives a very poor extraction of gold, while if a very weak solution (say .25 per cent.) a fair extraction may be obtained. This may also be shown to take place where no copper is present, but not quite so marked, which is due, no doubt, to the fact that the weak solution is nearer the point of maximum solubility.

As copper sulphide is easily reduced to the sulphate and carbonate on weathering, and especially so where the ore is dried in kilns previously to dry crushing, we get a series of reactions similar to those resulting from the sulphide of iron given above. Malachite and azurite are both readily soluble in the solution producing copper, potassium-cyanide and prussic acid.

Antimony ores are exceedingly injurious to the working of this process, but have not been met with in Canada. Antimonite in very small quantities will cause the process to become useless owing to its cyanide destroying nature. As an example, I might refer you to the treatment of the Boatman Creek tailings, near Beefton, which gave a very low rate of extraction, while the consumption of cyanide was high.

According to many good authorities on chemistry, mercury in the metallic state is not soluble in cyanide, yet, in practice, we find it carried in solution and precipitated in the zinc boxes. In fact, at many works at the present where tailings are treated a special furnace is used so as to obtain the mercury again. A condenser attached to an ordinary furnace for reduction of the bullion will do. In all probability it is dissolved by the cyanide when in a finely divided state, while associated with the gold in amalgam. Mr. Mason, of Halifax, states that after a series of experiments he found that the amalgam on the surface of gold protects it to an enormous extent from the solvent action of potassium cyanide.

The action of arsenical pyrites does not seem to have any serious effect upon the cyanide solution. It has been shown in working the Deloro ores that only three-fourths of pound is consumed per ton, which is quite within economical limits. Galena has a slight action but not at all serious.

Hydrocyanic acid is one of the weakest acids known and is expelled from its salts by all mineral and many organic acids. Carbonic acid of the air decomposes the solution as

$$2KCy + CO_2 + H_2O = 2HCy + K_2CO_3.$$

This accounts for the smell of prussic acid in the vats. Now the HCy in presence of air is converted into a cyanate, then to a carbonate as

$$KCy + O = KCyO.$$

$$2KCyO + 3H_2O = K_2CO_3 + CO_2 + NH_3.$$

This also accounts for the ammoniacal odor common around the vats and zinc boxes. Many other very complicated formulæ might be given to show the effect of passing even the neutral gas nitrogen through the solution. The effect of heating is also interesting, as it gives ammonia, formic acid, acetates, etc., etc. This shows clearly the great necessity of having the solution kept from the action of the air by means of closely fitting lids.

Charcoal has a very injurious effect upon cyanide solutions. This may often get into the leaching tanks where drying the ore previous to any crushing has taken place. From experiments made by myself I have noticed a loss of 10 per cent. in passing a 1 per cent. solution once through a carbon filter. I have also extracted the gold from the solution (up to 90 per cent.) by filtering several times through the same substance.

SODA SOLUTIONS

Where sulphide ores are decomposed we find the greatest loss of cyanogen. This is due to the mineral salts and acids produced by weathering (described above). If the solution containing gold be passed over this kind of ore we may have the gold precipitated in the leaching vat as the insoluble cyanide AuCy as

$$2KAuCy_2 + H_2SO_4 = K_2SO_4 + 2HCy + 2AuCy.$$

There is little danger however of this taking place, as there is always an excess of the KCy present. To prevent the loss in this way, and to the cyanogen, the acids may be neutralized and the salts which are soluble washed away. The usual solution added is a weak wash of caustic soda after a wash of clean water and followed by another water wash. Any insoluble basic salts are changed into hydrates of the metals and soluble sulphate as

$$Fe_2O_32SO_3 + 4NaOH + H_2O = Fe_2(OH)_6 + 2Na_2SO_4$$
$$2Fe_2O_3SO_3 + 4NaOH + 4H_2O = 2Fe_2(OH)_6 + 2Na_2SO_4$$

The alkalies are removed by the last wash. There is an objection to the use of the soda solutions as it attacks the zinc, so instead a solution of caustic lime is used which has no injurious action. The direct action of alkali on cyanide is hydraulysis and thus affects it considerably in warm solutions where it aids in forming complex organic compounds. I might also mention here the effect of the alkalies on sulphides, especially the potassium hydrate, which is formed in the solution according to Elsner's equation. The action consists in causing the sulphides to become soluble as sulphides of the alkali metals, which in turn has its deadly effect upon the cyanide. These sulphides are easily tested and their presence should be carefully watched.

PRECIPITATION OF THE GOLD FROM SOLUTION

The voltaic order of metals in different solutions of cyanide of potassium is given in the following table.

0.625 per cent. KCy.		12.5 per cent.	30 per cent. KCy.	
50°F	100°F		50°F	100°F
+ Al	+ Mg	+ Mg	+ Mg	+ Mg
Mg	Zn	Cu	Zn	Al
Zn	Cd	Cd	Cu	Zn
Cu	Al	Au	Al	Cu
Cd	Co	Ag	Cd	Cd
Sn	Cu	Ni	Au	Sn
Co	Ni	Sb	Ag	Au
Ni	Su	Hg	Ni	Ag
Ag	Au	Pd	Sn	Ni
Au	Ag	Bi	Hg	Hg
Hg	Pb	Fe	Pb	Pb
Pb	Hg	Pt	Co	Co
Te	Sb		Sb	Sb
Pt	Bi		Te	Te
Sb	Fe		Bi	Bi
Bi	Te		Fe	Fe
Fe	Pt		Pt	Pt
− C	− C	− C	− C	− C

The relative solubility, under the given conditions, of the different metals is given by their position in the above table. It will be noticed that mercury is inapplicable as a precipitant for gold, and that aluminum acts more rapidly than zinc in cold dilute solutions. The Moldenhaner process is based upon the latter fact. The aluminum is used in the presence of a free alkali and the gold separates quickly as

$$6AuKCy_2 + 6KOH + 2Al + 3H_2O = 6Au + 6KCy + 6HCy + 6KOH + Al_2O_3 = 6Au + 12KCy + 6H_2O + Al_2O_3,$$

which shows that it uses one quarter the weight of aluminum as is used in the zinc process. This method has not been used much owing to the expense of aluminum.

Zinc is the usual metal used for precipitating the gold. Pure zinc has a very slow action, but when it is combined with the precious metal the action becomes vigorous, due to a galvanic couple being formed, and which probably develops enough electro-motive force to decompose water, forming hydrogen, and zinc hydrate. The action of the zinc on the potassium-auro-cyanide is represented as

$$2AuKCy_2 + Zn = ZnK_2Cy_4 + 2Au,$$

which, if it were the only action which took place, would represent 1 oz. of zinc precipitating about 6 oz. of gold; but as it requries in practice 16 oz. of zinc to precipitate 1 oz. of gold, the excessive consumption must arise from the decomposition by free cyanide, as

$$4KCy + Zn + 2H_2O = ZnK_2Cy_4 + 2KOH + H_2.$$

And again,

$$2AuKCy_2 + H_2 = 2KCy + 2HCy + Au_2.$$

This has been stated by some authorities to be the way in which the deposition takes place. The hydrocyanic acid thus liberated combines with any free alkali present, and hence no loss of cyanogen, which was combined with the gold, takes place.

Hydrogen is always given off in the zinc boxes and carries off hydrocyanic acid with it. This hydrogen in the nascent state may also form with the prussic acid methylamine as

$$HCy + 2H_2 = CH_3\,NH_2.$$

This also accounts for the ammoniacal smell noticed in the neighborhood of zinc boxes.

The double cyanide of zinc and potassium is not a solvent for gold or silver; so long as there is an excess of zinc there is no dissolving of the gold by the free cyanide.

In treating ores on which a caustic soda solution has been used a white deposit is formed on the zinc which has been puzzling many metallurgists as to what it is. Messrs. Butters and Clennel suggest (in *Engineering and Mining Journal*, October 29th, 1892) that it is cyanide of zinc. $ZnCy_2$ which is formed by the sodic zincate ($Zn(ONa)_2$) on the cyanide of zinc and potassium as

$$Zn + 2NaOH = Zn(ONa)_2 + H_2,$$

$$Zn(ONa)_2 + K_2ZnCy_4 + 2H_2O = 2ZnCy_2 + 2NaOH + 2KOH.$$

This is doubtful as the zincic-cyanide formed would not be precipitated, but remain in solution in the excess of potassium cyanide.

The zinc used is generally in the shape of turnings or shavings, but of late a very economical form is that of "zinc fume." It is a product found in the manufacture of zinc, consisting merely of the condensed fumes of the metal. It has been used with perfect success at the Deloro mine, where an inverted cone is used as the precipitating box. As the solution of cyanide containing bromide of cyanogen will carry more gold than a plain solution, it is not necessary to run the solution through these cones till very rich in gold.

LABORATORY PRACTICE OF THE CYANIDE PROCESS

As the cyanide process is essentially a chemical one, it requires continual testing and assaying of the solutions and ores. It is necessary to have a man who is capable of making assays and analyses of the different products which are being handled.

The determination of free or available potassium cyanide is extremely easy and yet requires care and skill in treating the solutions by the proper method, for the conditions vary considerably in different localities.

There are three volumetric methods used for the determinations.

1. Standard solution of silver nitrate.
2. " " " mercuric chloride.
3. " " " iodine.

The first is, perhaps, more widely used than the others, owing to the fact that the second, although extremely delicate in pure solutions, is not so reliable where impurities are present. The third is also a good method in clear solutions, but in presence of sulphides or muddy and discolored solutions is not reliable.

The silver nitrate method is based upon the fact that when a solution of silver nitrate is added to a solution of potassium cyanide the cyanide of silver is formed, which is insoluble in water, but soluble in a solution of potassium cyanide. So as soon as all the cyanide is used up a white precipitate occurs. The equation is represented as

$$AgNO_3 + KCy = AgCy + KNO_3$$
$$AgCy + KCy = AgKCy_2$$
$$\text{Or } AgNO_3 \text{ saturates } 2KCy.$$
$$\text{Or } 170 = 130.$$

A decinormal solution of $AgNO_3$ may be used (i.e. 17 grs. of $AgNO_3$ in 1l). It is now easy to titrate a solution of cyanide and calculate the percentages therefrom. In practice if the decinormal solution is used, to save calculation 13cc of the cyanide solution is taken and titrated. The reading on the burette divided by 10 thus gives the percentage of the solution. I find it convenient in practice to make up a standard solution of $AgNO_3$, so that when we take 25cc of the cyanide solution and titrate our reading divided by ten, will give the percentage (I use 32.69 grs. of $AgNO_3$ per litre).

In titrating it is necessary to add the $AgNO_3$ slowly, and shake or stir well till a slight permanent pearly opalescence appears. If the solution is already slightly turbid two samples are placed side by side, so as to be able to see the slightest permanent turbidity.

It has been stated by Louis Janin, jr., that the process is not reliable with solutions containing zinc and other metals (which might also be said of the mercuric chloride method). Although the method is not reliable for the exact percentage of cyanogen present, it is sufficiently accurate for the amount of available cyanogen, which is the only thing which we are required to know in making up the strengths of the solutions.

To obtain the total cyanogen present the method used by Watts and Fresenius is as follows. The solution is boiled with an excess of oxide of mercury, when all the cyanogen is obtained as cyanide of mercury, and the other metals pass into oxides. The cyanide of mercury is then precipitated by nitrate of silver and weighed.

The mercuric chloride method depends upon a similar reaction o that of silver nitrate as

$$HgCl_2 + 2KCy = HgCy_2 + 2KCl$$

271 saturates 130.

Thus 27.1 grs. per litre is a decinormal solution. This may be used so as to read in percentages as before by using 25ce of the cyanide solution when we put 52.125 grs. of $HgCl_2$ per litre. The titration is similar in all respects to the former one except that the solution must be slightly alkaline. This is done by adding about 5cc of 3 per cent. ammonia. The permanent pearly opalescence marks the end of the reaction.

When iodine is used it is dissolved in potassium iodine solution. The reaction is as

$$2I + KCy = KI + ICy,$$

or 254 saturates 65.

The solution may be made up to such strength as the above by a very simple calculation. The iodine loses color until all the cyanide is used up, when the solution becomes yellow, or if a starch paste is used, a blue color is given. This method is extremely delicate in the end reaction, but as the iodine solution does not keep well, it becomes unreliable after some time and has to be titrated against standard cyanide or hyposulphite from time to time.

When titrating by the above methods it is often of advantage to remove the sulphides and zinc as their presence causes the methods to become inaccurate. The usual way to get rid of these is to add sufficient caustic soda so that the solution is distinctly alkaline, which prevents any loss by way of hydrocyanic acid. Next hydrogen sulphide or pure concentrated sodium sulphide is added and well shaken, which precipitates the zinc silver or mercury present as sulphides. The excess of sulphide after filtering can be removed by agitation with litharge (or any insoluble lead compound) added in small quantities at a time till a drop of the solution no longer gives the slightest coloration to a lead acetate solution. After filtering it may be titrated by any of the above methods.

It is often found that a certain amount of matter is suspended in the solution which cannot be removed by filtering. This causes considerable inconvenience in titrating. A very good method which has been recommended, and which I have used to advantage, is to add a small quantity of lime, shake well and filter. The solution comes through quite clear while the lime has no chemical effect on the available cyanide.

Where bromide of cyanogen is used its strength is determined by the following method. Add an excess of potassium iodide and acidify with hydrochloric acid. This will liberate iodine, which may be titrated by means of a standard hyposulphite solution. The method depends upon the following reaction :

$$BrCy + 2KI = KBr + HCy + I_2.$$

Potassium cyanide when purchased is never pure. It is put up in cans and is of two classes, one manufactured in Germany being about 98 per cent., while that made in Scotland is only 70.80 per cent. The impurities are usually black carbide of iron and carbonates, together with other insoluble matters. The strength may be tested by dissolving a known weight (taken as a fair sample) in water and titrating by any of the above methods. The percentage is easily calculated.

In assaying a gold solution there are several methods used, according to what is required. If the silver, as well as the gold, is wanted, the solution is evaporated down to nearly dryness, when litharge is added. The sides of the vessel should be carefully rubbed down as the evaporation takes place. When the litharge and residue is dry, it is transferred to an ordinary crucible, together with flux mode of glass-soda, argal, and a cover of borax. The flux used for ordinary siliceous gold ores, together with some powdered window glass, is all that is necessary. The remaining part is conducted as an ordinary assay.

Another method where gold only is required is to take a known quantity of solution (say $\frac{1}{2}$ pt. or 250cc.) and add silver nitrate solution till a precipitate ceases to form. This carries down all the gold as the aurous cyanide (AuCy) while the silver is as AgCy. The solution is filtered and the filter scorfied with best lead and cupelled as usual. The silver may be used over and over again, as none is lost, except mechanically in the cupel. Some hydrochloric acid added to the filtrate precipitates any free $AgNO_3$, which is also saved.

A method which is quite common where both silver and gold are required is as follows: A known quantity of solution is taken and sufficient sodium sulphide or iron sulphide is added to precipitate the silver and the base metals which are filtered and the filter assayed. The gold is then precipitated by zinc chloride and filtered, mixed with test lead, melted and cupelled.

It is often found necessary to test the acidity of the ore so as to ascertain the amount of alkali to be added for neutralization. This may be done by washing a known quantity of the ore with a standard solution of soda, which is afterwards titrated. The difference in strengths gives the amount of soda used up by the ore. From this the quantity for the vat of ore may be easily determined.

As the solutions have to be continually made up to a standard strength, it is necessary to have some simple and accurate method of making the calculation. The following formula is of great assistance. It is derived from the principles of allegation:

Let A = Desired strength of Stock solution in per cents.
" B = Present " " " " " "
" C = Strength of the dissolving tank in per cents.
" D = Quantities (in tons, lbs., litres, etc.)

Then $\frac{A = B}{C = A} \times D =$ Quantity of dissolving tank solution to be added (in tons, lbs., litres, etc.).

By the use of tell-tales on the tanks this becomes simple.

As mentioned before it is often necessary to test for alkaline sulphides. I find the following methods of great service:

1. Add a little acid to a clear solution of cyanide. If the sulphides are present a cloudiness is noticed due to liberated sulphur.

2. A bright silver coin will become tarnished.

3. Add a small quantity of nitro-prussides to the solution, and a brilliant deep blue color is seen, which is a very delicate test.

A very useful thing to have at hand when using cyanide solutions is a solution of cobaltic nitrate, which is a perfect antidote in case of poisoning. This was discovered by Dr. Johann Antel, of Hungary.

Before closing I wish to make a few remarks about the reduction of the gold precipitates to bullion. These slimes are very rich in zinc, and contain other base metals more or less according to the ore which is being treated. If mercury is present the ore should be well dried, and then the mercury retorted at a heat almost sufficient to drive off the zinc. This may then be treated in several ways. One of the more common ways is to dissolve the zinc in sulphuric acid in a wooden vat. The product is then washed and filtered in a filter press. Afterwards this is smelted in graphite crucibles, which gives a very fine bullion.

A method which has been long in use and which has resulted in a great loss in gold was to oxidize the zinc in a current of air in a muffle. The gold is carried off mechanically enclosed in the oxide. Mr. Hugh Rickard at the Deloro mine, Ontario, has invented a new process. He mixes the slimes intimately with a soluble hydrocarbon and borax into balls about the size of an egg. These are then placed in a retort with a wide mouth and connected with a dust chamber. The zinc is volatilized and does not carry any gold with it. The gold remains in the porous carbon and is easily reduced to a fine bullion. The proportions for the flux is 5 pounds of molasses, 1 pound of borax to 20 pounds of precipitate, which often contains as high as 30 per cent. gold.

In conclusion, I might say that I believe the cyanide process, and especially the bromo-cyanide method, to be one which shall be of great importance in Canada, owing to the cheapness with which many low grade ores may be treated. It is now far beyond being in an experimental stage, for it has proven itself to be a most powerful extractor of the precious metals.

DISPOSAL OF SEWAGE.

By P. H. Bryce, M.A., M.D., Secretary Provincial Board of Health of Ontario.

It is with pleasure that I again address you in response to the kind invitation of your President, to speak on sewage disposal works. It is not necessary, or indeed, possible for me to lead you through all the history of the attempts made to dispose of sewage, and therefore I shall confine myself to the practical methods which are yearly being more advanced as the principles underlying them are better understood, namely, disposal by artificially prepared filter beds, or by prepared beds upon land areas of sufficient extent, or by what is known as the broad irrigation plan in sewage farming.

The particular variety of sewage filtration to be adopted in any particular case, must, as in other matters, depend upon local conditions, the amount of sewage to be dealt with, local soil conditions, cost of land, pumping and other important mechanical conditions.

Varying these as we may, the one essential consideration is to aid in every way the natural processes by which organic matter is reduced to its simpler elements.

These are essentially those of fermentation, putrefaction and nitrification. Under the first we reduce especially non-nitrogenous matters; by the two latter, nitrogen-bearing compounds. Inasmuch, however, as putrefaction depends more essentially upon the more complex action of micro-organisms, depending rather upon the oxygen compounds of the organic tissues themselves, than upon the free oxygen of the air, upon which ærobic destruction of organic matters depends; and since the latter process, now called nitrification, as in the processes of plant growth in soils which are under cultivation, depends upon the free oxygen of the soil and air, it is of practical interest to you who are shortly to become Sanitary Engineers, to have outlined some of the principal methods of sewage disposal which have been found practical, and which are to-day in operation at different places.

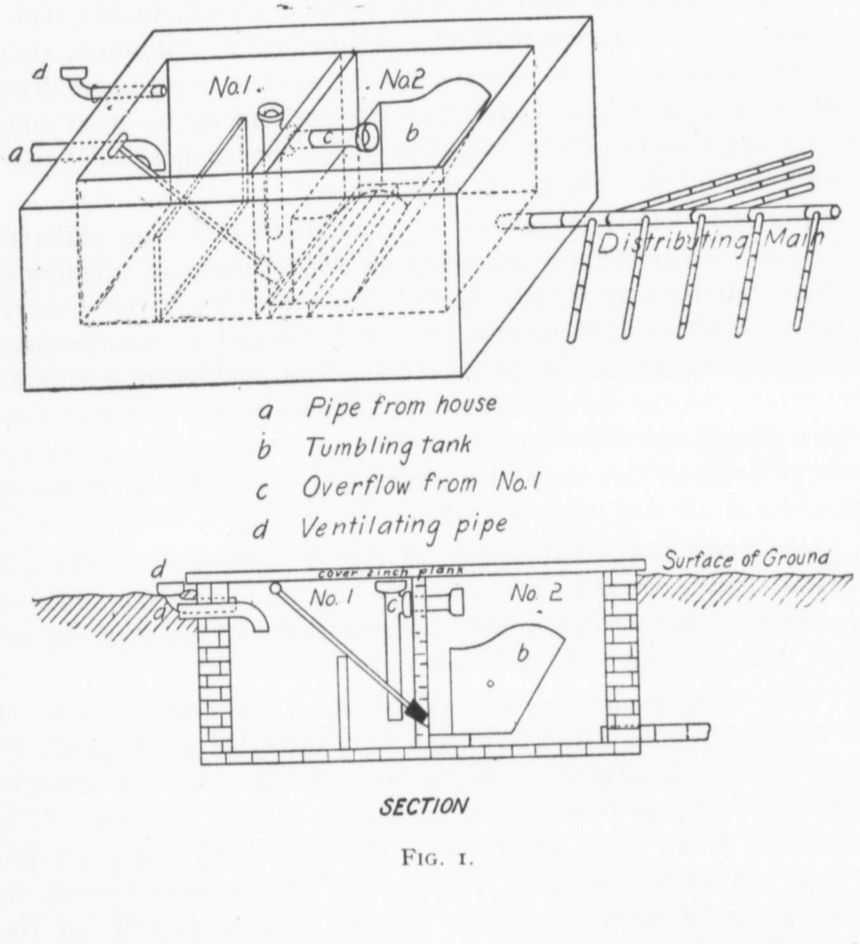
As you are aware, sewage varies notably in its composition according to its amount depending upon whether the *separate* or *continued* system of sewerage exists. With the first, we deal practically only with sewage of an amount and composition definitely known; while in the combined, the amount of storm water becomes an important additional factor to be dealt with. Nowadays when storm water is allowed to go into the sewers, it is common to arrange storm-water overflows at the mouths of the local and main sewers, which will, when the sewage has reached a certain amount of dilution, allow an overflow to run into some water-course, where it may be considered a practically harmless pollution. By such methods, it at once becomes possible to arrange filtration plants for handling a definitely known amount of sewage.

Further, however, since in manufacturing towns different industries are carried on, each polluting with its wastes, as tannery refuse, pulp-factory refuse, woolen factory refuse, gas liquors, chemical works, etc., it is apparent that if it hoped to have natural processes of organic decomposition carried on, and where it is found that certain wastes contain germicidal chemicals, it is apparent that such must either not be allowed entrance into the general sewerage system, or that they must be neutralized (as acids, for instance) before being allowed to be carried therein.

We therefore see that no rule of thumb method will suffice, if efficient sewage filtration is to be carried on, and hence the town or city engineer has need of a very notable range of scientific knowledge, if he is going to do definitely good work.

With these preliminary remarks, we may now refer to some of the working details. In every case it is desirable by strainers to remove as much suspended matter as possible from the sewage, especially if sewage receives any organic or mineral matters. It is sometimes found quite practical, as in the sewage from private houses, institutions, or small towns, with the Separate system, to allow paper or other suspended matters to pass directly on the receiving tank, or even the filters. As, however, a receiving tank of considerable capacity allows of a very notable sedimentation, and since it is always very easily ventilated, it is a notable economy to have it in any system, arranged, if possible, at the end of the main sewer, thereby preventing the filter beds from being clogged by solid matters.

For the purposes of a private house, large hotel or public building, as for instance a county poor-house, or asylum, such a method as set forth in Fig. 1 will amply supply all the needs by a system of sub-surface field tiles, which if placed under a garden, not only will dispose of the sewage without cost or nuisance, but will add materially by irrigation to its productiveness.



SECTION

FIG. 1.

Another method and one which has merit, is that where the sewage is turned on to beds of graded quicksand, coke, polarite, or granular hemætite, as carbonate of iron, or of burnt clay, such as red brick dust (coarse). Some of the results of such beds as affording a rapid means of nitrification of organic matter are now well known.

In the last number of the Journal of the Sanitary Institute, England, a description is given of the method whereby 54,000 gallons of sewage is daily disposed of. The quantity dealt with was 371 times the contents of the tank; without removal the amount of sewage solids left on the beds was only 66½ cubic yards, of which but four per cent. was organic matter. Had this been dealt with as sewage before nitrification, there would have been 556 cubic yards to dispose of. In this case the filters were formed of cinders and coke *breeze*, of size which would pass through a half inch mesh of the sieve.

Again at the Lawrence Experimental Station, Massachusetts, a filter of twenty-six ft. of sand was laid over ground with the usual under drains, and treated with tannery sewage 120,000 gallons per acre daily. This became soon clogged, owing to the amount of sludge accumulated on the filter. The matter was gelatinous and wet, but cracked when dry; when, however, the same sewage was treated on a coke filter two feet in depth, with 100,000 gallons daily, and flooded for two hours daily, much of the organic matter was removed, together with all the arsenic of the chemicals used, which had proved poisonous to the micro-organisms in the sand filter. This then was allowed to flow into the sand filter, and gave continuously a satisfactory effluent.

With the sewage from the pulp mills, it was similarly found that the chemicals prevented the nitrification organisms from breaking up the sewage. With, however, the use of the coke strainer, it was found that 100,000 gallons per acre daily could be filtered satisfactorily. Three hours in the sedimentary tank before application to the filter was further found to remove 30% with the addition of alum.

In dealing with the waste or wool scourings from woolen factories, it has been found in the Massachusetts experiments, by tanking the waste and adding calcium chloride, that a lime soap is thereby formed and much matter is precipitated, and the yellowish liquid drawn off holds but little matter in suspension.

To account for the action of filters with different kinds of sewage, it must be remembered that the microbic life present in water, soil and air, consists of a number of different micro-organisms. Thus as many as 35 different organisms were obtained from manufacturing sewages, while those mingled with city sewage had even more than others.

From what has been said in illustrating the methods of sewage filtration, it is apparent that while the methods must be varied to suit circumstances, yet the principles underlying efficient filtration, viz,—

that of the aeration of the materials of the filter, to the end of promoting nitrification, is the same in all cases. Taking advantage of this fact, the principle has been still further applied to sand or coke filtration, in which the natural provisions are supplemented by forced aeration. The following particulars are given from a pamphlet published by Col. George E. Waring, C.E., on "The Purification of Sewage by Forced Aeration."

Described briefly, the mode of operations is as follows:—

1. The sewage after passing through suitable screens, which withhold large solids, as rags, paper, etc., flows slowly, horizontally over a shallow bed (say six inches deep) of coarse broken stone,

A—False bottom.
B—Pipe from blower.
C—Broken stone.
D—Diaphragm.
E—Coarse gravel.

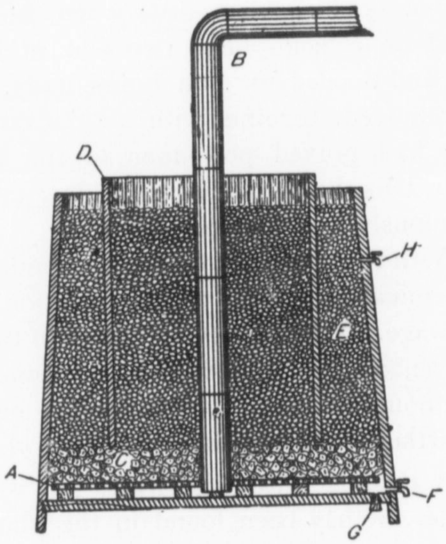


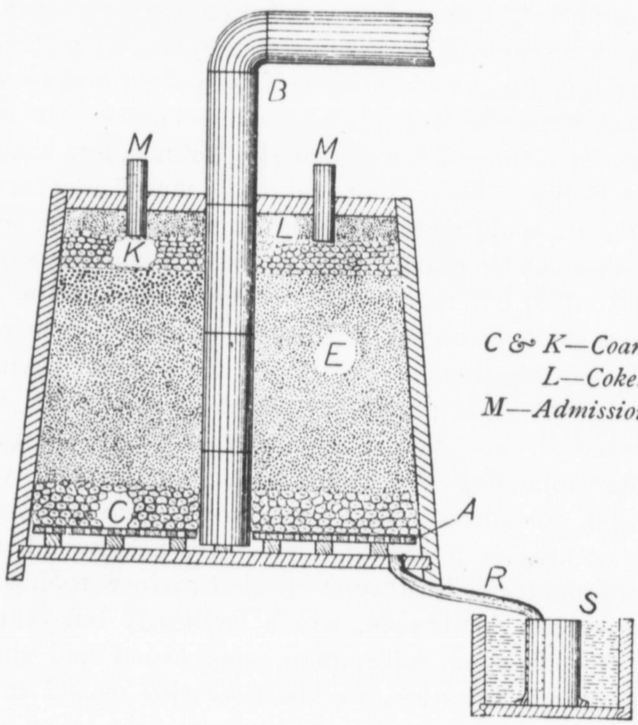
FIG. 2.

which serves to catch and retain the coarse floating particles. Three sets of such stones should be supplied, each with a capacity to receive the flow of a certain period, thereby allowing twice the time for recuperation that they are in use. When not in use, these filters drained, expose the filter to oxidation, which speedily results in decomposition and cleansing of the filter.

2. The sewage leaving these stones freed from coarser matters, passes to a straining tank filled with pure broken stone, coarse gravel, cinders or coke. This tank (Fig. 2) has two compartments divided

by a diaphragm extending nearly to the bottom. The sewage passes down one and up through the other, overflowing at its top. The rate of flow must be slow enough for the sediment to deposit on the surface of the filter. From this tank properly managed, the sewage goes as a clear opalescent fluid with a perceptible odor.

3. When such tank has been operated for a considerable time, its pores tend to become clogged at the surface especially, and filtration is slower. Then the sewage is turned to another tank similar; the



C & K—Coarse gravel.
L—Coke.
M—Admission pipe.

FIG. 3.

emptied tank now is treated by air driven into the false bottom of the tank by a blower. Under these conditions bacterial oxidation is rapidly set up and the filter becomes clean. It is best to provide four of these tanks, each resting three times as long as in use. At Newport such filters run five months without renewal of materials, and were practically as clean and effective as at first.

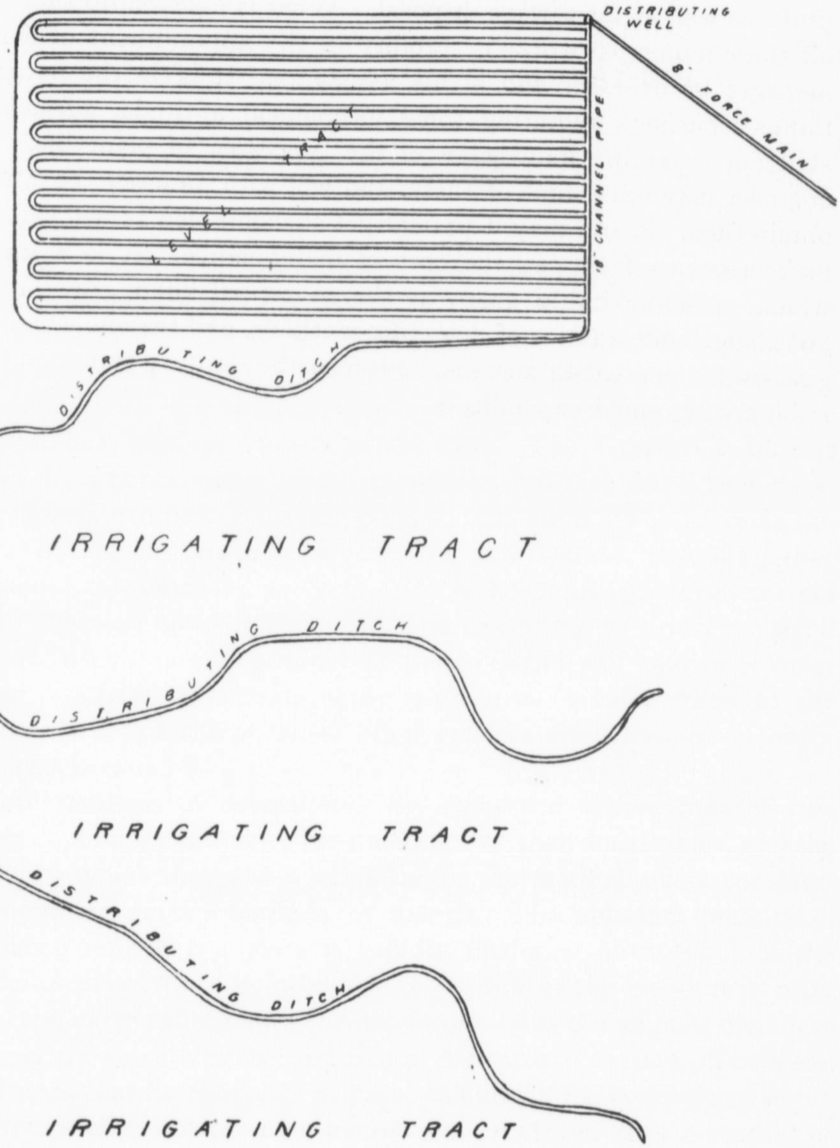
4. If such further purification is desired, the sewage is allowed to

filter in a thin film over the coke or other filtering material in a tank without diaphragm, as in Fig. 3, which is constantly aerated under pressure by a blower, clear water flowing out of a trapped pipe at the bottom.

One such tank is enough to cleanse the average flow of sewage, and is used continuously, nitrification completing the decomposition of the sewage.

While, however, the methods which I have indicated, fully indicate the principles of filtration, we must remember that no artificial methods can excel, if indeed they can equal nature's method of the destruction of organic matter and the purification of polluted waters. In England, France and Germany, we have illustrations of this at the sewage farms of large cities, as Birmingham, Coventry, etc., in England; at Gennevilliers, near Paris, France, and the great sewage farms of Berlin. There, with naturally sandy and pervious soils or sandy loams, made more pervious by sub-soil drainage, the raw sewage is either led by gravity or pumped to the carriers of the sewage beds, and either by broad irrigation on meadows, or by intervening broad and shallow ditches or furrows, paralleling the raised ridges, intermittent irrigation, or so-called intermittent downward filtration, is practised. The productiveness of these sewage farms has been phenomenal, and they have proved to be good investments, even after including the handling of the sewage, which ought not to be considered when comparing such with other methods where nothing is produced. Nowhere, however, has the method had better practical illustration in this Province than at the London Institution for the Insane, where for nearly ten years the flat bed system of cultivation, with intervening broad and shallow ditches for intermittent filtration, has been carried on. Products, approaching $500 per acre annually in value, have been taken off the beds, consisting of small fruits and vegetables of the most splendid quality, while the whole field has been developed into the most perfect gardens and parterres that may be seen in any vegetable garden in the country. All this is accomplished by the aid of one foreman; all the rest of the work being performed by the inmates of the institution.

The following diagram represents the sewage farm, including the level trail with its flat beds and intervening ditches; also the ditch broad irrigation on the inland surface. The latter has never been required.



PLAN FOR SEWAGE DISTRIBUTION.

Several of our towns have already introduced or are about to dispose of their sewage by sewage farms, although as yet they appear to be slow to realize the economic value of working the farm for profit as well as for sewage disposal. What is essential to success in all these municipal works is stability in the appointment of a farm manager or overseer and of his assistants. It is in the nature of things a business requiring some mechanical aptitude, agricultural skill and scientific knowledge of the processes of plant life. The engineer may have some of these, and will with practice be able to obtain them all, and may if patient get such foremen as will certainly make a sewage farm pay the cost of maintenance. It is a method which must inevitably grow, as the municipal mind grows in the knowledge and practice of scientific methods, and the one which will guarantee the greatest success, both from the sanitary standpoint and economy in annual expenditure.

LONGITUDE BY LUNAR DISTANCES.

BY L. B. STEWART, O.L.S., D.T.S., LECTURER IN SURVEYING.

The method of determining longitudes by lunar distances, although laborious and unsatisfactory owing to its lack of precision, is indispensable to the navigator and explorer. The ordinary method in use at sea is to compare the local time at a place whose longitude is required with that of Greenwich or some place whose geographical position has been previously established. The Greenwich time is shown by a chronometer whose correction and rate have been carefully determined before leaving port, so that from its indications the exact Greenwich time may be found at any instant, assuming that its rate has continued uniform. On a long voyage chronometers usually become unreliable, so that it is necessary to resort to some method which is independent of the uniformity of rate of a time-keeper. These statements apply with even greater force to the explorer, whose mode of travel often renders the transport of chronometers impossible.

All methods of determining the difference of longitude of two places depend upon finding the difference of their local times, and the means by which this end is achieved in the method under consideration may be briefly outlined as follows: The apparent position of the moon among the stars is rapidly changing on account of the motion of the earth in its orbit and the motion of the moon in its orbit about the earth; the Nautical Almanac contains the angular distances between the moon's centre and that of the sun and certain planets and fixed stars that lie nearly in its path, calculated for every three hours of Greenwich time; the observer at sea measures with a sextant or other reflecting instrument the angular distance between the moon's bright limb and one of those other heavenly bodies, and after correcting his measurement in a manner to be described below, he searches in the Nautical Almanac for the two values of the distance between which his value, found by observation, lies. The Greenwich time corresponding to his observed distance is then found by interpolation, and the difference between this and his local time, found at the time of observation, gives his longitude.

Stated thus, the method appears extremely simple, but the correction mentioned above involves a considerable amount of calculation. The angular distances given in the almanac have their vertices at the centre of the earth, while the observed angle is measured at a point of its surface; the correction therefore consists in reducing the observed angle to the centre of the earth. To effect this, corrections must be applied for refraction and parallax, and also for semi-diameter, the distance being measured from the limb of the moon to

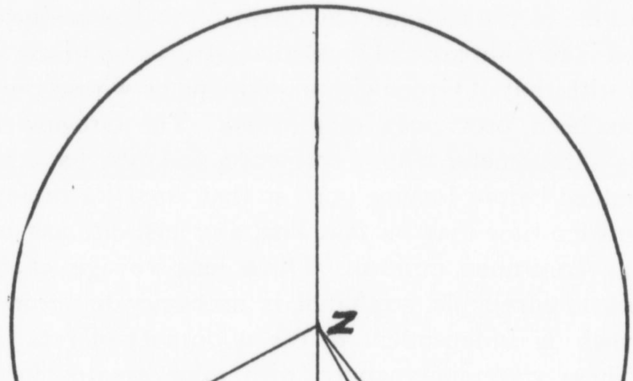


FIG. 1.

a star or the limb of the sun, if that be the other body observed. The semi-diameters tabulated in the Nautical Almanac must also receive corrections on account of the apparent contraction of their discs produced by refraction, which is quite apparent to the eye at small altitudes, and of measurable amount for altitudes up to 40° or 50°. The semi-diameter of the moon also varies with its altitude above the horizon, on account of its proximity to the earth. As the corrections for refraction and parallax depend upon the altitudes, a knowledge of those quantities at the time of observation is necessary, and if two additional observers are available the altitudes may be measured at the same instant that the distance is measured. If there be but one observer he may proceed as follows: Measure first the altitude of the moon, then the distance several times in quick succession, and finally the altitude of the moon again, the times of all the measurements being noted. The mean of all the measured distances is then taken, and also that of the corresponding times, and the altitude at this time is then found by interpolation between the two measured values. The altitude of the sun may be calculated; and it should also be observed as soon as convenient after the completion of the other observations, in order to find the chronometer correction on local time, if that be not known already.

In order to show the manner of applying the corrections enumerated above it is necessary to have recourse to spherical trigonometry, and that the investigation may be as general as possible we shall assume that the other body is the sun; if a star be employed the only alteration in the reduction is the omission of certain corrections.

Figure 1 is a projection of the celestial sphere on the plane of the horizon, Z is the zenith; ZA the meridian, S' and M' the apparent positions of the centres of the sun and moon respectively, S and M the true or geocentric positions of those points. In the case of the moon parallax exceeds refraction, so that the position of M is above M'; parallax also causes a slight displacement in azimuth; but in the case of the sun refraction exceeds parallax and the parallax in azimuth may be neglected.

Let—

ζ'_1 = apparent zenith distance of the moon's centre $= ZM'$.

$\zeta_2{}'$ = apparent zenith distance of the sun's centre $= ZS'$.

d' = apparent angular distance between their centres $= M'S'$.

The first operation is to find the angle Z or $S'ZM'$, which may be found by the equations:

$$\tan^2 \tfrac{1}{2} Z = \frac{\sin (S - \zeta'_1) \sin (S - \zeta'_2)}{\sin S \sin (S - d')} \qquad (1)$$

$$S = \tfrac{1}{2} (\zeta'_1 + \zeta'_2 + d')$$

In preparing the data for these equations the augmented semidiameter of the moon must be computed; this may be found by the following approximate expression (see Chauvenet, vol. 1, p. 184):

$$S' = S + [5{\cdot}2495]\ S^2 \cos \zeta'', \qquad (2)$$

In which S is the horizontal and S' the augmented semi-diameter, ζ'', the observed zenith distance corrected for refraction and semi-diameter, and the quantity in the brackets the logarithm of a constant term. The semi-diameters must also be corrected for the contraction produced by refraction, that of the vertical semi-diameter being first found from a refraction table by taking from such a table the amount of the refraction for the limb and for the centre, and taking their difference. The contraction of an inclined semi-diameter, required in correcting the observed distance, may be found as follows :

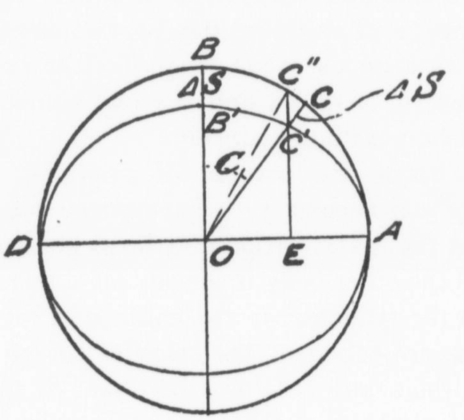


FIG. 2

Let $ACBD$ (Fig. 2) be a portion of the moon's disc, $AC'B'D$ the same contracted by refraction, BB' is the contraction of the vertical semi-diameter and CC' that of the inclined, making an angle C with the vertical. We have then, regarding $AC'B'D$ as an ellipse

$$\frac{C'E}{C''E}=\frac{b}{a} \quad \text{or} \quad C'E=C''E\ \frac{b}{a}$$

a and b being the semi-axes of the ellipse ; therefore

$$C''E-C'E=C''E\left(1-\frac{b}{a}\right)$$

or, if ΔS be the vertical contraction and $\Delta' S$ the contraction of the inclined semi-diameter, then we have very nearly

$$\frac{\Delta' S}{\cos C}=S\cos C\left(1-\frac{OB'}{OB}\right)$$

$$=S\cos C\ \frac{\Delta S}{S}$$

$$\therefore \qquad \Delta' S=\Delta S.\cos^2 C \qquad (3)$$

This angle C is the angle at S' or M', Fig. 1, and may be found with sufficient accuracy from the triangle $ZM'S'$ using the horizontal semi-diameters in correcting the data.

Having found Z the next operation is to find the geocentric distance d, assuming for the present that M lies on ZM', by means of the equations

$$\tan \theta = \tan \zeta_1 \cos Z$$
$$\cos d = \frac{\cos \zeta_1 \cos (\zeta_2 - \theta)}{\cos \theta} \qquad (4)$$

in which ζ_1 and ζ_2 are the geocentric zenith distances of the centres of the moon and sun respectively, found by correcting the observed quantities for refraction, parallax in zenith distance, and semi-diameter. The horizonal semi-diameters and parallaxes are given in the Nautical Almanac for given instants of Greenwich time, which necessitates a knowledge of the Greenwich time at the instant of observation, which requires an approximate knowledge of the observer's longitude. The parallax in zenith distance of the sun may be found by the equation

$$\Delta \zeta_2 = \pi_2 \sin \zeta''_2 \qquad (5)$$

and that of the moon by the equations

$$\Delta\zeta_1 = \frac{\rho}{a} \pi_1 \sin (\zeta''_1 - \gamma) \qquad (6)$$
$$\gamma = (\varphi - \varphi') \cos A'$$

(see Chauvenet, Vol. I., p. 112), in which

π_1 and π_2 are the horizontal parallaxes of the moon and sun, respectively.
ρ the radius of the earth for the given latitude.
a the the radius of the equator.
φ the geographical latitude.
φ' the geocentric latitude.
ζ_1'' the same as in equation (2).
A' the azimuth of the moon reckoned from the south.

The quantities $\frac{\rho}{a}$ and $\varphi - \varphi'$ may be taken by interpolation from the table given below. The azimuth A' may be computed from the data δ (the moon's declination) φ and ζ_1 using in finding ζ_1 an approximate value of $\Delta\zeta_1$ given by the equation

$$\Delta\zeta_1 = \pi_1 \sin \zeta_1''$$

After having found d as above described there still remains a small correction for parallax in azimuth, for which the following expression is taken with slight alteration from Chauvenet

$$\Delta A = \frac{\rho}{a} \cdot \frac{\pi\ (\varphi - \varphi') \sin 1'' \sin A'}{\sin \zeta_1''} \qquad (7)$$

The true position of the moon is nearer the meridian than its apparent position. This correction affects directly the angle Z, and the influence upon d of a small change in Z may be found by means of a differential formula. Taking the equation

$$\cos d = \cos \zeta_1 \cos \zeta_2 + \sin \zeta_1 \sin \zeta_2 \cos Z$$

we have by differentiation

$$-\sin d\ \delta d = -\sin \zeta_1 \sin \zeta_2 \sin Z\ \delta Z$$

or

$$\delta d = \frac{\sin \zeta_1 \sin \zeta_2 \sin Z}{\sin d}\ \delta Z$$

and substituting in this the value of δZ given by (9), and assuming that $\zeta''_1 = \zeta_1$ we have

$$\delta d = \frac{\rho}{a} \cdot \frac{\pi(\varphi - \varphi') \sin 1'' \sin \zeta_2 \sin Z \sin A'}{\sin d} \qquad (8)$$

If preferred, the angle Z as first determined by equations (1) may be corrected for parallax in azimuth found from (7) before using it in equations (4), which will then give the true distance, requiring no further correction.

Having thus computed the true distance, or having "cleared the lunar distance," as it is technically termed, the corresponding Greenwich time must be found by interpolating between the instants of time corresponding to the two nearest distances given in the almanac. This operation is facilitated by the use of proportional logarithms, which may be thus explained: Let d_1 and d_2 be the two distances given in the almanac between which d falls, T_1 and T_2 the corresponding Greenwich times, and T the required time; then $T_2 - T_1 = 3^h$, and if we denote $T - T_1$ by t, $d_2 - d_1$ by Δ and $d - d_1$ by Δ', then by simple interpolation

$$t = \frac{\Delta'}{\Delta} \times 3^h$$

or in seconds

$$t = \frac{\Delta'}{\Delta} \times 10800$$

$$= \Delta' \times \frac{10800}{\Delta}$$

or

$$\log. t = \log. \Delta' + P.\ L.\ \Delta \qquad (9)$$

in which $P.\,L.\,\Delta = \log.\,10800 - \log.\,\Delta$

Δ' and Δ being in seconds. Therefore finally the required Greenwich time =

$$T = T_1 + t \qquad (10)$$

The Nautical Almanac gives $P.L.\,\Delta$ and towards the end of that work is given a table by which a correction for second difference may be applied.

The method just given for computing the true distance gives a rigorous solution of the problem, and in practice, if only an approximate result is aimed at, some of the smaller corrections may be neglected. For instance, the correction for parallax in azimuth in the example given below amounts to less than 4″, though in some cases it may greatly exceed that amount. The example will show the relative importance of the various corrections. Works on navigation give methods and tables for reducing a lunar distance, many of them giving only an approximation to the truth, which, however, should be used where time saving is of the utmost importance; Chauvenet remarks, after working out an example by his approximative method, that the same example worked out by Bowditch's first method gives a value of the true distance, differing from his value by 59″, which would result in an error of 32′.5, or over half a degree in the longitude. The method used in the reduction of an observation should be such that the error of computation lies well within the error of observation, especially if a large number of observations are to be reduced and the mean of the resulting values of the longitude taken, otherwise a large constant error may be introduced which will make the mean of but little more value than a single result.

Tables have been calculated from which may be taken the augmentation of the moon's semi-diameter, the contraction caused by refraction, etc., by using which the calculation of those quantities may be avoided.

EXAMPLE

At a place in latitude 35° N. and assumed longtitude 150° W., 1856, Mar. 9, at the local mean time $T = 5^h\ 14^m\ 6^s$ the following observations were made :

Alt. of moon's lower limb	= 52° 34′
Alt. of sun's lower limb..............	= 8° 56′ 23″
Dist. between nearest limbs	= 44° 36′ 58.″6
Height of bar.	= 29.5 ins.
Attached therm.	= 60° F.
External therm.	= 58° F.

The approximate Greenwich time is $15^h\ 14^m\ 6^s$ with which we take from the American ephemeris:

Semi-diam. of moon	=	16′ 23.1″
Semi-diam. of sun	=	16′ 8″
Hor. parallax of moon	=	60′ 1.9″
Hor. parallax of sun............ ..	=	8.6″
Decl. of moon	=	+14° 19′
Decl. of sun	=	— 4° 3′

To find the augmentation of the moon's S.D. The complete expression is:

$$S'-S = h\,S^2\cos\zeta' + \tfrac{1}{2}\,h^2\,S^3 + \tfrac{1}{2}\,h^2 S^3\cos^2\zeta'$$

$$\log h = \bar{5}\cdot2495$$

$$S = 16'\ 23\cdot1'' = 983\cdot''1$$

log h	=	$\bar{5}\cdot2495$
" S	=	$2\cdot9926$
" S^2	=	$5\cdot9852$
" cos 37° 9′	=	$\bar{9}\cdot9015$
" 1st term	=	$1\cdot1362$
" 0.5	=	$\bar{1}\cdot6990$
" h^2	=	$\overline{10}\cdot4990$
" S^3	=	$8\ 9778$
" 2nd term	=	$\bar{1}\cdot1758$
" cos² 37° 9′	=	$\bar{9}\cdot8030$
" 3rd term	=	$\bar{2}.9788$

therefore

1st term	=	13·″68
2nd "	=	·15
3rd "	=	·10
		13·″93
S	=	16′ 23· 1
S'	=	16′ 37″

To find the contractions of *S. D.'s* of sun and moon:

App. alt. sun's limb	= 8° 56′ 23″	r =	5′ 43·″ 10
" " " centre	= 9° 12′ 31″	r =	5′ 33· 94
contraction		=	9·″ 4
App. alt. moon's limb	= 52° 34′	r =	43·″14
" " " centre	= 52° 50′ 37″	r =	42· 75
contraction		=	0· 39

In order to find the contractions of the inclined semi-diameters it is necessary to find the angles at M' and S'. Denoting them by these letters, respectively we have :

$$\tan^2 \tfrac{1}{2} M' = \frac{\sin (s-\zeta_1') \sin (s-d')}{\sin s \sin (s-\zeta_2')}$$

$$\tan_2 \tfrac{1}{2} S' = \frac{\sin (s-\zeta_2') \sin (s-d')}{\sin s \sin (s-\zeta_1')}$$

in which

$$\zeta_1' = 37^\circ\ 9' \qquad \zeta_2' = 80^\circ\ 48' \qquad d' = 45^\circ\ 10'$$

whence

$$\begin{aligned} s &= 81^\circ\ 33.'5 \\ s-\zeta_1' &= 44^\circ\ 24.\ 5 \\ s-\zeta_2' &= 0^\circ\ 45.'5 \\ s-d' &= 36^\circ\ 23.'5 \end{aligned}$$

and

log sin 44° $24.'5 = 9.84495$	
" " 36 $23.\ 5 = 9.77327$	
" " 81 $33.\ 5 = 9.99527$	
" " 0 $44.\ 5 = 8.12172$	log cos $159^\circ 52' = 9.97262$
19.61822	" $\cos^2$ 159° $52' = 9.94524$
18.11699	" 0.4 $= 1.60206$

" $\tan^2 \frac{1}{2} M' = 11.50123$

" $\tan \frac{1}{2} M' = 10.75061$

$\frac{1}{2} M' = 79^\circ\ 55.'8$

$M' = 159^\circ\ 51'7$

log sin 0° $45.'5 = 8.12172$	
" " 36 $23.\ 5 = 9.77327$	
" " 81 $33.\ 5 = 9.99527$	
" " 44 $24.\ 5 = 9.84495$	log cos 12° $10' = 9.99013$
17.89499	" $\cos^2 12^\circ$ $10' = 9.98026$
19.84022	" 9.4 $= .97313$
" $\tan^2 \frac{1}{2} S' = 8.05477$	" 8.98 $= .95339$

" $\tan \frac{1}{2} S' = 9.02738$

$\frac{1}{2} S' = 6^\circ\ 4.'8$

$S' = 12^\circ\ 9.'6$

$\therefore$ the contraction of the moon's incl. $S.D. = 0.''4$

" " " sun's " " $= 9.''0$

To find the angle Z

Correcting the observed altitudes by the quantities found above we have:

$$\zeta_1' = 37^\circ\ 9'\ 23''.4$$
$$\zeta_2' = 80^\circ\ 47'\ 38''.4$$
$$d' = 45^\circ\ 9'\ 34''.2$$

whence

$$s = 81^\circ\ 33'\ 18''$$

and

$$\begin{array}{lll}
s-\zeta_1' = 44^\circ\ 23'\ 54''.6 & \text{log. sin} = & 9{\cdot}8448775 \\
s-\zeta_2' = \quad 45'\ 39''.6 & \quad " \quad " = & 8{\cdot}1232492 \\
s-d' = 36^\circ\ 23'\ 43''.8 & \quad " \quad " = & 9{\cdot}7733151 \\
s = 81^\circ\ 33'\ 18'' & \quad " \quad " = & 9{\cdot}9952653 \\
 & & 17{\cdot}9681267 \\
 & & 19{\cdot}7685804 \\
\text{log. } \tan^2 \tfrac{1}{2}Z & = & 8{\cdot}1995463 \\
\quad " \quad \tan \tfrac{1}{2}Z & = & 9{\cdot}0997731 \\
\tfrac{1}{2}Z = 7^\circ 10'\ 18'' & & \\
Z = 14^\circ 20' 36'' & &
\end{array}$$

We require next the parallaxes of the sun and moon. To find that of the moon an approximate knowledge of its azimuth is necessary. The moon is evidently to the west of the meridian. Denoting the zenith, the pole, and the moon's centre by the letters A, B and C respectively, and the opposite sides of the triangle formed by those points by a, b and c respectively, then to find the angle A we have

$$\tan^2 \tfrac{1}{2}A = \frac{\sin (s-b) \sin (s-c)}{\sin s \sin (s-a)}$$

in which

$$b = \zeta_1 = 36^\circ 34' \text{ approx.}$$
$$c = 90^\circ - \varphi = 55^\circ$$
$$a = 90^\circ - \delta = 75^\circ\ 41'$$

In finding b an approximate value of the parallax was found by the equation

$$p = \pi_1 \sin \zeta_1''$$

thus

$$\begin{array}{lll}
\log\ 3602 & = 3{\cdot}55654 & \\
\ "\ \ \sin 37^\circ 10' & = 9{\cdot}78113 & \\
\ "\ \ p & = 3{\cdot}33767 & p = 2176'' = 36'\ 16''
\end{array}$$

We have then from above data

$$\begin{array}{ll}
s-b = 47^\circ\ 3'.5 & \log \sin = 9{\cdot}86454 \\
s-c = 28^\circ\ 37'.5 & \quad " \quad " = 9.68041
\end{array}$$

$$s-a = 7^\circ\ 56'.5 \qquad \log \sin = 9.14039$$
$$s = 83^\circ\ 37.5 \qquad \text{“} \quad \text{“} \quad = \underline{9.99731}$$
$$19.54495$$
$$\underline{19.13770}$$
$$\log.\ \tan^2 \tfrac{1}{2} A = 10.40725$$
$$\text{“} \quad \tan \tfrac{1}{2} A = 10.20362$$
$$\tfrac{1}{2} A = 57^\circ\ 58'$$
$$A = 115^\circ\ 56'$$

$\therefore$ the azimuth reckoned from the south$=64^\circ 4'$

Then to compute the parallax by (6):

$$\log(\phi-\phi') = 2.81786 \qquad \log \frac{\rho}{a} = 1.99952$$
$$\text{“} \quad \cos 64^\circ\ 4' = \underline{9.64080} \qquad \text{“} \quad \pi_1 = 3.55653$$
$$\text{“} \quad \gamma = 2.45866 \qquad \text{“} \quad \sin 37^\circ 5' = \underline{9.78030}$$
$$\gamma = 287.''5 \qquad \text{“ par. in } z.d. = 3.33635$$
$$= 4'\ 47.''5$$
$$\zeta_1'' = 37^\circ\ 10'\ 6'' \qquad \therefore \text{ par.} = 2169.''5$$
$$\zeta_1'' - \zeta = 37^\circ\ 5'\ 18.''5 \qquad = 36'\ 9.''5$$

The parallax of the sun is found by (5). Thus:

$$\log 8.6 = .93450$$
$$\text{“} \quad \sin 80^\circ\ 53' = \underline{7.99448}$$
$$\text{“} \quad \text{par.} = .92898$$
$$\text{“} \quad \text{par.} = 8.''49$$

To find d the true distance.

The observed altitudes are now corrected as follows:

		Of Moon.		Of Sun.
Obs'd alt.	=	$52^\circ\ 34'$	=	$8^\circ\ 56'\ 23''$
Refr.	=	$0'\ 43.''1$		5 43·1
		52 33 16·9		8 50 39·9
Par.	=	36 9·5		8·5
		53 9 26·4		8 50 48·4
S. D.	=	16 37		16 8
		53 26 3·4		9 6 56·4
	ζ_1 =	36 33 56·6	ζ_2 =	80 53 3·6

and we found previously $Z=14^\circ\ 20'\ 36''$

Then, using equations (4) we have:

$$\log\ \tan 36^\circ\ 33'\ 56.''6 = 9.8702503$$
$$\text{“} \quad \cos 14^\circ\ 20'\ 36'' = \underline{9.9862469}$$
$$\text{“} \quad \tan \theta = 9.8564972$$

$$\theta = 35^\circ\ 42'\ 5.''9$$
$$\zeta_2 = 80^\circ\ 53'\ 3.''6$$
$$\zeta - \theta = 45^\circ 10' 57.7$$

$$\log \cos 36^\circ\ 33'\ 56''6 = 9.9048096$$
$$\text{" \quad " } 45^\circ\ 10'\ 57.''7 = 9.8480958$$
$$\text{" \quad " } 35^\circ\ 42'\ 5.''9 = 9.9095918$$
$$19.7529054$$
$$\text{" \quad " } d = 9.8433136$$
$$d = 45^\circ\ 48'\ 10.''2$$

This value must receive a further correction for parallax in azimuth, computed by (10)

Thus:

$$\log \frac{\rho}{a} = \bar{1}.99952$$
$$\text{" } \pi_1 = 3.55653$$
$$\text{" } \varphi - \varphi' = 2.81786$$
$$\text{" } \sin 1'' = \bar{6}.68557$$
$$\text{" } \sin \zeta_2 = 9.99048$$
$$\text{" } \sin Z = 0.39398$$
$$\text{" } \sin A' = 9.95391$$
$$\text{" } \sin d = 9.85549$$
$$.39785$$
$$\text{" } \delta d = .54326$$
$$\delta d = 3.''486$$

Value of distance just found

$$= 45^\circ\ 48'\ 10.''2$$
$$\therefore\ d = \underline{\underline{45^\circ\ 48'\ 13.''7}}$$

To find the longitude corresponding to this distance we find in the Nautical Almanac that on March 9th

$$\text{at } 15^h\ d = 45^\circ\ 40'\ 54'' \quad P.L. = .2510$$
$$\text{" } 18^h\ d = 47\ \ 21\ \ 53$$

We must then interpolate between these two values. We find

$$\Delta' = 7'\ 19.''7 = 439.''7$$

thus

$$\log \Delta' = 2.6432$$
$$P.L. = .2510$$
$$\log t = 2.8942$$
$$t = 783.^s8$$
$$= 13^m\ 4^s$$

∴ Greenwich time	=	$15^h\ 13^m\ 4^s$
Corr. for 2nd diff.		-1
		15 13 3
Local time of observation	=	5 14 6
Resulting long.	=	$9^h\ 58^m 57^s$

In the above computation all the corrections have been allowed for, though some of them were of very small amount. Thus, the correction for the contraction of the moon's semi-diameter might have been omitted without much sacrifice of accuracy.

Table giving the values of log. $(\varphi - \varphi)$ and $\log \frac{\rho}{a}$ for each degree of the quadrant, calculated by the formulæ.

$$\varphi - \varphi' = 700\cdot''44 \sin 2\varphi - 1\cdot''19 \sin 4\varphi$$

$$\frac{\rho}{a} = 1 - \tfrac{1}{2}\, e^2 \sin^2 \varphi$$

$$\log e^2 = \bar{3}\cdot 83050257$$

φ	$\log(\varphi - \varphi')$	$\log. \frac{\rho}{a}$	φ	$\log(\varphi - \varphi')$	$\log. \frac{\rho}{a}$
1°	1·38686	0·	22°	2·68608	$\bar{1}$·99979
2°	1·68744	0·	23°	2·70127	$\bar{1}$·99977
3°	1·86314	0·	24°	2·71546	$\bar{1}$·99976
4°	1 98744	$\bar{1}$.99999	25°	2·72868	$\bar{1}$·99974
5°	2·08357	$\bar{1}$·99999	26°	2·74099	$\bar{1}$·99972
6°	2·16182	$\bar{1}$·99998	27°	2·75246	$\bar{1}$ 99970
7°	2·22760	$\bar{1}$·99998	28°	2·76312	$\bar{1}$·99968
8°	2·28427	$\bar{1}$.99997	29°	2·77301	$\bar{1}$·99965
9°	2·33395	$\bar{1}$·99996	30°	2·78216	$\bar{1}$·99963
10°	2·37803	$\bar{1}$·99996	31°	2·79061	$\bar{1}$·99961
11°	2.41757	$\bar{1}$ 99995	32°	2·79838	$\bar{1}$·99959
12°	2·45333	$\bar{1}$.99994	33°	2·80550	$\bar{1}$·99956
13°	2·48588	$\bar{1}$ 99993	34°	2·81198	$\bar{1}$·99954
14°	2·51568	$\bar{1}$ 99991	35°	2·81786	$\bar{1}$.99952
15°	2·54306	$\bar{1}$·99990	36°	2·82312	$\bar{1}$·99949
16°	2·56833	$\bar{1}$·99989	37°	2·82781	$\bar{1}$·99947
17°	2·59171	$\bar{1}$·99987	38°	2·83191	$\bar{1}$·99944
18°	2·61340	$\bar{1}$·99986	39°	2·83547	$\bar{1}$·99942
19°	2·63355	$\bar{1}$·99984	40°	2·83846	$\bar{1}$·99939
20°	2·65320	$\bar{1}$·99983	41°	2·84091	$\bar{1}$·99937
21°	2·66979	$\bar{1}$·99981	42°	2·84283	$\bar{1}$·99934

φ	$\log(\varphi - \varphi')$	$\log. \frac{\rho}{a}$	φ	$\log(\varphi - \varphi')$	$\log. \frac{\rho}{a}$
43°	2.84420	$\bar{1}$·99932	67°	2·70333	$\bar{1}$·99875
44°	2·84505	$\bar{1}$·99929	68°	2·68821	$\bar{1}$·99873
45°	2·84537	$\bar{1}$·99926	69°	2·67198	$\bar{1}$·99872
46°	2·84515	$\bar{1}$·99924	70°	2·65456	$\bar{1}$·99870
47°	2·84441	$\bar{1}$·99921	71°	2·63587	$\bar{1}$ 99868
48°	2·84314	$\bar{1}$·99919	72°	2·61578	$\bar{1}$ 99867
49°	2·84133	$\bar{1}$·99916	73°	2·59415	$\bar{1}$·99865
50°	2·83898	$\bar{1}$·99914	74°	2·57083	$\bar{1}$·99864
51°	2·83608	$\bar{1}$·99911	75°	2·54562	$\bar{1}$·99863
52°	2·83263	$\bar{1}$·99909	76°	2·51829	$\bar{1}$·99861
53°	2·82862	$\bar{1}$·99906	77°	2·48854	$\bar{1}$·99860
54°	2·82403	$\bar{1}$·99903	78°	2·45602	$\bar{1}$·99859
55°	2·81886	$\bar{1}$·99901	79°	2·42032	$\bar{1}$ 99858
56°	2·81309	$\bar{1}$ 99899	80°	2·38079	$\bar{1}$·99857
57°	2·80670	$\bar{1}$·99896	81°	2·33676	$\bar{1}$·99856
58°	2·79968	$\bar{1}$·99894	82°	2·28711	$\bar{1}$·99856
59°	2·79200	$\bar{1}$·99892	83°	2·23047	$\bar{1}$·99855
60°	2·78364	$\bar{1}$·99889	84°	2·16468	$\bar{1}$·99854
61°	2·77458	$\bar{1}$·99887	85°	2 08650	$\bar{1}$·99854
62°	2·76477	$\bar{1}$·99885	86°	1·99038	$\bar{1}$·99853
63°	2·75420	$\bar{1}$·99883	87°	1·86611	$\bar{1}$·99853
64°	2·74280	$\bar{1}$·99881	88°	1·69046	$\bar{1}$·99853
65°	2·73057	$\bar{1}$·99879	89°	1 38970	$\bar{1}$·99853
66°	2·71743	$\bar{1}$·99877	90°		$\bar{1}$·99853

THE ACCURACY OF THE ASSAY OF PYRITIC ORES.

By J. W. Bain, B.A.Sc.

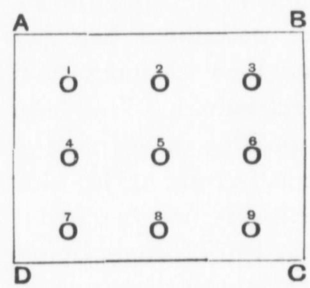
Some time ago my attention was attracted by a paragraph in "Beringer's Text-book of Assaying," which is partly reproduced here for the benefit of those to whom the work is not accessible.

"In assaying, when much pyrites is present, this oxide (Fe_2O_3) will be found in the preliminary calcination, or by the action of nitre, if this reagent be used. The separation of the gold, when much peroxide of iron is present in the slag, is by no means always complete. We have known a student, in attempting to recover gold from hematite (in which there was about half an ounce of the metal) to find nearly a gram of gold on cleaning the slag; although, in the fusion, the slag was perfectly fluid. This failure, no doubt, was due to the peroxides of iron present in the slag. To get a good result, the iron must be present as ferrous oxide, and the slag should not be too acid." Some experiments are described in support of these statements, and show clearly the unsatisfactory degree of extraction in certain cases. "Unless precautions are taken, it is easy to leave 5 or 6 milligrams of gold in the slag, and it is difficult to recover them completely. It is probably because of the loss of gold in the slag in this way, that some writers have asserted that a certain proportion of gold is volatilized during calcination. Many gold assayers are chary of calcining their samples." Such words from an able chemist could not pass without remark, and it seemed quite possible, in the light of the experiments quoted above, that some of the standard methods, in the assay of pyritic ores, might be vitiated by a slight and constant error. To determine whether such was the case or not, I made the investigation, which is now to be described.

Since the pyrites, evidently, was a source of trouble in the assay material, it was, in the most careful determination of the value of the ore under consideration, removed with the aid of nitric acid. The residue from this treatment ranked as a material carrying no sulphides, and was subjected to the crucible process, the inaccuracy of which has been carefully investigated. In this manner I arrived at a value for the ore, which approximated much more closely to the truth than any which could be attained by assay methods of doubtful value. The details of the experiments will now be described.

MATERIAL.—The most pyritic ore at hand, a quantity of concentrates containing 85% of iron pyrites was used; of this, 5 lbs. were taken, and crushed to pass an 80 mesh sieve. The ore was then thoroughly mixed and stirred for 20 minutes, in order that the samples might be as uniform as possible.

SAMPLING.—In selecting a sample for determination by the combined wet and dry method, the ore was spread out in a thin sheet, and small portions were taken at equal intervals and placed in a dish, care being taken to reach the bottom of the sheet as well as the top. The contents of the dish were thoroughly stirred, and the sampling operation repeated until approximately 100 grams was obtained. In sampling for assays, a certain procedure was observed. If A B C D



represent the sheet of ore, the numbers indicate the points at which samples were taken. For instance, at the point 1, a quantity weighing approximately 30, or in some cases 15 grams, was removed to a small dish, and from this, after a thorough mixing, the amount of ore necessary for the assay was selected. It will be seen that, in this manner, it was proposed to determine the conditions which prevail in the sampling of an ore, under circumstances similar to those described.

FURNACE OPERATIONS.—All roastings were made in 4-inch Battersea dishes of the well-known type; all fusions in Battersea round crucibles, size F. The fuel used was gas.

PARTING.—It was considered expedient to add the excess of silver necessary for parting, before cupellation, and accordingly, a quantity of that metal, bearing the proportion of $2\frac{1}{2}:1$ to the gold was added to the lead buttons, as they were being laid upon the cupels. When the oxidation of the lead was complete, the gold and silver beads were placed on a smooth anvil, and beaten out till quite thin; they were then transferred to porcelain capsules, and treated with dilute nitric acid, sp. gr. 1.17. After boiling for 15 minutes, the acid was decanted, a fresh quantity of sp. gr. 1.42 added, and boiling continued for 10 minutes. After the removal of the strong acid, the residue was washed three times with water, and the capsules were subjected to a gradually increasing heat, which gently drove off the moisture, and caused the gold to assume its normal color, on reaching a certain temperature. The precious metal now appeared as a thin yellow disc, possessing a considerable degree of coherency, and resembling, in miniature, the gold cornet of the bullion assay.

WEIGHING.—The only balance available was unsatisfactory in some respects. It had been constantly used by students for a number of years, and had, in consequence, lost some of its original sensibility; in any case, the instrument, by reason of its design, was ill-adapted to the measurement of very small weights. In order that some idea of the correctness of the weighings might be obtained, a small piece of platinum foil was weighed ten times, by the method of vibrations, with the following results. A 4 milligram rider was used.

Weight.	Diff. from mean.
1.65 mgrm.	—0.004 mgrm.
1.63 "	— 24 "
1.67 "	+ 16 "
1.67 "	+ 16 "
1.66 "	+ 06 "
1.65 "	— 04 "
1.67 "	+ 16 "
1.63 "	— 24 "
1.66 "	+ 06 "
1.65 "	—0.004 "
1.654	

The greatest difference from the mean, which in this case is the most probable value, is 0.024 mgrm., and we may assume that, if the other weighings were as carefully conducted, the error would not exceed ± 0.030 mgrm. The slight difference in load, from 1.8 mgrms. to 3 mgrms., would cause no appreciable difference in the sensibility.

The value of the 10 mgrm. weight, used in some of the determinations, was carefully ascertained in terms of the weight of the rider, and it was found that, if the rider be assumed=4 mgrms., the 10 mgrm. weight=10.02 mgrms.

REAGENTS.—Nitric acid: During some of the operations, advantage was taken of the fact that gold is insoluble in nitric acid, and precautions had to be taken to ensure that no free chlorine, an excellent solvent for gold, should be present. One part of the acid, marked C.P. sp. gr. 1.42 was, diluted with two parts of distilled water, and to the mixture a concentrated solution of silver nitrate was added in considerable excess. The whole was well shaken, and, when the slight precipitate had completely settled, the clear liquid was decanted, and placed in a clean bottle. This, and some strong acid prepared in the same manner, except for dilution, were used exclusively in the experiments.

Litharge: 100 grms. were fused with soda and argols, producing a button of 40 grms. weight, which was cupelled and parted in the usual manner. Only a trace of gold was found.

DETERMINATION OF THE VALUE BY THE COMBINED METHOD — From a sample selected in a manner previously described, 100 grms. were weighed out, and treated with a few c.c. of nitric acid, in a large covered beaker. The reaction which ensued was very violent, and not more than 5 to 10 c.c. of the acid could be added at a time, so that a day or more was required to effect solution. When the addition of acid produced further action, the whole was boiled for some time, and then diluted to 500c.c., 10c.c. of dilute sulphuric acid were added, then a concentrated solution of lead acetate, and the precipitate was allowed 12 hours to settle. The insoluble residue, the sulphur liberated from the pyrites, and the lead sulphate were collected on a filter, washed and dried. The filter was placed in a roasting dish, and gradually heated to ignition in a muffle, without being stirred, or agitated in any way. No decrepitation was observed, and it is hardly credible that any loss could have occurred in this operation. The roasted product was ground in a clean mortar, and fused with the following charge:

Litharge	100 grms.
Soda	20 "
Borax	20 "
Argols	6 "

The fusion yielded a fluid slag, and a lead button weighing 40 grms., which was reduced to 15 grms. by scorification. After cupellation, the bead was parted and weighed, as already described. The crucible and scorification slags, together with the cupel, were ground up, and fused with litharge and argols, to recover any gold which had been lost; as a result 0.02 mgrm. was found. The mean value from four determinations is 2.99 oz. per ton.

DETERMINATIONS BY DRY METHODS.—As has been stated previously, the object in making these experiments, was to determine the degree of accuracy which could be claimed for each of the common methods of assay. In selecting the processes, to which could be applied the adjective common, a list of six was open for choice. Descriptions of the three which were chosen, will be given later, while, in regard to the others, perhaps a few words may be necessary to justify their rejection. Accounts of the latter will be found in W. L. Brown's Manual of Assaying.

The first is a method in which enough nitre is added to oxidize all the sulphur present; and after fusion is complete, charcoal is added to reduce a sufficient quantity of lead. The objections in this case are, the use of a large quantity of nitre, which is always troublesome, and the necessity of adding charcoal before the completion of the operation, making an extra demand on the time of the assayer.

In the second process, the desulphurization is effected by means of metallic iron, in the form of nails, which is more or less unsatisfactory for very pyritic ores, while, in addition, Brown states that the method is not reliable.

The third, known as Aaron's process, consists in smelting to a matte, which is afterwards scorified. The additional operation is a decided disadvantage.

The methods which were adopted will now be described in detail.

ROASTING.—One assay ton of the ore was carefully roasted in a muffle, and fused with the following charge:

Litharge	30 grms.
Soda	48 "
Borax	32 "
Argols	4 "

The fusions were excellent, yielding lead buttons of 23 grams on an average.

NITRE METHOD.—In this method a lead button of sufficient size is reduced by the action of the sulphides, and nitre is employed to oxidize the excess of sulphur. There are several objections to the process, on the score of inconvenience in working. The large amount of nitre precludes the charge of an assay ton of ore, if F crucibles are to be used, and, indeed, by taking half quantities, all danger of boiling over is not avoided. In addition, the desulphurization does not always seem to be satisfactory, and in the experiments, several fusions had to be rejected for this reason. Charge:

Ore	½ A.T.
Litharge	22 grms.
Soda	30 "
Nitre	30 "
Borax glass	3 "

When properly executed, the slag is very fluid, and the lead averages 15 grams in weight.

Nitre-Iron Method.—With ores treated by this process, the desulphurization is effected partly by iron, and partly by nitre, sufficient sulphur being left to reduce a lead button of desirable weight. The great advantage lies in the fact that only a small quantity of nitre is required, which reduces, to a considerable extent, the danger of boiling over. Charge:

Ore	1 A.T.
Litharge	30 grms.
Soda	45 "
Borax	22½ "
Nitre	

Three 5-inch wire nails.

The results are good in the great majority of cases, although now and again there is failure. The slag is very fluid, with a lead button of 23 grms. The method can be heartily recommended for the expeditious assay of pyritic ores.

RESULTS.

	Roasting.		Nitre.		Nitre-Iron.	
	Oz. gold per ton.	Diff. from mean.	Oz. gold per ton.	Diff. from mean.	Oz. gold per ton.	Diff. from mean.
1	2.96	—0·015	3·04	+0·063	3·04	+0·08
2	2·92	— ·055	3·00	+ ·023	3·02	+ ·06
3	2·96	— ·015	2·88	— ·970	3·06	+ ·10
4	2·96	— ·015	3·00	+ ·023	2·94	— ·02
5	3·18	+ ·205	2·96	— ·017	3·06	+ ·10
6	2·68	— ·295	3·04	+ ·063	2·98	+ ·02
7	3·10	+ ·125	2·76	— ·217	2·80	— ·16
8	3·00	+ ·025	3·08	+ ·103	2·94	— ·02
9	3·02	+ ·045	3·04	+ ·063	2·80	— ·16
	2·975		2·977		2·96	

VALUES.

Roasting	2·975 ± 0·03	oz. gold per ton.
Nitre	2·977 ± 0·03	" "
Nitre-Iron	2·96 ± 0·03	" "
Combined	2·99 ± 0·01	" "

Discussion of Methods and Results.—The combined method, which was adopted for the more accurate determination of the value of the ore, is liable to error from several sources, of which may be mentioned.

(*a*) Solution of gold in the acid.

(*b*) Loss of gold when igniting the filter, and roasting off sulphur.

(*c*) Loss in the crucible fusion, and subsequent cupellation.

(*a*) It has long been known that, if nitric acid contain as impurities hydrochloric, or nitrous acids, gold is attacked and dissolved by the reagent. G. H. Makins in a paper "On Certain Losses of Precious Metals in Some Operations of Assaying,"* has shown that gold is soluble in nitric acid, free from chlorine, a reaction, which he attributes to the presence of nitrous acid. He points out, however, that the actual loss is very small, since the presence of gold was noticed only in acid which had been used many times. T. K. Rose,† while discussing the accuracy of the bullion assay, states that he found in the spent acid, from the second boiling of the cormets, after standing several weeks, 1.2 mgs. of gold per litre, and that, even after dilution with 25% of water, and another interval of several weeks, 0.45 mgs. per litre were still present. The solution of gold in nitric acid, probably as auric oxide, is also mentioned by A. H. Allen.‡ Reviewing the evidence presented by the authorities, it appears improbable that the very slight percentage which is dissolved, would have any appreciable influence on the result under consideration, although it must be remembered that during the operation of removing the pyrites, large quantities of the oxides of nitrogen are evolved, regarding whose offect, in the presence of nitric acid, it is impossible to make a definite statement. The precaution which was taken to remove hydrochloric acid, or free chlorine, is believed to be the best possible, and the acid may be said to contain only traces of those impurities.

(*b*) As has been stated previously, it is extremely unlikely that any loss of gold takes place, during the quiet heating and ignition of the filter and its contents.

(*c*) The errors, introduced by the crucible fusion and cupellation, have been studied by Furman, who, by determining the differ-

* *Journal of the Chemical Society*, Vol. 13, p. 77.
† *Journal of the Chemical Society*, Vol. 63, p. 710.
‡ *Chemical News*, Vol. 25, p. 85.

ence on a known quantity of gold, finds that the average loss is 0.3%. The direct result obtained has, therefore, been increased in this proportion, the correction amounting to 0.03 mgrm.

The mean values of the determinations agree as closely as could be expected, and this points to the probability that the sum of the errors of each series is nearly equal to zero. Among the individual results quite serious discrepancies are noticeable. It must be remembered, however, that a single result may be affected by two errors, both tending to depress or elevate the value, which would give rise to a comparatively large difference, from some other result similarly affected. To illustrate, suppose that a true value, 2.96 mgs., is lowered by an error in weighing of 0.03 mg., and an error in sampling of 0.06 mg., the result becomes 2.87 mgs., while if instead, the effect were to raise the observed value to 3.05 mgs., the difference between the two would amount to 1.8 mgs. It is to errors of this nature, rather than to defects in the methods themselves, that the discrepancies are probably due. It may be concluded, that, as far as can be determined by the methods and apparatus available, the processes, which were investigated, are accurate, at least for ores whose value does not exceed 3 oz. of gold to the ton. In the opinion of the writer, they are also accurate for ores, whose richness will much exceed the limit just mentioned.

A slight contribution may be made to the subject of sampling. Since the assays were carried out in a uniform manner, any error inherent in the process will have been constant, while the probable maximum error in weighing has already been shown to be within 0.03 mg. The large differences from the means, must then have been caused by the sampling, and these results are indicative of the value to be attached to samples, selected in the special manner adopted. Of a total of 27 results, the difference from the mean in each series reached in,

9 cases	0.1 mg.
3 "	0.2 "
1 "	0.3 "

A number of statements can readily be made as to the limits of variation. For example, if an ore, not richer than 3 oz. to the ton, be passed through an 80-mesh sieve, the greatest variation from the true value will very rarely exceed 0.3 oz. per ton, while the chances are 26 to 1 that the difference will reach that amount.

As was stated in the paragraph, quoted at the commencement of this paper, some assayers hold that gold is lost during the operation of roasting. When salt is added, this certainly is the case*, but it appears from the experiments, herein described, that the deficiency must be extremely slight. The decrepitation of the pyrites, at the beginning of the operation, is probably responsible for a very small mechanical loss.

Conclusions—From the results of the experiments, the following inferences may be made:

1. That the methods, which were investigated, will give accurate results, apart from errors of sampling.

2. That if 5 lbs. of an ore, not richer than 3 oz. to the ton, be crushed to pass an 80-mesh sieve, the maximum difference between the values of two distinct samples will not exceed 0.3 oz. In a series of separate samples, 96% will not differ by more than 0.2 oz. per ton from one another.

3. That the loss of gold, which is said to accompany roasting, is either inappreciable, on an ore of the value just mentioned, or does not take place at all.

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* Vide, Küstel, "Roasting of Gold and Silver Ore," 1880.
" Aaron, "Leaching of Gold and Silver Ores," 1881.
" Christy, Trans. Am, Inst. Min. Eng., 1888.
" Stetefeldt, Trans. Am. Inst. Min. Eng., Vol- XIV.

THE SHAFT GOVERNOR.

By Robert W. Angus, B.A. Sc.

The steam engine governor, more especially the one which is the subject of this paper, has become of so much interest to engineers and has been so much developed within the past few years, that I have thought the attention of the members of this Society might profitably be called to the subject. As far as I have been able to find, the literature on the subject is somewhat scarce, most of that extant being in the transactions of engineering societies, and therefore cannot be in very broad circulation.

Webster gives the following definition: "Governor—a contrivance applied to steam engines, water wheels and other machinery to maintain nearly uniform speed when the resistance and motive force are variable."

The sole object of the governor seems then to be to regulate the speed, and this result may obviously be accomplished in a number of ways, such as (1) by using a suitable brake which can be applied or released as the external load varies, (2) by changing in a suitable manner the weight of fluid entering or passing out of the working cylinder of the machine, and in many other ways.

In the case of steam engines, the governors for which only are to be dealt with here, the first method suggested above does not seem to be used at all, the almost universal practice being to adopt the first part of the second method, that is, to control the weight of working fluid admitted to the cylinder. It is true the latter part of the second method has been strongly advocated by some successful engine-builders, but at present there do not seem to be any cases in which it is made the only governing factor and it is generally used as an adjunct to the former method.

Two classes of governors have been suggested to automatically control the weight of steam admitted to the cylinder, and these derive their names from the forces used in their adjustment, and their position on the engine respectively. They are (*a*) *The Gravity Governor*, in which a pair of weights is made to move in a horizontal plane in a circular path around a vertical spindle, the spindle being driven at a speed proportional to that of the crank shaft of the engine. For equilibrium in any plane, the centrifugal force of the balls must be just balanced by the attraction of gravity on them; (*b*) *The Shaft Governor*, which is so named because it is always placed on the crank shaft, and usually occupies one of the driving pulleys.

It is with the latter class of governor that I propose to deal, but before doing so I wish to briefly call attention to the former class, as it was the original form and is still much in use.

It was invented, as long ago as 1650, by Huyghens, who used it to regulate an horological mechanism, and in the year 1784, Watt, recognizing the need of some method of regulating the speed of his engine, applied this governor. As used by him, however, it scarcely conforms to the definition of governor, since a wide range of speeds was possible, the mechanism practically only preventing the engine from going too fast. This governor is shown in Figure 1, and consists of a vertical spindle *A* driven by, and therefore at a speed proportional to, the crank shaft of the engine. To this spindle two arms, *B B*, are pivoted, having at their extremities weights, *W W*, the pivots being so fixed that the weights are caused to rotate with the spindle, but have freedom of motion in a vertical plane. The weight arms *B B* are connected by light rods to a loose sleeve, *C*, sliding on the spindle, *A*, which in turn communicates by means of the rod, *D*, with the butterfly valve in the main steam pipe.

At any certain speed it is evident there will be a definite position of the weights, *W W*, in which their moment about the pivot due to the action of gravity on them will be just balanced by the moment about the same point due to centrifugal force, so that for any speed there will be a position of equilibrium of the weights. It will, therefore, be evident that it is possible to so connect the weights and valve in the steam pipe that the latter will close when the speed increases and open when it decreases, thus regulating the amount of steam per stroke flowing through the pipe.

More particularly, suppose that the figure corresponds to the position which the balls would occupy at normal speed, there being in this position equilibrium between the gravity and centrifugal moments. An increase of speed will then increase the centrifugal moment on the weight relatively to that of the moment due to gravity, so that the balls will rise to a higher plane, drawing up the sleeve, C, and thus partially closing the valve in the steam pipe and admitting a less weight of steam to the cylinder, which will, of course, tend to reduce the speed to the normal. A decrease of speed will cause the opposite result. If such a governor is to keep the speed constant the path of the balls should evidently be such that they will remain in equilibrium in any plane from that of their lowest to their highest range, at the same speed. If this condition can be realized the throttle valve may be fully open or nearly closed, according to the demands of the load without change of speed, which is what is desired in a governor.

Let us glance briefly at the conditions governing the position of the plane of rotation of the balls:

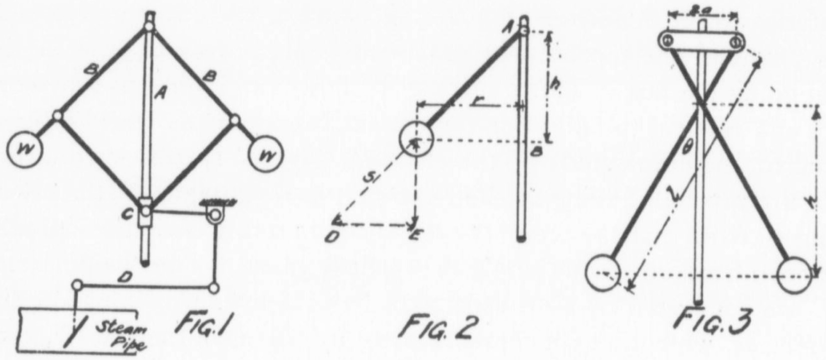


FIG. 1 FIG. 2 FIG. 3

FIGS. 1, 2 AND 3.

In Figure 2 let $A\,B$ represent the vertical spindle, W the ball, the centre of which is at C, and let $A\,C$ represent the weight arm. In this position the balls are moving in a circle of radius $B\,C$, which may be designated by r, and the plane of their rotation is at a distance $A\,B$, denoted by h—h is called the height of the governor—below the point A of suspension of the weights.

Suppose now that the spindle has an angular velocity ω radians per second, then, considering only one ball, its centrifugal force will be

$$F=\frac{W}{g}r\omega^2$$

Where g is the acceleration due to gravity and W is the weight of the ball; and the moment of this force about the point A will be

$$Fh=\frac{W}{g}r\omega^2h$$

since the force acts along the radius r. The vertical force due to the action of gravity will be

$$G=\frac{W}{g}\cdot g=W$$

and its moment about A is

$$Gr=Wr$$

For equilibrium in the given plane these moments must be equal.

$$\text{Therefore } \frac{W}{g}r\omega^2h=Wr$$

$$\text{or } h=\frac{g}{\omega^2}$$

which shows that if ω is to remain constant h must also, from which it may be shown that the balls should move in a parabolic path. In order to obtain this motion approximately the weight arms may be crossed and pivoted at a distance a from the spindle, as shown in Figure 3.

A practical application of the formula

$$h=\frac{g}{\omega^2}$$

to a particular case will show, however, that in general h so found for any speed will be very small. For example, take a governor revolving at 120 revolutions per minute. Then

$$\omega=2\pi\times\frac{120}{60}$$

and hence

$$h=\frac{32\cdot16}{(4\pi)^2}=0\cdot2036 \text{ feet or } 2\cdot44 \text{ inches,}$$

which is too small for any design.

In order to increase the height Charles T. Porter devised the plan of increasing the weight of the sliding sleeve, which increased the gravity effect without affecting the centrifugal force, and it may easily be shown that if a weight w be placed on the sleeve the total height h of the governor will be given by

$$h=\frac{W+w}{W}\cdot\frac{g}{\omega^2}$$

so that h may be made as great as desired.

This gives then, in a brief form, the advances that have been made in the perfection of this form of governor, and it would seem if only the controlling mechanism needed improvement there should not be great difficulty in making the gravity governor as nearly isochronous as desired, but its method of application to the engine was at fault, principally from the reason that the controlled valve was too far away from the cylinder in the very best forms of engine.

On the defects of this application I cannot do better than refer to a paper* read by Mr. Wilson Hartnell of Leeds, before the Institute of Mechanical Engineers, in which he has shown that the control of the speed by regulating a valve in the steam pipe is neither economical of fuel nor steam, and at the same time admits of great irregularities in the speed of the engine.

In order to understand the reasons for this it should be remembered that in any case in which the eccentric is rigidly keyed to the crank shaft—as it is where the governing is done through the throttle valve—the cut-off of the steam will always occur at the same distance from the beginning of the stroke, and this distance must evidently be great enough to admit sufficient steam under the least allowable boiler pressure to drive the engine under full load. If then an engine running under normal conditions have, say, one-half its load suddenly thrown off, the tendency will be for a sudden increase of speed which will be participated in by the governor and the throttle will be partially closed. On the next succeeding stroke then, since the eccentric is fixed, the admission part will remain open for the same fraction of the stroke as before, but since the throttle has been partially closed a less weight of steam will be admitted than during an ungoverned stroke, a consequence of which will be that the weight of steam between the throttle and the cylinder will expand, decreasing the admission pressure below that of the boiler and consequently, since the temperature of the steam will be lowered, cooling the cylinder walls. The loss by the resulting condensation where boiler pressure steam is again admitted will be readily understood.

So much then for the economical value of this method, but let us now look at its efficiency as a speed controller:

In order to see this more clearly let the engine be running under normal load and let the *whole* load be suddenly thrown off. There will then be a sudden increase of speed, and the governor participating in this will immediately close the steam pipe. There is, however, a

*"*On Governing Engines by Regulating the Expansion.*" Proceedings Institute of Mechanical Engineers, August, 1882.

considerable volume of steam between the throttle valve and the cylinder, and this will expand and keep up the increased speed for several strokes, since it requires very little steam to run the engine under a friction load only. When the speed does finally slacken and the valve opens, these spaces before referred to have to be again filled before the cylinder is affected so that a decrease considerably below the normal speed may result.

The above method then is evidently defective both in point of economy and speed, and it would appear that although the defects may be considerably lessened by placing the throttle close to the cylinder, yet there is always the volume of the steam chest between them, so that such a precaution cannot prevent the troubles.

Before going farther, I would wish to draw attention to the fact that the gravity governor is usually used with slow speed engines—probably less than one hundred revolutions per minute—while the shaft governor is generally used on high speed engines—of, say 250 and over, revolutions per minute—and, if each has an opportunity of acting in one stroke and will not permit a change in speed of over one revolution, the former cannot act in less than $\frac{60}{200}$ seconds, and will allow a variation in speed of one per cent., while the latter can act in $\frac{60}{500}$ of a second, and only allows a variation of $\frac{2}{5}$ of one per cent. This fact, I believe, helps to show such a remarkable superiority of the shaft governor over the gravity one.

Engineers then soon turned their attention to some other means of governing, and began to see what could be done in the way of operating directly on the slide valve, thus avoiding as much as possible the waste spaces, and since they still adopted the idea of controlling the weight of steam admitted per stroke, the only possible solution seems to have been to vary the point of cut-off, and this leads to the automatic cut-off governor.

What seems logically—although I am not sure of its chronological order—to have been the stepping stone between the gravity and shaft governors, is one which is the invention of Mr. John F. Allen, in which an ordinary gravity governor of the Porter type is made to vary the throw of the eccentric and travel of the valve, and consequently the point of cut-off. The device seems similar to the method of controlling locomotives, except that the operation of the link lever is automatic. A part of the description of this invention, copied from a catalogue of the Porter-Allen steam engine, written about 1882 by Charles T. Porter, is here given.

"*The Valve Gear and Valves.* These are the invention of Mr. John F. Allen, and in the history of engineering will hold a prominent position, not only on account of their originality and excellence, but because they first made high speed in the variable cut-off engine practicable. . . ."

"This system employs the link motion and flat slide valve, the superiority of which over the countless forms of valves and movements that have been designed is shown by the fact that, in the trying service of the locomotive, nothing else has been found to answer.

"The eccentric is placed on the shaft in the same position with the crank instead of at right angles with it, as it is in the ordinary valve gear, and it cannot be advanced from that position. The lead of the valves is adjusted by other means. The first requirement in

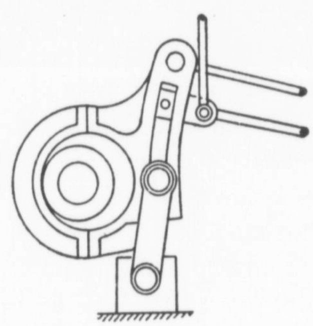


FIG. 4.

this system is that the crank and the eccentric shall have coincident movements, and so shall arrive on their dead points, or lines of centres, simultaneously."

"The construction of the link is also shown in the cut (Figure 4). It is of the form known as the stationary link, and consists of a curved arm, partly slotted, formed in one piece with the eccentric strap, and pivoted to its middle point on trunnious, which vibrate in an arc (whose cord is equal to the throw of the eccentric) about a sustaining pin secured rigidly to the bed. The radius of the link is equal to the length of the first rod by which its motion is communicated to the admission valves."

"In the slot is fitted a block from which the admission valves only receive their motion. This block is moved by the action of a

(Porter) governor, which thus varies the point of cut-off. If the centre of the block be brought to the centre of the trunnions, the valve is not opened at all, except by the lead given to the valves, and this opening has closed before the piston has sensibly advanced. If on the other hand the block is brought to the end of the slot as here represented, the steam is not cut off till the piston has reached the half stroke, which is the usual limit of admission.

"The exhaust valves are driven from a fixed point on the link and have, of course, an invariable motion."

Here we have then a contrivance in which the quantity of steam required per stroke is regulated by changing the length of the part of the stroke during which it is admitted, the steam being admitted to the cylinder at boiler pressure.

Having seen the advantages of the automatic cut-off, engineers were not slow in adopting it, and a simple method of affecting this was accomplished by the invention of the Shaft Governor, by Prof. John E. Sweet, in which the eccentric had a slot cut in it so that its centre could be slid across the shaft, thus changing the distance between the centre of the eccentric and centre of the shaft, and the consequent travel of the valve. The mechanism for controlling this motion of the eccentric is now to be dealt with somewhat in detail.

THE SHAFT GOVERNOR.

Before going into the governor, one point might be mentioned in regard to the opportunities offered in this form of regulator for instantaneous action. It will, no doubt, be understood that early compression will tend to slacken the speed, offering as it does resistance in the early part of the stroke, so that if the compression becomes early as the cut-off does, there will be a double chance in each stroke for control. Now in cases where one valve serves the double purpose of admitting and exhausting the steam, it may be seen from a simple study of the valve diagram, that in the case where the admission line is lengthened, the compression line is shortened and *vice-versa*, hence when the circle of the eccentric centre becomes smaller the cut-off becomes earlier, the compression also earlier, and hence if the load is suddenly thrown off an engine controlled by a shaft governor (which changes the effective throw of the eccentric) immediately after cut-off in one end of the cylinder, the governor will immediately respond, increase the length of the compression line, and consequently begin to prevent an increase in speed during the same stroke as the change of load has occurred in, and does not need to wait for the next stroke as in the case of throttling-governed engines.

An elementary form of what has been a most common type of this governor is shown in Figure 5, in which a circular disk is usually keyed to the crank shaft and caused, therefore, to rotate at the speed of the shaft. This disk, which is usually one of the driving pulleys of the engine, carries two pivots, *a*, normal to its surface and plane of rotation. These pivots form the axles, about which a ball, *P*, and its supporting arm, *p*, are free to turn in the plane of rotation. Spiral springs, *s*, of which one end is attached to the disk, and the other to the arm, *p*, resist the moment of the weights, *p*, about their supporting pivots, *a*, produced by rotation of the disk. These arms, *p*, are also connected by means of a light and rigid link to the eccentric eye.

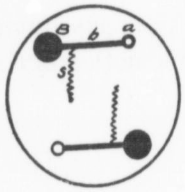


FIG. 5

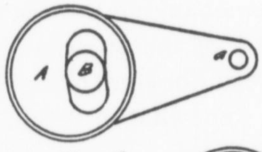


FIG. 6

FIG. 7

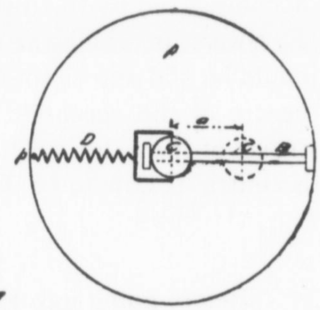


FIG. 8

The eccentric eye is of one or other of the two forms shown in Figures 6 and 7.

In the form shown in Figure 6 the eccentric, *A*, is pivoted to the rotating disk above referred to, at a point *a*, and is slotted so as to clear the crank shaft, *B*. It is seen that the throw of the eccentric will be a maximum when one end of the slot is against the shaft, and will be a minimum when the centre of the pivot, *a*, the centre of the eccentric *A*, and the centre of the crank shaft, *B*, are all in one straight line.

Figure 7 shows the other form in which the slot is straight and the centre of the eccentric moves in a straight line across the shaft, this motion being obtained in a suitable manner.

The former type is the more usual, and in fact in regard to the latter, Halsey* says: "So far as is known to the writer, there are but two engines in the American market employing a shifting eccentric" (Figure 7). "These are the Armington and Sims and Russell (Giddings) engines." These eccentrics give a constant lead† to the valve, and are on that account superior to the former ones, but since the motion is so difficult to obtain have not been generally adopted.

The mechanical methods of connecting these eccentrics to the moving weights are varied to suit circumstances and in a few of the examples of governors given in the sequel will be clearly shown; suffice it to say here that the connections should be made so that the two weights assist one another in moving the eccentric to its position of equilibrium under any given load.

Let us now consider the conditions of equilibrium of a weight rotating about an axis. In Figure 8 let A be a disk, supporting a radial rod B, on which a ball C is free to slide from end to end, the inner end being so fixed that the ball in its innermost position will have its centre of gravity in the axis of the disk. Let a spiral spring D be attached between the ball C and a point p situated on the disk in the same diameter as the rod B, but at the other end of it as shown.

Suppose now that the spring D is just fully extended when the ball is in its innermost position, and suppose the disk to be rotated with constant angular velocity about an axis through its centre, and perpendicular to the plane of the paper. Then under these conditions and a spring of the proper strength the ball C will remain in any position on the radial rod in which it may be placed whether far or near the axis of the disk, assuming that we may neglect friction, and the effects of gravity, which will be comparatively small.

This is easily shown, for with the above assumptions the only forces acting on the ball are those exerted by its centrifugal force, which acts away from the centre, and the pull of the spring which acts towards the centre of rotation.

If we assume an angular velocity, ω, radians per second of the disk and that the ball is in equilibrium in some position at distance a from the centre of the disk, then we will have the centrifugal force $C=Ma\omega^2$ where M is the mass of the ball; and the force exerted by the spring will be $F=Sa$, where S is the strength of the spring, *i.e.*, the number of pounds that will be necessary to increase its length by one unit.

* Halsey—Slide Valve Gears, fifth (1896) edition, p. 80.

† For a general analysis of valve diagrams and the motions of the valve the reader is referred to Halsey's Slide Valve Gears, which is of great value in this connection.

For equilibrium these two forces must be equal, *i.e.*, $Sa=Ma\omega^2$ which is independent of a, or, in other words, if the strength of the spring is proportioned according to the law $S=M\omega^2$, the ball will remain wherever placed on the bar, B, at a given velocity, ω, or, in other words, the system is *isochronous*.

In order to get clearly at the meaning of this particular condition of affairs let us take a practical case. Given a ball weighing twenty-five pounds held in equilibrium by a spiral spring, at a distance of twelve inches from the centre of a disk rotating at 200 revolutions per minute:

First, let it be required to find the strength of the spring for isochronism. The formula given already $Sa\omega=Ma\omega^2$ becomes of the form

$$Sa=\cdot 0000284\ Wan^2$$

where inch and pound units are used, *i.e.*, where W is the weight of the balls in pounds, a is measured in inches, n is the number of revolutions per minute. Substituting then the elements of the given problem in the above formula we find the value

$$S=\cdot 0000284 \times 25 \times (200)^2=28\cdot 4 \text{ pounds}$$

being the strength of spring required for isochronism, and this condition will also insure equilibrium at distances ten and fourteen inches from the axis.

Figure 9 represents graphically this result. Let W represent the axis of rotation, and distance along the horizontal line represent distances from this axis of the disk, and let vertical ordinates represent the forces. At W_1, a distance of twelve inches from the axis the centrifugal and spring forces are each $28.4 \times 12=380.8$ and are represented by the vertical line W_1S_1, and since these two forces are equal for any position of the ball, the locus of the tops of ordinates representing the forces will be a straight line WS_1 cutting the axis at W since where $a=o$ $C=F=O$.

Next, let the strength of the spring be given and let us see the conditions of the problem. Two cases may occur; first, that in which the spring is stronger than that required by the condition of isochronism, and second, that in which the spring is weaker.

For the first case, let us suppose a spring of fifty pounds strength, then, since equilibrium is desired at a distance of twelve inches from the axis of rotation, the centrifugal force will be $340\cdot 8$ pounds as before, and hence the spring must have an extension of

$$\frac{340\cdot 8}{50}=6\cdot 816 \text{ inches.}$$

But since when the disk stops rotating the centrifugal force is zero, so also will the spring extension be and the ball will then be held at a distance

$$12-6{\cdot}816=5.184$$

inches from the centre of rotation and not in the same point as before. This case is shown in Figure 10, in which W represents as before the centre of rotation and ordinates to the line, $WC_3C_1C_2$ represent the

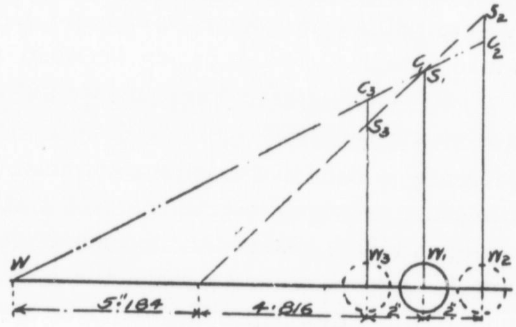


FIG 10

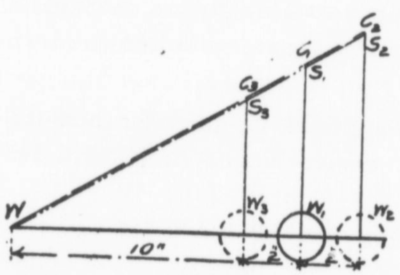


FIG. 9

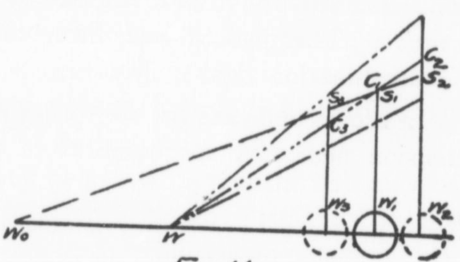


FIG. 11

centrifugal force precisely as before. But these same ordinates no longer represent the spring tension, for at a point 5·184 inches out $F=O$ and since the locus of the tops of the ordinates representing the spring tension is a straight line, and since this line must pass through C_1 it will be found to be $S_3S_1S_2$, and it is readily seen that at the given speed equilibrium can only occur at a radius of twelve inches from W. For equilibrium at W_3 the speed must be

$$n=\sqrt{\frac{F}{.0000284\ Wa}}$$

$$\text{or } n=187.9\sqrt{\frac{F}{Wa}}=187{\cdot}9\ \sqrt{\frac{50\times4{\cdot}816}{25\times10}}=184{\cdot}1 \text{ revs. per minute.}$$

(The strength of the spring being in this case 50 and its extension 4·816 inches, the force, F, is given by $F=50\times 4{\cdot}816=240{\cdot}8$.) For equilibrium in the position W_2 at distance fourteen inches from W the speed must be

$$n=187{\cdot}9\sqrt{\frac{50\times 8{\cdot}816}{25\times 14}}=210{\cdot}8 \text{ revolutions per minute.}$$

If, therefore, these two positions represent the extreme positions which the ball must occupy from variations of the load on the engine, it is clear that such a governor will allow a decrease of speed of 8% below the normal, and will also permit a total variation between maximum and minimum loads of $210{\cdot}8-184{\cdot}1=26{\cdot}7$ revolutions per minute, or nearly 13%, which is of course too great.

Although the governor is defective in that way, however, it presents another interesting feature which rests in the fact that when any position of equilibrium is obtained, a considerable force is necessary to disturb it. Thus in this case, in order to make the ball remain in equilibrium at ten inches from the centre, it would be necessary to apply an external force equal to $C_3\ S_3$ pounds, so that such an arrangement, since it will not be disturbed by small external forces, is said to be *stable*. This should be contrasted in this regard with the *isochronous* combination already described.

In the second case let the spring strength be 24 pounds, then its extension necessary for equilibrium at twelve inches radius will be

$$\frac{340{\cdot}8}{24}=14{\cdot}2 \text{ inches,}$$

hence, when the disk is at rest the ball will be at a distance $12-14{\cdot}2=-2{\cdot}2$ inches from its centre, *i.e.*, 2·2 inches on the opposite side of the centre from which the twelve inches is measured. This condition of affairs is shown in Figure 11, where ordinates to WC_1 represent as before the centrifugal force and those to W_0S_T the spring forces, the distance W_0W_T being 2·2 inches. The same point in regard to the variation in speed as before referred to, should be again noted here, but we have an altogether different result in regard to stability, for it will be noticed if the ball is slightly disturbed to the left it will fly right to its inner stop, and if to the right to its outer stop, the combination is therefore easily disturbed by the slightest external forces, and is therefore said to be *unstable*.

These simple examples therefore show that a combination of ball and spring placed as above on a rotating disk is *stable* or *unstable* according as the spring is too strong or too weak to fulfil the condition of isochronism.

In the ordinary form of shaft governor already referred to at Figure 5, adjustment is made for isochronism as near as is practicable, and stability is obtained by means of an additional part such as a dash pot, which will be mentioned in the sequel.

Let us now return to the simple form of governor shown at Figure 5, and make an analysis of the conditions and requirements in regard to its general dimensions and the position of its parts to produce isochronism.

In the first place the disturbance due to the differing effects of gravity at different parts of the revolution is neutralized by making the two parts of the governor similar and symmetrical about a diameter of the pulley, so that it will be evident that the effects of the different balls will just counteract one another in this respect.

For a number of other propositions in reference to the position and strength of the spring, the size and position of the weight, etc., I wish to refer to a most excellent paper by Mr. A. K. Mansfield, of Salem, Ohio, entitled, "Notes on the Theory of Shaft Governors," which was read before the American Society of Mechanical Engineers, in June, 1894, and may be found at page 929, volume xv. of their transactions. In most of these propositions I have adhered closely to his paper, and have copied some of the cuts.

The first proposition which may be proved is that the ball may be placed in any position on the weight arm, *i.e.*, at any distance from the weight pivot, and the centrifugal moment will remain constant provided the weight varies inversely as the distance. This seems evident without proof, but may be easily demonstrated on reference to Figure 12, in which A is the centre of rotation, B the weight, and C the weight pivot, and let E be any other assumed position of the weight; it will then be required to show that if the weight at E is greater than the weight at B in the ratio

$$\frac{CB}{CF}$$

the centrifugal moments about C will, in the two cases, be equal.

Under constant speeds the centrifugal forces will be proportional to the radial distances of the weights from the centre of rotation, and may therefore be represented by these radii, and since the direction of the centrifugal force is from the centre, the moment of the ball at B is

$AB \cdot CG$ multiplied by its weight, and at E is given by $AE \cdot CD$, multiplied by the weight at E, so that we simply require to show that

$$\frac{AB \cdot CG}{AE \cdot CD} = \frac{BC}{CE}$$

Since the triangles BFA and BGC are similar, we have

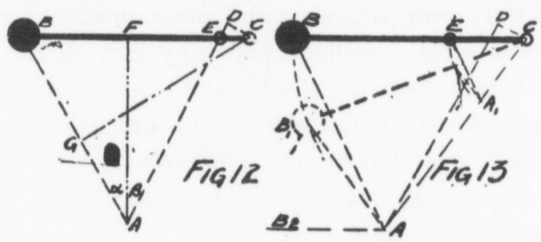
$$AB{:}AF = CB{:}CG \text{ or } AF = \frac{AB \cdot CG}{BC}$$

and since the triangles AFE and CDE are similar we have

$$CE{:}CD = AE{:}AF \text{ or } AF = \frac{AE \cdot CD}{CE}$$

$$\text{Hence } AF = \frac{AB.GC}{BC} = \frac{AE \cdot CD}{CE}$$

$$\text{or } \frac{AB \cdot CG}{AE \cdot CD} = \frac{BC}{CE} \text{ as was to be proved.}$$



This seems to show that as far as turning effect is concerned, no advantage can be gained by moving the ball closer or farther from the pivot, provided that the weight of the ball is altered in the proper proportion.

Now, in the theory of a ball properly placed for isochronism, it was shown that a spring may be designed with proper strength to produce this condition if its line of action passes through the centre of rotation, and its zero of extension is just obtained when the centre of gravity of the ball coincides with the axis of rotation. Referring now to Figure 13, in which as before A is the axis of rotation, C the weight pivot and B the weight, it will readily be seen that a spring proportioned as previously described and which has its zero of extension at A and is swung to move pivotally about this point, will make the whole device isochronous in all positions of the ball B, and from the proposition just proved the condition of isochronism will not be altered by attaching the spring at any point E on the arm of the weight, provided the other conditions be complied with and the spring is increased in strength in the proper ratio, so that the spring may lie anywhere between the positions AC and AB_2 parallel to BC_1 for the position of the ball B shown here.

Again, if through the position E, a line EA_1 be drawn parallel to BA, meeting the line AC in A_1 it seems evident from the propositions already proved, that a spring pivotally swung about A_1 and being attached to the weight arm at E, and having its zero of extension at A_1 is properly placed to produce isochronism, and further, it seems to be evident that instead of the spring being attached at a point E on the weight arm in a line joining the centre of gravity to the pivot, it may be attached at any point F, Figure 14, provided, also, the line of zero

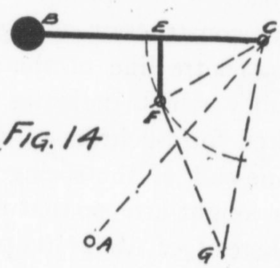


FIG. 14

tension, CG of the spring, be turned back from the line AC, by the angle ACG, equal to ECF. These several propositions seem to follow directly from what has already been shown of the principles governing isochronism, so that no proof is here attempted.

It would, therefore, appear that the point of connection of the spring to the weight arm and the direction of action of the spring may be selected conveniently and the device may still retain the property of isochronism, provided the spring be properly proportioned. Now, since the spring strength must just balance the centrifugal force when the spring is pivoted at A, and has its end connected to the centre of gravity of B, it is evident that if it is still pivoted about A, but has its end connected to the weight arm at E, the strength of the spring must be increased in the ratio

$$\frac{BC}{EC}$$

but if also the spring be made to swing pivotally about a point A_1, Figure 13, it must again be increased in the ratio

$$\frac{AC}{A_1C}.$$

Hence the spring force required for a spring pivotally swung about A_1 having its zero of extension at this point and its end pivoted to the weight arm at a point E, may be found by multiplying the centrifugal force of the ball by the ratio

$$\frac{BC}{EC} \cdot \frac{AC}{A_1C}$$

or what is the same thing, the spring force per inch of extension is equal to the centrifugal force of the ball per inch of radius multiplied by the ratio

$$\frac{BC}{EC} \times \frac{AC}{A_1C}.$$

EA_1 represents the extension of the spring.

Now in the ordinary construction of a governor it is evidently impracticable to keep the centre line of the spring always over the point of the governor pulley which indicates its zero of extension, a condition which is required for isochronism, for it is evident that in the actual construction one end of the spring must be pivoted to the pulley and one end to the weight arm, so that the spring swings pivotally about its outer end instead of about its point of zero extension. In practice, therefore, approximate isochronism only is attempted, and this may be obtained evidently by making the pivotal point of the spring as far distant as possible from the weight arm, and by so constructing the parts that the weight arm shall have a minimum angular motion. Both these conditions assist in producing approximate isochronism. It may easily be shown that where the spring is so pivoted that its central line always remains parallel to itself and is adjusted in one position for isochronism, it will be exactly isochronous in some position between that and the one where the ball is at the centre of rotation.

In Figure 15 let A be the centre of rotation, B the ball and C the weight arm pivot, and let CA be the position of the arm where the ball coincides with A, and let CB_1 be a position midway between CA and CB. Let the spring be pivoted at some point O on BA produced in such a way that its centre line will be always parallel to BA_1, *i.e.*, when the ball is at B_1 the centre line of the spring will be O_1B_1. It may be shown now that if the spring is isochronously adjusted for the position B it will also be isochronous in the position B_1. For the moment of the spring about C is proportional to its extension A_1B_1 and the arm CB which may be denoted by R so that calling the angle $ACB_1, -\alpha$ the spring moment will be $R \cdot R \sin\alpha = R^2 \sin\alpha$, while the moment of the centrifugal force of the ball is proportional to $AB_1 \times CD$ and will be

$$2R \sin\frac{\alpha}{2} \times R \cos\frac{\alpha}{2} = 2R_2{}^2 \sin\frac{\alpha}{2} \cos\frac{\alpha}{2} = R^2 \sin\alpha$$

and since these expressions are equal it is evident the ball is isochronous in the position B_1.

It is not nearly so easy to prove that there are two positions of isochronism where the spring is pivoted at a reasonable distance, so that its centre line moves radially about the pivot, but it would appear that such is the case, especially where the angle BCB_1 between the extreme positions of the arm is small and the length of spring great.

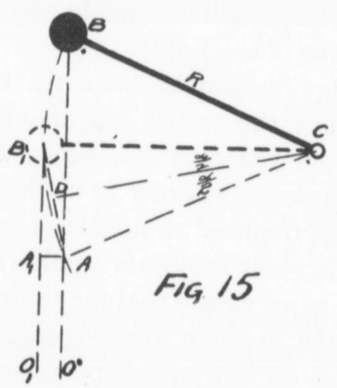


FIG 15

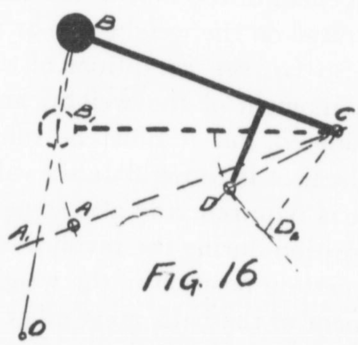


FIG. 16

In usual constructions then the spring would be pivoted at O Figure 16 on BB_1 produced, and its zero of extension should be at A_1 on CA produced, as will be evident from preceding demonstrations, in order to produce isochronism at B and B_1 and the mechanism will be approximately isochronous in all positions between the limits B and B_1, and exactly isochronous in some one particular position about midway between these points. Should it prove inconvenient to connect the spring at O the above conditions will still hold if the pivot be along the line DD_1, the angle DCD_1 being equal to BCB_1, D being any point on the arm CB. In this case, however, the zero of extension of the spring will not be along the line CA, but along a line turned back from CA by an angle equal to the angle BCD.

In regard to the influence of the weight of the spring on the mechanism, little need be said as its effect may be calculated in any case. It may be remarked, however, that its effect is best allowed for by finding the equivalent weight and its point of application on the arm BC, which will produce about C the same moment as the spring does about this point. This weight should be taken into account in determining the centre of gravity line BC of the arm.

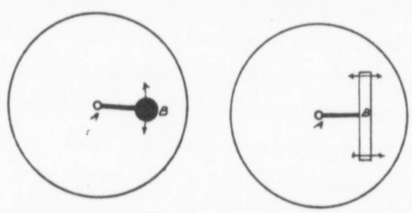
The weight of the link connecting the arm with the eccentric should also be taken into account in determining the centre of gravity line, but this is also done very easily, for since one end of the link is attached to the eccentric and one to the arm, all that is necessary to do is calculate the part of the link actually acting on the arm. This may be shown to be the weight which would—if the link were supported horizontally on two abutments—be the abutment pressure of the end of the link actually connected to the arm. Since the link is usually symmetrical, this part is one half its weight, so that the centrifugal effect of the link is the same as if one half its weight were concentrated on the weight arm at the pivot of the link.

So far, the conditions of exact equilibrium between the centrifugal moment of the weight, and that of the spring only, have been considered, but it must be remembered that the sole object of the mechanism is to regulate the valve travel, and in order to do this some force is required, and also some force is required to hold the eccentric in position during the revolution, and in order to obtain equilibrium in any certain position of the weights, it is necessary that the centrifugal moment of the balls must differ from the moment of the spring by an amount exactly equal to that required for the above purposes, and that of friction in the governor. To go fully into this question would require a paper far more comprehensive than the present one, and the subject will have to be investigated in the particular design in all cases. In no case can these disturbing elements remain constant, for the acceleration of the valve varies from zero about the centre of its travel to infinity at the end, and so there must be a continually varying force required to draw it along. This must be taken into account in the design of the governor, and while the force itself always remains, the governor mechanism may be made so powerful that its variations become negligible, and a condition of stability readily realized. For instance, if there be a one per cent. variation in these forces for a certain size of governor, properly proportioned, it will be evident that there will be twice as great a disturbing element as for a governor twice as powerful and so on. These variations, however, are prevented by other considerations and appliances from making the trouble that may be anticipated, and some of these appliances appear in the sequel.

Up to the present we have been considering conditions of equilibrium only, without considering what occurred while a change of position of the weights was taking place, that is, so far, what is usually called inertia has been left out of the calculation, and since it has no effect on the position of equilibrium, but only, as will be seen, on the time taken for a change from one position to another, it practically comes in as a separate problem, and may profitably be dealt with here at some length.

In order to first grasp a clear idea of what is meant let us take a few examples, and see what the inertia forces are.

Suppose a disk, Figure 17, to have a ball B attached to its



FIGS. 17 AND 18.

axis of rotation A, by a light but stiff rod, and suppose the disk to be rotated about an axis through A, perpendicular to the plane of the paper, at constant angular velocity. Then if rotation be continued the only force acting on the ball will be that which is commonly called centrifugal force, and which will act along the line, AB, and produces no turning moment, since the line of action is through the pivot. Assume now that the disk is suddenly increased in velocity, then the tendency is for the weight to be left behind, and there will therefore be a turning moment about the axis, A, since the ball tends to move in the direction of one of the arrows shown, and in that of the other one if the speed be suddenly decreased. After a short interval the changed velocity will be entirely participated in by the weight, and the moment will cease. The force producing this moment has been called *tangential accelerating force*, since it acts along a tangent.

It has been said there is only one effective force acting on the weight B, but this is not strictly true, for it will be evident that for each revolution that the disk makes, the ball also makes one on its own axis about its own centre of gravity. So long as the speed remains constant of course this speed of rotation of the ball is constant, and consequently no force is required to keep it up, but let the rate of rotation of the disk about its axis, and consequently that of the ball about its centre of gravity, suddenly change, then since the angular velocity of the ball changes, some force will be required to effect the change. It is true this force is negligible in the case shown in Figure 17, where the ball is round and pivoted as there shown, but let the weight be elongated as shown in Figure 18, then it is evident this turning effect may become great comparatively, and should receive as much attention as its companion accelerating force. This force described here is called *angular accelerating force*, and is becoming a very prominent factor in the more recent forms of governor. I do not desire to go too deeply into this subject, and will assume that these illustrations bring out the difference between these forces, and also that it is thoroughly understood that both of these forces become zero where no change of velocity of their centres of gravity occur. This does not necessarily mean a change in the rate of rotation, for evidently a change in radius of rotation will accomplish the same object even with a constant rate.

Another point should be noted here, and that is these accelerating forces may be made to oppose or assist centrifugal force, or may have a neutral effect. Take the tangential force for example; it will have no turning effect about the weight pivot, provided the line joining the pivot to the centre of gravity of the weight is perpendicular to the radius of rotation of the weight; it will assist the centrifugal force provided the rotation is in such a sense that the weight is in advance of the pivot and a tangent to the path of the former passes outside the latter, or if the pivot be in advance, and the tangent to the path of the weight passes inside the pivot. On the other hand these accelerating forces will oppose centrifugal force under the same conditions as before, provided the sense of rotation only be reversed.

A few sketches will convince the reader of the correctness of the above statements, and may serve to illustrate other cases which may occur.

It will therefore be observed that the mass of the weight may be so distributed about its centre of gravity and its supporting pin as to either assist in an extremely rapid adjustment under changed conditions, or, if this is not desired, a dash-pot effect—producing stability but slow adjustment—may be secured. Whatever may be the advantages of the second method, it seems certain that if the inertia forces are made to assist centrifugal ones, a very rapid adjustment is secured without sacrificing stability.

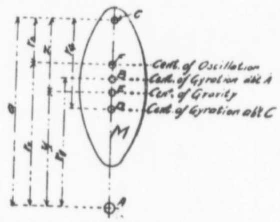


FIG. 19.

Consider the case shown in Figure 19, in which A represents the centre of rotation or shaft, M the flyweight which is pivoted to the rotating disk at some point C. Let F, B, E, D be the centres of oscillation, gyration for rotation about A, gravity, and gyration for rotation about C respectively, and let the various radii and distances be represented by the letters shown in the figure.

Now as the disk rotates the weight M moves about the centre A, but at the same time the centre of gravity E rotates about its pivotal point C, once for each revolution of C about A, and the senses of these rotations are opposite if the centre of gravity E always lies between C and A. Let us then assume that the disk is rotating about A with a constant acceleration α, then the weight which will be assumed to have a mass M—equivalent to its weight divided by the attraction of gravity on one pound—will produce a torque or turning movement about A, which is given by

$$T = I\alpha$$

where T is the torque, I is the moment of inertia of M considered as rotating about A; so that the value of I will be given by $I = Mr_1^2$, r_1 being the radius of gyration of M about A

$$\text{Hence } T = I\alpha = Mr_1^2\alpha$$

Since this is a moment about A the force which if acting at F the centre of oscillation would produce the same moment, is given by dividing T by r_2 and is

$$M\frac{r_1^2}{r_2}\alpha$$

and this force will produce a moment

$$Mr_3\frac{r_1^2}{r_2}\alpha$$

about the pivot C, which represents the moment about C due to the rotation of the weight about A. But since the weight has a rotation of its centre of gravity about C, a moment about the latter point will be produced equal to Mr_4^2, which is simply the moment of inertia of M about the pin C.

It will now be readily observed that a division of the former moment

$$Mr_3\frac{r_1^2}{r_2}\alpha$$

by the latter Mr_4^2 will give the effective angular displacement of M. produced by these forces simultaneously acting, *i.e.*, the effective displacement is

$$\frac{Mr_3\frac{r_1^2}{r_2}\alpha}{Mr_4^2}=\frac{r_3r_1^2}{r_2r_4^2}\alpha$$

which becomes

$$=\frac{r_3r_1^2}{r_2r_4^2}$$

for unit acceleration. If, then, it is desired to use these accelerating forces for rapid adjustment of the weights, the above fraction should be a maximum.

In order to reduce the fraction to a suitable form to find the conditions for such a maximum, we shall proceed as follows:—If we denote by x the radius of gyration of M about its own centre of gravity, then it may be shown that*

$$r_2=\frac{x^2+y^2}{y}$$

Also the radius of gyration of any mass considered as about a point a distance y from its centre of gravity is

$$\sqrt{x^2+y^2}$$

being the radius of gyration of the mass about its centre of gravity. Hence, since in the figure r_1 represents the radius of gyration about A we have

$$r_1=\sqrt{x^2+y^2} \text{ or } r_1^2=x^2+y^2$$

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*See WORTHINGTON—*Dynamics of Rotation*, page 124.

and similarly
$$r_4^2 = z^2 + x^2$$

$$\text{also } r_2 = \frac{x^2 + y^2}{y} = \frac{r_1^2}{y}$$

and
$$r_3 = a - r_2 = \frac{ay - y^2 - x^2}{y}$$

Substituting these values in the above fraction we get

$$\frac{r_1^2 r_3}{r_4^2 r_2} = \frac{x^2 + y^2}{z^2 + x^2} \cdot \frac{ay - y^2 - x^2}{y} \cdot \frac{y}{x^2 + y^2}$$

$$= \frac{yz - x^2}{z^2 + x^2}$$

$$= \frac{Myz - Mx^2}{Mz^2 + Mx^2}$$

Now it has been shown that the turning moment due to the tangential accelerating force is $I\alpha$, I being the moment of inertia about the centre of revolution, and α the angular acceleration, hence the amount of this force is found by dividing the moment by the radius to the line of action of the former, and hence the force is

$$\frac{I\alpha}{y}$$

but $I = My^2$ and taking α as unity we get the amount of this tangential accelerating force to be My, so that in the above formula Myz is the moment of this force about C, while the latter quantity Mx^2 is the moment of inertia about the weight's own centre of gravity, and is a negative moment about the same point C.

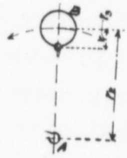

FIG. 20.

By successive differentiation of the above ratio with respect to x, y and z respectively, it will be found to become a maximum as x and z approach zero, and y approaches a in value.

If, as some writers seem to think, the centrifugal moment should be reduced to a minimum, the arrangement shown in Figure 20 fulfils very well the requirements of the above considerations, for in this case both the tangential and angular accelerating forces are a maximum.

If the effect of centrifugal force is not, however, reduced to zero, its rate of change should also be taken into account in determining the time of adjustment of the governor. This is readily done, since the centrifugal force $F = Mr\omega^2$ where r is the radius of rotation of the centre of the gravity of the mass M, and ω is its angular velocity. The rate of change of this force is

$$\frac{dF}{dt} = 2Mr\omega\frac{d\omega}{dt} = 2Mr\omega\alpha$$

where α is the angular acceleration and t the time. This should be taken into account and combined with the effect produced by the accelerating forces.

A vast difference of opinion seems to exist amongst engineers as to what are the important points and forces in the governor. Mr. F. M. Rites—from whose paper on "Shaft Governors" much of the preceding is taken—is of the opinion that the centrifugal force should be reduced to a minimum, and the inertia effects made important, while Mr. Frank H. Ball concludes* that the centrifugal force is the most important governing force, and that "angular accelerating force is next in importance because it is an unqualified help as an actuating force, and its practical usefulness is limited only by constructional considerations." Tangential accelerating force he regards as "of questionable utility, because of the disturbing forces that it is almost sure to put into operation."

Professor John E. Sweet, to whom too high a tribute cannot be paid by those who are interested in this piece of mechanism, of which he was the originator, said in an address to the students in Wisconsin University in 1895, "I can show that there is no possibility of the inertia element being applied to a governor beneficially, except as a dash-pot, and to call the thing by its right name is to call it an inertia dash-pot rather than an inertia governor."

"The claim made for the inertia governor is this: If the load be instantly removed from the engine the surging ahead of the fly-wheel will, as it exceeds the uniform motion of the inertia weights,

*Steam Engine Governors. By F. H. Ball, Trans. Am. Soc. Mech. Eng., Vol. XVIII.

shift the eccentric and cause an instantaneous complete or partial cut-off to take place, or if an excessive load is thrown on the engine the retarding of the wheel prolongs cut-off."

"Allow me first to direct your attention to the fact that the fly-wheel of an engine receives an impulse intermittently, receiving * * * a maximum about the middle of the stroke. * * * * The mean in work given off by these successive impulses being (aside from friction) that given off by the belt, which is ordinarily pretty nearly uniform, and probably from 60 to 75 per cent. of the maximum impulse. From this * * * * it will be seen that the disturbing element tending to vary the cut-off is only from mean load to light load, or usually not one-half the disturbing element that obtains in every half revolution of the engine, and the same is the result if the full load be suddenly thrown on."

It does, however, appear that the inertia effect is becoming in the later forms of governor the most important, especially that due to rotation of the weight about its own centre of gravity. In a number of governors recently brought out, the distribution of the mass shows the inventor's belief in the value of this effect, and the perfection with which this class of governors operate fully justifies the belief. One example of this form is given at the end of this paper.

I merely wish to conclude this paper by giving a few examples of typical forms which have from time to time appeared.

EXAMPLES OF SOME FORMS OF GOVERNORS.

What is said to have been the pioneer* shaft governor is illustrated in Figure 21, and is built by the Buckeye Engine Company of Salem, Ohio. It consists of a flywheel made in two sections, and bolted together on the shaft, and having pivots cast on two of the arms at *b*. On these are pivoted weight arms, carrying weights, *A*, which may be slid along the arms in either direction, these being the actuating weights in the governor; the object of adjusting the position of the weights evidently being to permit a change in the centrifugal moment by changing the radius of rotation. The end of the weight arms *e* are connected through the links *B* to the eccentric *C*, which is placed loosely upon the shaft.

The unbalanced part of the centrifugal force of the weights, *A*, is resisted by spiral springs *F*, attached to the weight arms by

*Halsey—Slide Valve Gears.

means of clips *d*—which may be slipped along and fastened at any point on these arms—and also attached to the rim of the wheel by screws, by which the initial tension of the springs may be adjusted. The springs, *P*, are only placed on the later forms of governor, and are used only during about the first half of the motion of the weights to assist their centrifugal action, for it is claimed the springs, *F*, are too powerful for one-half the range of the weights, and just powerful enough for isochronism during the outer half of their range.

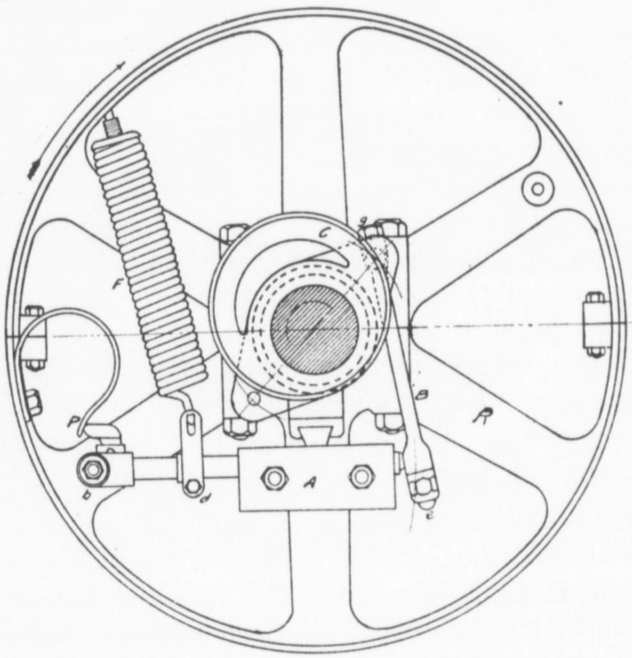

FIG. 21.

It should be noted that this governor only changes the angular advance of the eccentric, and therefore the cut-off in the cylinder. There are in this case two valves, one for cut-off only and one exhaust valve, the former of which is the one operated by the eccentric attached to the governor.

Only one-half of the governor is shown in the Figure.

The Straight Line Engine Governor is shown in Figure 22, and is the invention of Professor John E. Sweet. In connection with its priority Halsey[1] says: "It is believed that the first engine to embody

[1] Slide Valve Gears, page 68.

the above features (multiple ported valves) in connection with the shifting eccentric, was the Straight Line; and hence that engine is entitled to be recognized as the progenitor of a large and vigorous family. So far as known these features were first combined in an engine built in the Cornell University shops, and exhibited at the Centennial Exhibition." This was in 1875, three years before the Buckeye patents, and is still in use. The present form of governor is shown in the Figure, and as the valve is partially balanced the mechanism does not need to be powerful. *W* is the flyweight—which produces the centrifugal moment—and is cast to an arm having

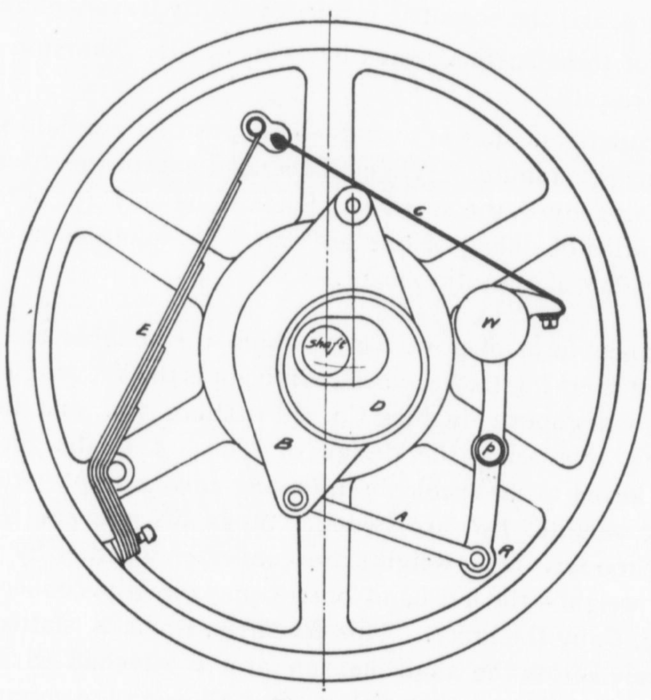

FIG. 22.

pivots at *p* and *R*. The pivot *p* about which the weight arm turns is cast to the pulley. A plate *B*, on which is cast the eccentric sheave *D*, is also pivoted to the flywheel at *S*, about which it is free to swing in a plane normal to the axis of revolution; a slotted hole cut through the casting and eccentric sheave allows freedom of movement past the shaft.

The plate *B* is connected by means of a link, *A*, to the pivot, *R*, of the arm of the weight *W*. The centrifugal force of *W* is resisted by a leaf spring, *E*, fastened to the rim of the wheel as shown and connected by means of a flexible metallic band, *C*, to the flyweight, *W*. The object of using this flexible band is to "reduce the number of parts, avoid friction, and relieve the central pivot, *p*, of a large portion of its strain."

The action of the governor is very simple. An increase in speed causes the weight *W* to fly out, moving *R* to the left, and giving *B* a slight clockwise rotation about the pivot *S* which will bring the centre of the eccentric more nearly into line with the pivot, *S*, and the axis of rotation, and thus shorten the throw of the eccentric and consequent travel of the valve. The governor will have a gravity disturbance, and the eccentric centre evidently travels in an arc.

One of these engines tested by Prof. R. H. Thurston gave the following result*:

Indicated horse-power at full load, 25, with a normal speed of 230 revolutions per minute. When all the load except the frictional one of 5% was removed, the speed was found to be only 231 revolutions, a total variation of only $\frac{2}{5}$ of one per cent. No information is given as to the rapidity of the adjustment.

A typical form of governor in which the weights are so distributed that their inertia becomes prominent is the Westinghouse made by the Westinghouse Machine Co. of Pittsburg, Pa., and illustrated at Figure 23, this being the design of 1890. *A* is the rotating disk which is keyed to the crank shaft, having pivots at *bb* on which the actuating weights, *BB*, are swung. Stops *SS* prevent the weights swinging too far. The weights are connected together by the link *e*, only one weight—the left hand one—being directly connected to the eccentric, *C*, by the link *f*. This eccentric, which is slotted so as to move freely across the shaft has an arm *c* attached to it which is pivoted at *d* to the rotating disk *A*, thus allowing the eccentric centre to swing through an arc. Spiral springs *DD*, pivotally connected between the disk and weights, resist the action of the force of the weights.

The action of the governor will be readily understood from the explanation given in connection with the preceding one. The two weights neutralize one another as far as gravity is concerned, but assist in moving the eccentric.

*Trans. Inst. C E. Vol. cxx., page 214.

The builders give a degree of regulation of two per cent., but in a more recent form of governor built similarly to the above and then entirely encased in oil, a maximum variation of speed of one-half of one per cent., with a regulation in one second, and it has been stated in a test made not long ago on a 500 I.H.P. engine there was not one revolution difference in speed between no load and full load, although the normal speed was 220 revolutions per minute.

A governor invented by Mr. Jesse M. Smith, of Detroit, and

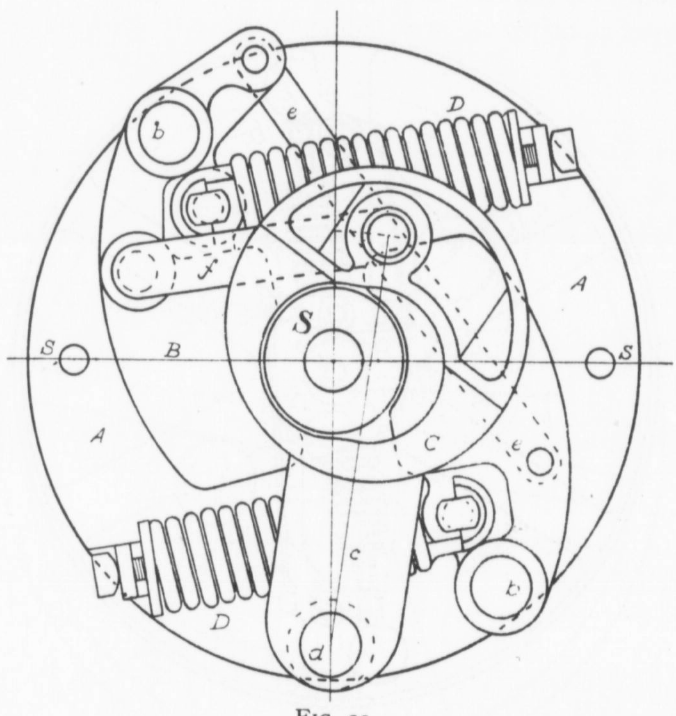

Fig. 23.

illustrated in Figure 24, possesses some novel features having all knife edge bearings, which are therefore frictionless, and also having a weight acting against the main fly weight.

Referring to the figure, *W* is the actuating weight, and has an arm *a* cast on it which has three pivotal points as shown. The arm is suspended from the arm of the pulley by a knife edge *P* about which it may swing, while its extreme end is attached by another knife-edged point *b* to the end of the spiral spring *S*, which is always in a partial state of compression. As before stated a weight *E* is attached to the end of the spring *S*. Motion is communicated to an eccentric pin at *D* by means of a lever *dd*, pivoted at *B*, to which this eccentric pin is attached; the lever *dd* is attached through the rod *C* to the weight arm *a*.

The action of the governor will, no doubt, be readily understood, but it is some question as to whether there are really any great points of superiority in such a design, for a little friction cannot be detrimental, but would seem to be rather beneficial, as it produces a certain amount of stability. The exact value of the weight *E* is questionable, and is said by the designer to replace the usual custom of giving an

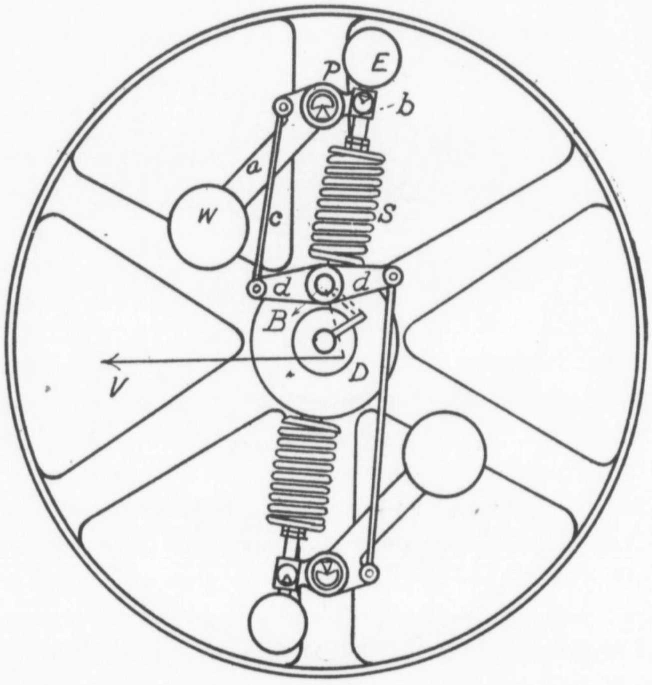

Fig. 24.

nitial tension to the springs, although this does not seem reasonable nor advantageous.

Some discussion on the governor is found in paper CCCCIX, Volume XI., Transactions of the American Society of Mechanical Engineers.

I wish to conclude this paper by giving a sketch of the excellent work done by Mr. Frank H. Ball, one of the originators of both the Ball Engine Co., of Erie, Pa., and the Ball & Wood Co., of New

York, and is now with the American Engine Co., of Bound Brook, New Jersey. His work in the governor line cannot be too highly spoken of. He has spent years experimenting on and improving his own designs, until to-day he has one of the best, if not the best, shaft governors built. His contributions to the literature of the subject form no small part of that now extant.

Mr. Ball's original governor seems to have been brought out in 1878, and has been called by him a "weigh the load" governor, on account of the dynamometric action employed. The governor is shown in Figure 25 and consists of a loose pulley *P* connected by spiral springs *E* to the double-ended arm *A*, which is keyed to the

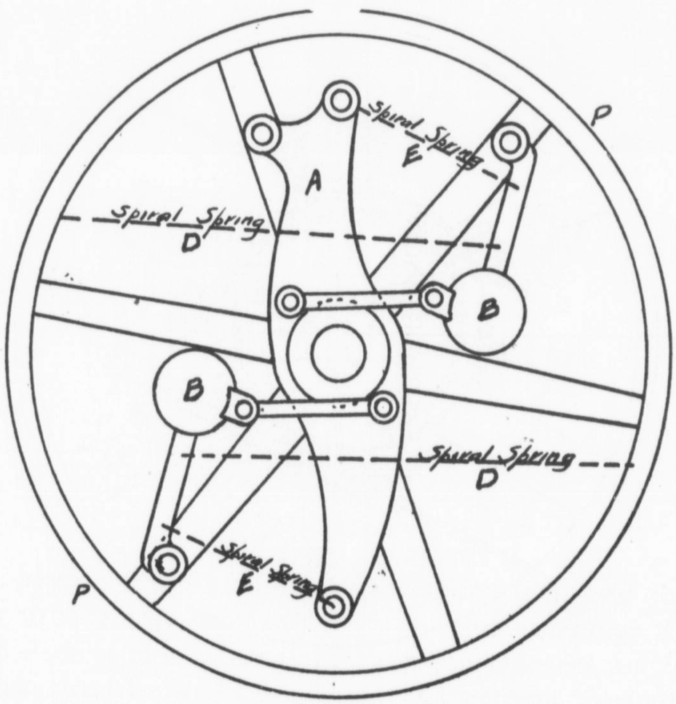

FIG. 25.

shaft. The spiral springs *E* are not connected directly to the pulley, but are joined to the arms of the weights *BB* near the pivotal points of the latter. Other springs *D* are also connected between the weight arms and the pulley, and the eccentric is operated by means of the two links shown pivoted to the two balls *BB*.

The pulley is thus connected to the shaft through spiral springs, so that the device should adjust itself rapidly for changes of load, and with carefully adjusted springs very close regulation may be secured. I am sorry I have no data at hand in regard to the matter, but I have been informed on reliable authority that the governor held the speed remarkably constant, and adjusted itself with a rapidity which has seldom been surpassed. It was given up on account of the difficulty, owing to its construction, of making repairs.

In the year 1887, Mr. Ball then patented and placed on the market the governor shown in Figure 26, and which will be seen to resemble, in a general way, a great number of other governors, and will doubtless be understood without any additional explanation.

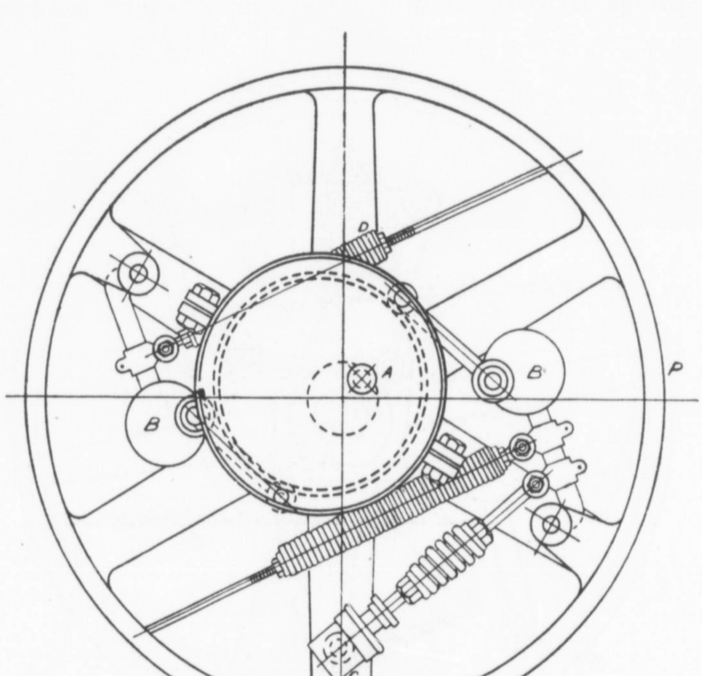

FIG. 26.

The eccentric pin A is set in the disk, which turns on another disk on the pulley set concentrically with the shaft centre. It will be readily seen that a revolution of the disk to which A is attached about the other one causes the pin A to advance towards or recede from the centre of the shaft and thus vary the travel of the valve.

The feature of this governor consists in the use of the dash-pot C and the spiral supplemental spring S for the purposes of securing stability. The dash-pot shown in Figure 27 merely consists of a cylinder filled with oil in which is fitted a piston with a small hole in it, through which the oil may slowly pass. If, therefore, tension be put upon the piston rod of the dash-pot, the piston will slowly advance toward the stuffing box end of it and towards the opposite end for compression in the rod.

The following extract from the evidence of Dr. Thurston of Cornell University in the case of the Ball Engine Co. *vs.* A. L. Ide, explains fully the points to be gained by using this device. He says, "In the usual construction of the governor of the steam engine of the class known as the centrifugal governor it has been sought to make the device efficient by giving it the property technically known as isochronism, * * * * * . It has, however, always been found

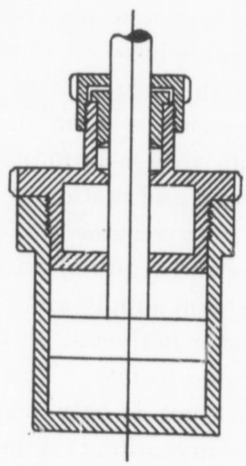

FIG. 27.

that while possible in certain forms of governors, including that here alluded to, to secure isochronism * * * this advantage is invariably more than counterbalanced by the inconveniences and dangers arising from irregular and often rapid and extreme fluctuations of speed about the correct mean speed occurring momentarily and often continuously, so that often, although the engine may make the same number of revolutions in each minute or each hour, these momentary changes may cause more trouble than the permanent but constant difficulties due to imperfect isochronism. For this reason it has been customary to sacrifice a certain amount of isochronism so as to adjust the governor to secure more stability, while losing some governing power and theoretically more perfect regulation."

"In the truly isochronous governor of the form here described, the condition of isochronism is that the action of the springs shall be variable precisely in proportion to the variation of the action of the centrifugal force affecting the balls. To secure steadiness these springs are usually given less tension at the normal position of the governor balls when the engine is at rest than would be required for isochronism."

"The device of the inventor here referred to, permits of the adjustment of the springs in such manner as to give isochronism to the governor, by adding an auxiliary spring to control the unsteadiness anticipated as a consequence of this exact equilibrium between the opposing forces of gravity and spring tension. But this auxiliary spring to be effective and yet not disturb the isochronism of the governor, must act only at the time of movement of the balls, and not when they have taken their new position of equilibrium. For this reason it is given a movable base or point of support instead of being rigidly connected to the structure of which it forms a part. The movable support is given the form of a dash-pot, since that apparatus allows the spring to act promptly and strongly on the instant of the change of speed and motion of the governor; while at the same time permitting it to resume its freedom and isochronal property as soon as the new and required position of equilibrium and consequent restoration of steam supply and engine speed is reached." He also says that the governor is thus at all times isochronal without the unsteadiness found in other forms. The auxiliary springs is the main element, and the dash-pot is only used to provide a yielding base for this spring. "The invention is I believe new, and it is certainly as important as it is ingenious."

The rapidity of adjustment was found by Prof. Jacobus by fastening a pencil in the centre of the pin *A*, and holding a card so that a spiral would be described as the pin moved inwards. He found the entire range of the governor to be covered in about one second. Such a result is excellent and the governor is yet built as at first designed.

The latest governor of Mr. Ball is certainly a magnificent piece of mechanism, both in point of simplicity and efficiency. It resembles, considerably, the design shown in Figure 28, although what is shown here is that of Mr. Robert Angus, superintendent for E. Leonard & Sons, London, Ont., the design having been considerably altered and experiments made by Mr. Angus in order to facilitate its use in their engine.

In this governor the two manifestations of inertia previously mentioned in this paper have been brought prominently out, special attention being given to the angular accelerating force as shown by the disposition of the weight about its centre of gravity.

The actuating weight CC is pivoted to the flywheel A, at a point B on one of the arms, and is prevented from swinging through too great an angle by means of the stops GG on the arms of A. K is the eccentric, having an arm cast to it so that it may swing about the pivot E in the pulley, a slot being cut in K so that it will clear the crank shaft D. On the actuating weight is an arm BF, having a pin at F

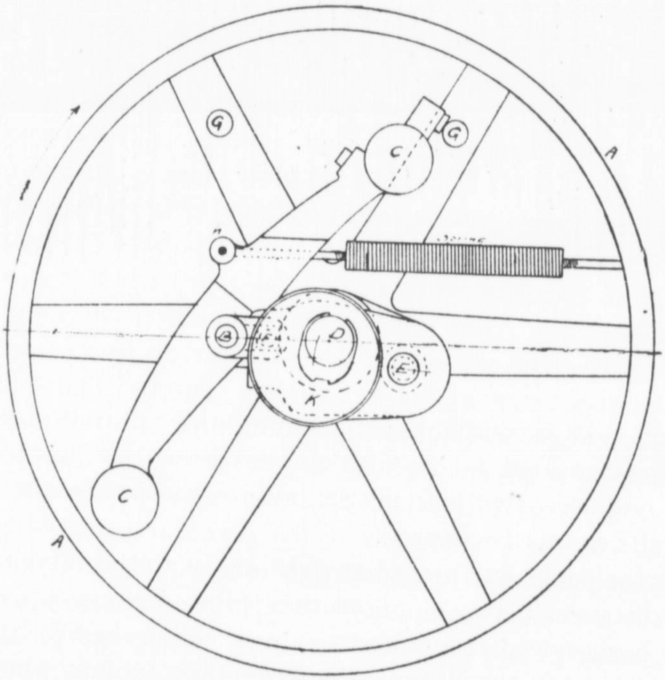

FIG. 28.

which slides in a slot on the back of the eccentric. As the weight changes its angular position, moving in a contrari-clockwise sense about B, due to an increased speed of rotation, the point F rises, causing the centre of the eccentric to approach a line from B to D, thus shortening the throw of the eccentric and decreasing the cut-off. The angular motion of the weight is resisted by a spiral spring pivoted at h to CC.

In order to show the rapidity of adjustment of this governor,

Figure 29 was drawn by fixing a pencil to the valve rod and moving a paper at right angles to its motion. The valve travel at any time is the distance vertically between two lines drawn touching the points. It will be seen that the governor adjusted itself in one revolution when the load varies from zero to a maximum, and that the mechanism is perfectly stable in this position. Since the engine was

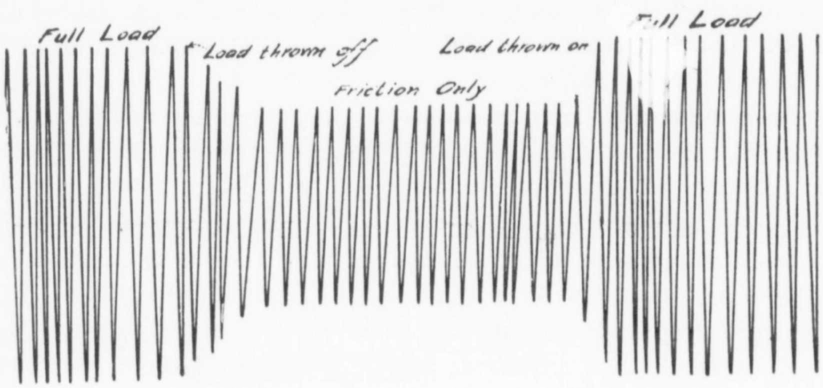

FIG. 29.

running at 300 revolutions per minute, the entire range of the governor was covered in one-fifth of a second, while no variation in the total number of revolutions per minute was noticeable. Surely better results cannot be desired.

Since the paper was moved at right angles to the valve travel by hand the distances between consecutive points are not equal. Had the paper been moved at a uniform rate it could readily have been seen by measuring the distance between these points whether the speed of the engine was constant, but this was found to be true by other means in this case.

The simplicity of construction of this latter class of governors, as well as the efficient results given by them, cannot fail to make them displace all others.

A DISCUSSION OF THE CONSIDERATIONS AFFECTING THE COST OF ELECTRIC POWER.

By H. P. Elliott, B.A. Sc.

The great problem before the Central Station manager of to-day is how to decrease the cost of production and distribution of electric power. Figures appear in various newspapers and electrical journals from time to time, either pretending to show phenomenal results from plants now in existence, or that in some particular case under consideration the power could be produced at a figure very much lower than is afterwards found to be the case. These figures are very often entirely misleading, either because they are intended to be so or that some of the items which should be included in the cost of production or distribution are left out altogether. A question that is being agitated now in one place and now in another, is the municipal control of street lighting, and figures are brought forward to show that plants controlled by municipalities supply arc lights at phenomenally low rates. Upon investigation, however, it is generally found that some items such as interest, depreciation and taxes are not accounted for at all, and that part of the wages and various running expenses are charged to other departments. The peculiar conditions under which electric power has to be supplied, make the cost of production much higher than would at first sight seem possible, and the effect of various things upon this cost is often greatly underestimated. At a convention of Central Station managers some time ago a pamphlet was circulated which contained authentic figures from a large number of plants throughout the States. Some of these were controlled by the municipalities, and for these the pamphlets gave the actual cost of production, including expenses not actually incurred, such as taxes, but which should properly be charged up to lighting. The other plants on the list were operated by companies, and for these was given the amount charged.

The actual costs of production as given in this list do not appear to be on the average very much lower than those charged by the lighting companies, and these authentic figures seem to show that the figures usually quoted must be, as said before, either intentionally misleading or compiled without a proper knowledge of the circumstances.

The object of the present paper is to study the various items that go to make up the total cost, to see of what relative importance these may become, and to point out the effect of different styles of engines, difference in the cost of coal, and difference in the method of supply upon the total cost.

In order to deal with a question of this kind the writer has considered it best to take at first a particular case, and thoroughly analyze it. Then we can generalize, and see how far the results obtained for this particular case would indicate what to expect.

The following has been taken as the basis of calculation. A plant consisting of a 500 h.p. (net) engine, boilers, dynamo, etc., is to be used for the purpose of supplying incandescent lights in a city of from forty to fifty thousand inhabitants. The problem is to find the cost per k.w. hour delivered at the switch board, under conditions as to cost and efficiency of apparatus, etc., representing as nearly as possible the present practice. Calculation will be made on four different styles of engines, two prices for coal, and under each of the following conditions :

1. The plant runs fully loaded for 365 days in a year, and 20 hours a day.
2. The plant is used only ten hours per day, and 308 days in a year.
3. A variable load has to be carried all day (24 hrs.) and 365 days in the year.

In estimating the cost of power for lighting or any other purposes there are a great many items to be considered. These are not all of a definite nature, and in a great many cases much judgment is required in order to say how much should be charged to each. The expenses of a station supplying light and power to a city will generally be included under the following heads :

1. Fuel.—The cost of coal is generally estimated on the coal as laid down in front of the boilers. This includes the cost at the mines, freight duty, and cost of all labor in handling the coal until it reaches the boiler room.

2. Renewals.—This includes the cost of all new parts of engines, boilers, dynamos, and other apparatus, incandescent lamps, etc.

3. Repairs.—This is an item which varies greatly in amount in places of the same output. In estimating the probable cost of repairs the condition of the plant must be taken into consideration. The best way to make such an estimation is to take each part separately and calculate the probable cost of repairs by comparison with the known cost of repairs on apparatus of a similar nature.

4. Supplies and Stores.—This includes the cost of water, oil waste, carbons for the lamps, dynamo brushes, and a hundred and one items of a similar nature not included under any other head.

5. Salaries and Wages.—Under this head is included all money paid to all employees of the company from the manager and superintendent down to the man who wipes the machinery.

6. Insurance.—This includes boiler insurance and inspection, fire insurance and accident insurance. The amount can be estimated as a certain percentage on the capital invested, or on part of the capital.

7. Taxes, or Rent and Taxes.

8. Interest.—When a man puts capital into a stock company formed for the purpose of carrying on a business such as electric lighting he expects to receive a much larger return for his money than if he puts it into some well established investment. For this reason, in estimating the cost of electric power, from 7 to 10 per cent. of the capital stock is charged as interest or dividend.

9. Depreciation.—The value of the plant decreases as time goes on. Apparatus wears out in spite of repairs and renewals, and gradually becomes out of date. In time the old plant can be sold for only a small fraction of its original cost, and if no provision were made for this depreciation, as it is called, a false profit would be shown or the cost would appear to be less than it really is. There are different methods of making allowance for depreciation. In most cases a certain amount is written off at the end of every year and put into a reserve fund before a dividend is declared. This reserve fund can be drawn upon instead of calling in more capital, so that no false profits are shown. In order to show how large an item depreciation may be, the following figures taken from the *Electrical World*, July 11, 1896, are given:—Wabash, Indiana, paid $18,000 for a plant, and after six years sold it for $3,000. Xenia, O., paid $35,000, and in eight years sold it for $10,000. Moline, Ill., invested $15,000 in a plant, used it for four years and then sold it for $8,000. It cost

Oxford, O., $25,000 to install a plant a few years ago, and now, on account of the cheaper methods of manufacture, a better plant could be built for $12,000. The rate allowed for depreciation will be very small or nothing, if by means of renewals and repairs the apparatus is kept thoroughly efficient and up to date. Different rates are allowed for buildings, boilers, engines, etc., and though no definite figures can be given, the following will indicate to some extent the usual practice in per cent. of capital stock :

	Per cent.		Per cent.
Buildings	1	to	2
Turbines	7	"	9
Boilers	8	"	10
Dynamos and Engines (belted)	5	"	10
Belts	25	"	30
Large Slow Speed Engines	4	"	6
Large Direct Driven Dynamos	4	"	8
Stationary Transformers	5	"	6
Accumulators in Central Stations	9	"	11
Trolley Line	4	"	8
Feeder Cables	3	"	5
Boilers and Engines	6	"	10.

The General Electric Co. allows a depreciation of about three per cent. per annum on ordinary alternating and direct current lighting apparatus, and about six per cent. on wood arc lighting apparatus, including lamps.

The plant upon which we will base our calculations is one that would be large enough to supply the incandescent lights of a city of from forty to fifty thousand inhabitants. That is if gas is used also by part of the residents.

Four different kinds of engines will be used in order to see how the total cost of production is affected by the original cost and efficiency of the apparatus. Calculations will in each case be based upon good modern water-tube boilers. The size of engine will be such that it is able to supply 500 H. P. net to the dynamo belt. The dynamo in the case of the high speed engine is belted directly to it, but with the slow speed engine a jack shaft is used. This requires that the slow speed engine should have a slightly higher I.H.P. The dynamo is a direct current multipolar and is assumed to have a commercial efficiency of 93·8 % at full load. The General Electric Company claims an efficiency of ninety-four per cent. on both alternating and direct current apparatus of this size. Cost of buildings will be based upon a structure having plain, white brick walls and a slate roof. The boilers are in a separate room from that containing the engine and dynamo.

The amount allowed for cost of dynamo includes plain skeleton switch boards with the necessary apparatus.

The result of the calculations are shown in table No. 1:

COLUMNS 1, 2, 3, 4, 5, 6, 7, 8, are taken from a paper by Dr. Chas. Emery, a prominent consulting engineer of New York, read before the American Institution of Electrical Engineers. Dr. Emery chose these figures to represent good modern practice. The cost of apparatus was determined by sending circulars to the various engine and boiler makers throughout the States. An evaporation of 8·5 pounds of water per pound of coal is assumed.

COLUMN 9.—The cost of a 350 K.W. multipolar direct current dynamo is estimated at $8,750.00, which is $17.50 per net H.P. This includes plain skeleton switch-board and all necessary station apparatus. The estimation is made from figures supplied by the Canadian General Electric Co. As a matter of fact this company does not build 350 K.W. machines of this type and recommend that a 300 K.W. machine should be used with a 500 H.P. engine. In order to better satisfy the conditions of this calculation however, a 350 K.W. machine is assumed.

COLUMN 10.—The cost of buildings and chimney is estimated by the writer. The building is plain white brick with slate roof. It is assumed that there are no expensive foundations to prepare.

COLUMN 13.—The total capital invested being determined by adding 7, 8, 9, 10 and 11, and allowing 8½% for loss of interest during construction and for inspection, ten per cent. of it is taken as dividend or interest.

COLUMNS 14, 15, 16 and 17.—These figures are determined by allowing certain rates of depreciation on the different parts of the plant. These rates are marked at the heads of the columns.

COLUMNS 18 and 19.—The supplies and repairs as far as the engine and boilers are concerned are taken from the figures of Dr. Emery. The supplies and repairs to the dynamo include brushes, oil, waste, etc., are estimated from the writer and added to those given for the engine and boilers.

COLUMNS 20 and 21.—Cost of wages is estimated by assuming that besides the labor required for the steam plant, an electrician will have to be employed at the rate of $2 per day when the plant runs ten hours, and two extra men will have to be employed at the rate of $2 per day each when the plant runs for twenty hours.

Column 22.—This includes the insurance, taxes, and renewals. Renewals are taken at 3·3 per cent. of the capital invested as given in column 11.

Columns 25, 26, 27, 28, are obtained by adding all the expenses calculated above, and show the total cost of production per net H.P. at the shaft.

Columns 29, 30, 31, 32, contain the cost per 1,000 watt hours delivered at the switch-board, and were calculated on the assumption that there is a 4 per cent. loss in the belt transmission. This leaves 480 H.P. at the dynamo pulley. Assume that the dynamo efficiency is still 93·8 per cent. and 336 kilowatt are delivered, which is 672 watts per effective H.P. at the engine and countershaft. This gives

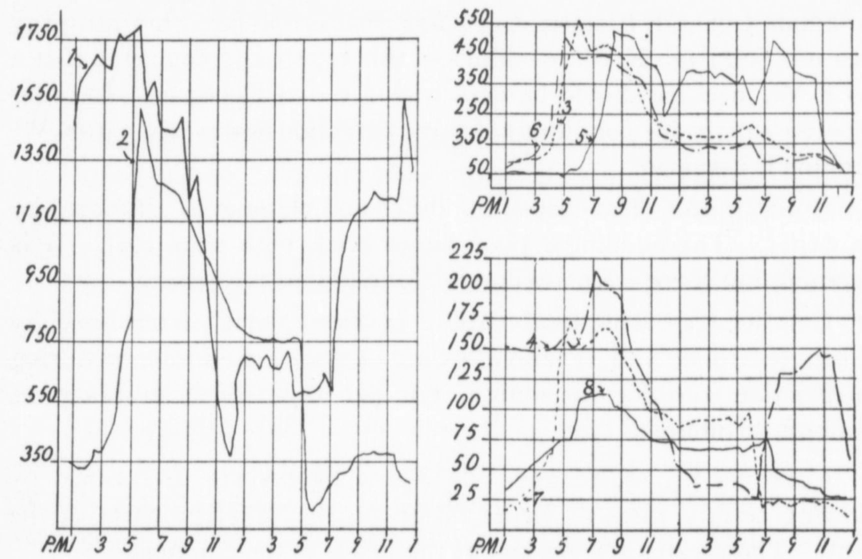

Plate I.

2069.76 K.W. hours per H.P. per year of 308 days, and 4905.6 K.W. hours per H.P. per year of 365 days. Using these results the figures given in column 25, 26, 27 and 28 are obtained.

Columns 33 to 42 contain the above result in a percentage form which allows a comparison to be more easily made.

The next thing is to calculate the cost on the assumption that the plant has to supply a variable amount of power 24 hours a day and 365 days in the year. In order to do this a load line must be assumed and a calculation made to find the efficiency at various loads. By this means we can arrive at the cost of coal under these new conditions.

Curves Nos. 1 to 12, Plate I., are load lines belonging to various power houses throughout the States, and are taken from the Report of the Committee on Data to the National Electric Light Association, in May, 1896. Of these curves Nos. 3 and 6 have the characteristics generally shown by load lines belonging to stations supplying incandescent lights for inside illumination. There is a comparatively light load up to from 4 p.m. to 6 p.m., which then increases rapidly and

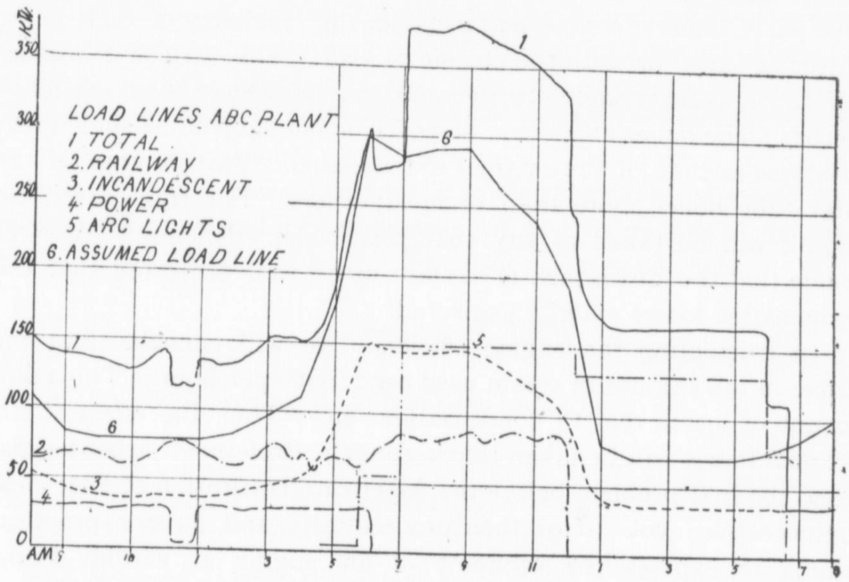

PLATE II.

remains high until from 11 p.m. to 2 a.m., with a peak generally between 6 p.m. and 9 p.m. In winter when the mornings are dark the load increases again at about 5 a.m. or 6 a.m., and falls off at 9 or 10 a.m.

The load factors belonging to these curves are given in table No. 2:

Curves No. 1 to 5, plate 2, were plotted by the writer from a plant supplying light and power in a city of about forty thousand inhabitants. The curve representing power supplied to the street railway is, of course, only an average, as the actual load has an extremely rapid variation. The incandescent load line has the same general characteristics as the others shown, and a curve No. 6, Plate 2, is constituted on the model of this one to represent the load upon the plant used as our basis of calculation. It will be found that this load line is so constructed as to show a total output per day of 3,360 K.W. hours, which is the same as if the plant was running at full load for ten hours a day.

Before going further it will be necessary to determine the efficiency curve for the plant. When referring to column 50, in table No 1, it is seen that with a Compound Condensing Slow Speed Engine, the output at full load is 285·4 K.W. per pound of coal. The calculation will be made for this engine alone. The plant may be divided into three parts, boilers, engine and dynamo, the efficiency of each being subject to variation with the change of load. The various transmissions, such as from engine to dynamo, may also have their efficiency altered. The boiler efficiency, however, is not subject to such great variations as that of the engine and dynamo. This is true of the water tube boiler more than of most others. For this reason no account will be taken of any change in boiler efficiency. We will assume that the steam pipe is so short or so well protected that the condensation losses may be neglected.

In calculating the engine efficiency we will consider first the change in the amount of steam used per I.H.P. per hour. This takes account of losses due to condensation, etc. Then the effect of the frictional losses will be taken into account. Professor Carpenter discussed the first problem before the American Institution of Electrical Engineers (see vol. 10 of their proceedings), and gave a practical formula to express the economy of an engine at various loads. Although this appears to be too general a problem to be discussed in this way, the manner of fixing the constants according to the engine and the conditions under which it is working, makes the formula appear much more reasonable. Moreover, in his discussion, Professor Carpenter gives results obtained experimentally from a number of engines, which agree very well with those obtained by calculation. At any rate this is probably the best method available for attacking this problem. As a basis, the amount of steam used per I.H.P. by a perfect engine is calculated. Not an engine working in a Carnot's cycle, but one in which the heat available is the latent heat of the steam, less the heat of the liquid for the temperature of the exhaust. The formula is really a modified form of Thurston's proposition that the wastes vary as the square root of the number of expansions and is based upon the square roots of the variation in the load.

Let m = I.H.P. of the engine at its most economical load.

" w = steam actually used per I.H.P. by the engine when working at m I.H.P.

Let p = steam required by the perfect engine under the same conditions.

Let f = least waste of the engine = $(w-p)$.

" n = I.H.P. for which the steam consumption is required.

" y = the steam consumption when working at n H.P.

$$\text{then } y = p + f\sqrt{\frac{m}{n}} \text{ when } n \text{ is less than } m.$$

Engine No. 4 is assumed to be working under a steam gauge pressure of 110 pounds per square inch, which is 114·7 pounds per square inch absolute, and the amount of coal used was calculated on a basis of eighteen pounds of feed water per I.H.P. per hour. The back pressure is taken as two pounds per square inch. On looking up the table calculated for the perfect engines, it is seen that at an absolute steam pressure of 124·7 and a back pressure of 2, 8·62 pounds of dry steam are used per I.H.P. per hour. The formula now becomes

$$y = 8{\cdot}62 + 9{\cdot}38\sqrt{\frac{556}{n}}$$

From this the following table has been calculated:

Indicated Horse Power.	Pounds of Steam per I.H.P. per hour.
556	18.000
500	18.511
450	19.044
400	19 679
350	20.440
300	21.389
250	22.608
200	24.260
100	30.737

The loss in friction is often assumed to be constant for the same speed no matter what the load. Pambour suggested that the friction could be divided into two parts, one of which is constant and the other varies with the load. The variable part will generally be small compared to that which is constant. Professor Thurston says that as a result of a large number of experiments he concludes that for a non-condensing engine the friction is independent of the load and can be determined by indicating the engine when running unloaded. In the following calculation Pambour's suggestion will be adopted. We have 56 H.P. for the friction loss at full load, and of this 40 H.P. will be considered constant. The remaining 16 H.P. is a function of the I.H.P., and is assumed to vary directly with it. The following results are obtained:

Indicated Horse Power.	Effective Horse Power.
556	500
500	445·7
450	397·2
400	348.6
350	300.0
300	251.4
250	202.8
200	154·3
100	57·2

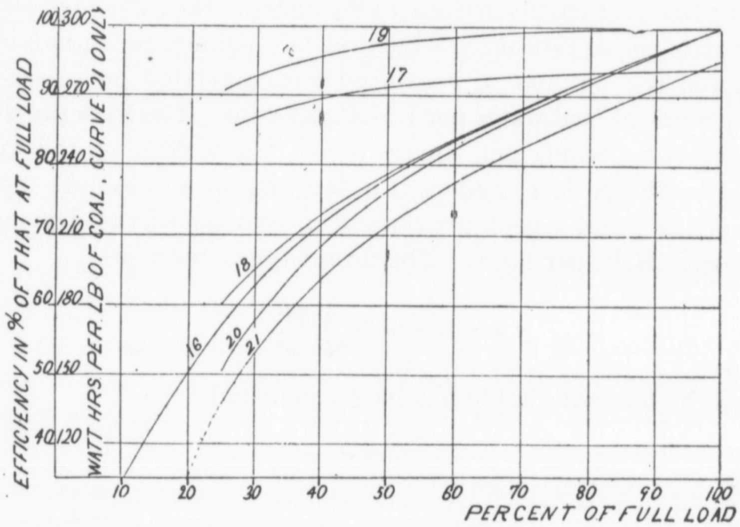

Plate III.

Having the steam per I.H.P., and the E.H.P. at various loads, the amount of steam used per E.H.P. per hour is determined and used as a basis for comparing the efficiency as the load varies. The results of this calculation are given in table No. 3. Taking 100 as the efficiency at full load, the other efficiencies are put into a percentage form. The results are plotted in curve No. 16. The efficiency curve for the dynamo is given No. 17. It is plotted from information furnished by Canadian General Electric Co. Our object is to find the combined efficiency of engine and dynamo, but the efficiency curve of the engine has for its abscissa distances representing loads upon the countershaft pulley, while that for the dynamo has distances representing the output of the dynamo. We must get a curve showing the engine efficiency at various outputs of the dynamo. This is given No. 18.

No. 19 is the curve of dynamo efficiency. No. 20 is the combination of Nos. 18 and 19. Curve No. 21 shows the output per pound of coal as the load decreases.

Now taking an output for twenty-four hours according to load line No. 6, pl. 2, and using the efficiency curve, No. 20, the result is found to be that the average output per pound of coal is reduced to 62·5 % of what it would be if the plant were running fully loaded. From this we are enabled to get the new cost of coal per H.P. per year, and thence the cost of coal per K.W. hour. We can now find the total cost per K.W. hour under the new conditions. As the plant now runs twenty-four hours a day, we may assume that the depreciation, supplies and repairs, wages and insurance, are at least as great as when it runs twenty hours per day. The capital invested is the same, so that the same amount must be allowed for interest. No calculation has been made in regard to the change in efficiency of Engines 1, 2, and 3, but for the sake of comparison let us assume that the average efficiency of the plant is reduced to $62\frac{1}{2}$ % when these are used.

We have the total of all expenses per E.H.P. per year, except cost of coal, given in column 24, table No. 1, and in order to find the cost per K.W. hour, we can proceed in this way:

3,360 K.W. hours per day for 365 days = 1,226,400 K.W. hours.

This is = 2,452·8 K.W. hours per H.P. per year.

Therefore figures in column 24, divided by 2,452·8, give the total of all expenses but coal per K.W. hour. The results are given in column 33. The cost of coal under these conditions can be obtained from columns 5 and 6. Under the conditions for which these were obtained, 4,905·6 K.W. hours were produced per H.P. per year. Therefore, by dividing the figures in these columns by 4,905·6, we get the cost of coal per K.W. hour at full efficiency. As the efficiency has been reduced to 62·5 % of this, we will get the cost on dividing by 3,066 instead of 4,905·6. The results thus obtained are given in columns 34 and 35. The totals are given in columns No. 36 and 37.

There is no doubt that the efficiency of electric power production is low compared to what it could be made if the power were to be supplied at a uniform rate, and modern machinery were used. The combination of a large multipolar direct driven dynamo with a good compound condensing engine, should give an efficiency as great or nearly as great as that of a pumping engine having the same output.

Comparing results obtained, however, it is apparent that even in the largest and best equipped central stations the output per pound of coal is *very much* less than that of pumping stations. The committee on data appointed by the National Electric Light Association gave the following in their report for 1896. The report included figures from 82 central stations burning coal. Four of the largest of these had an average production of 179 watt hours per pound. Forty-nine smaller ones, but having an aggregate output of 212,617 K.W. hours per day, produced per pound of coal 161 watt hours. Thirty-two still smaller stations, those averaging less than 1000 K.W. hours in a day, showed an average production of about 62 watt hours per pound of coal. Of these power houses the one numbered three in the report was equipped with modern water-tube boilers and multipolar dynamos directly connected to triple expansion condensing engines. The output per day was 26,402,256 watt hours, which is equivalent to 70,065,600,000 foot pounds. The output per pound of coal was 212 watt hours, which equals 562,679 foot pounds. The coal was a mixture of bituminous and anthracite. Now compare this with the official figures of a test upon the engines at the spring garden pumping station at Philadelphia, Pa. The total output per day is only about half that of the central station mentioned above, yet the output per pound of coal was 1,250,227 foot pounds, or 472 watt hours. In the same way following all through the list of these central stations and comparing the results shown with results from pumping plants of somewhat the same size, we find that the pumping plants have an output from two to four and five times as large as the central stations.

This surprisingly low efficiency is due directly and indirectly to the way the electric power has to be furnished. It has been shown already how, with the combination of a single engine and dynamo, the efficiency falls off as the load decreases, and we can see that in most central stations the same causes for this decrease, along with many others, must be present.

1. Waste of coal in starting and banking fires.
2. Increased percentage of steam pipe condensation.
3. Increased cylinder condensation.
4. Increased percentage of friction loss.
5. Increased percentage losses in the dynamos.

Now by using a number of units so arranged that they can all be worked together or separately or in different combinations, it may be possible to keep each fully loaded and at its maximum efficiency. We would find, however, that the multiplication of units must not be carried on to a very great extent for several reasons. First of all, small units have generally a much smaller efficiency than large ones. Also, there is so much complexity and extra friction losses introduced by an increased number of units, that even if each were kept at its maximum efficiency the total efficiency of the plant would be small. It is true that a combined engine and dynamo efficiency of eighty per cent. can easily be obtained, and that assuming 1·5 pounds of coal per I.H.P. per hour, a production of about 400 watt hours per pound of coal would be the result. Looking at column No. 50, in table No. 1, however, we find that with the engines there used the following is the production per pound of coal.

159.7	watt hours	for	engine	No.	1
177.1	"	"	"	"	2
263.5	"	"	"	"	3
285.4	"	"	"	"	4

The figures belonging to these engines were carefully chosen by Dr. Emery to represent good modern practice for units of this size, and considering the conditions under which most power houses work we can see why the production should fall even lower than this. The curves on plate 2 represent the different loads that have to be taken by a certain station, and in most places something of a similar nature will have to be encountered. That is, units of various kinds and sizes will have to be employed to suit the different kinds of work, and it would be impossible that even under most favorable conditions units of this kind could have as high an efficiency as the combination of engine No. 4, with a 350 K.W. dynamo.

The writer calculated the efficiency of production of the power for arc lights in the central station where the output is represented, plate 2. The power was furnished by two slow-speed compound condensing engines, belted to a counter shaft to which is belted the arc lighting machines. These are partly of the Wood and partly of the Thomson-Houston systems. Calling these engines Nos. 1 and 2, the former is rated at 150 H.P., the latter at 120 H.P. Diagrams were taken when the full load was on and showed that No. 1 was developing 152·12 H P. and No. 5, 117·71 H.P., or a total of 269·8 I.H.P. The output at the switchboard was 153,400 watts, so the efficiency was 76%, which is very good under these conditions. Now, even supposing that these engines developed one I.H.P. on eighteen pounds of steam per hour and the coal had an evaporation of 8·5, the production per pound of coal is only ·269 watt hours. The arc lights are the most uniform load of any that this station has to carry, but even with the load line shown the average production will probably be brought down to nearly 200. The efficiency of the whole plant is even further reduced by light and variable loads. Curves No. 1 and 2, plate 4, are plotted from the monthly report sheet of this station. One shows the daily output, and the other the output per pound of coal. These curves show very clearly how the daily output and therefore the load affects the efficiency.

Looking over all the curves supplied by the committees on data before mentioned, and taking into consideration the kinds of engines and dynamos, we cease to wonder at the low efficiency.

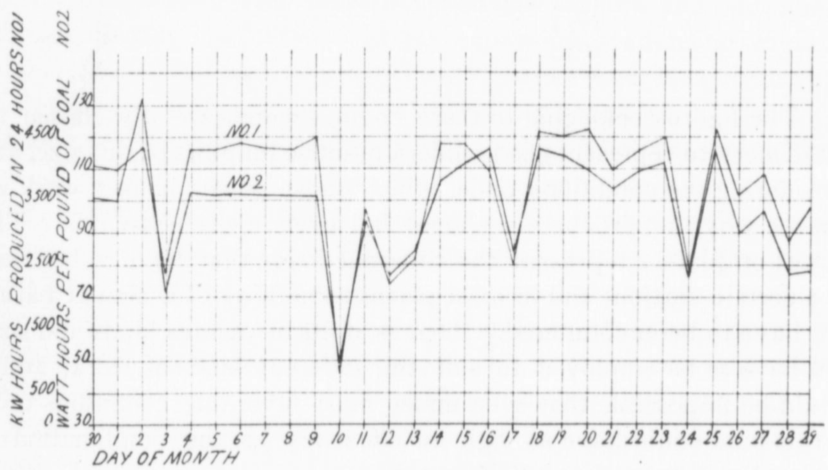

PLATE IV.

A study of the relation between the items in table No. 1, and of the effect of the variable load, will teach some important lessons. As was mentioned by Dr. Emery, it might turn out that power could be produced more cheaply with an inefficient engine than with an efficient one. He showed this in the following way: Consider the cases of two plants, such that for a given output one saves $12\frac{1}{2}\%$ of the amount of fuel used by the other, so that the relative costs of the fuel are 8 and 7. Now, if the other operating expenses in the two cases are equal, and equal the cost of fuel for the uneconomical engine, the efficiency, considering all the costs, will be in the ratio of 16 to 15. The machinery in the more economical plant is of a better type than in the other, and would cost more. If the dividend to be paid on the capital invested is 10%, it would only require a difference in the cost of such an amount that 10% of it would represent a saving of 6½% of the fuel in order that the cost of production should be the same in both cases.

On referring to table No. 1, columns 38-49, where the costs are

TABLE NO. 1

Style of Engine.	No. of Engine.	Horse Power. 1. I.H.P. of Engine.	2. Commercial H.P. of Boilers.	Cost of Coal per H.P. per Year. On basis of 308 days of 10 hrs. each. Cost of Coal per ton 3. $2.67	4. $4.46	On basis of 365 days of 20 hrs. each. Cost of Coal per ton. 5. $2.67	6. $4.46	Capital Invested per H.P. 7. Boilers erected and connected $22 to $25 per H.P.	8. Engines erected and ready for work.	9. Dynamo including switch-board, etc.	10. Buildings and Chimney.	11. Total.	12. Total + 2½% for inspection and 6% for loss of int. during constr'ct'n	13. Interest on capital 10% on col. No. 12.
				$	$	$	$	$	$	$	$	$	$	$
Simple high speed non-condensing	1	542	596	19 09	31 82	43 20	72 00	26 22	17 50	17 50	19 18	80 40	87 23	8 72
" low " "	2	556	537	17 22	28 69	38 95	64 92	23 63	25 00	17 50	18 72	84 85	92 06	9 21
Compound high speed condensing	3	542	361	11 57	19 79	26 18	43 64	15 88	24 50	17 50	14 86	72 74	78 92	7 89
" low " "	4	556	334	10 69	19 29	24 18	40 30	14 70	30 00	17 50	14 92	77 12	83 67	8 37

Style of Engine.	Depreciation on. 14. Boilers at 7%	15. Engines at 4%.	16. Dynamo at 3%.	17. Buildings at 2%	Cost per net H.P. per Year of. Supplies and probable repairs. On basis of 18. 308 days of 10 hours.	19. 365 days of 20 hours.	Wages. On basis of 20. 308 days of 10 hours.	21. 365 days of 20 hours.	22. Insurance 0·5% Taxes 1·2% Renewals 3·3%.	Total of all expenses but Coal. On basis of 23. 308 days of 10 hours.	24. 365 days of 20 hours.	Total Cost per net H.P. per Year. On basis of 308 days of 10 hours. Cost of Coal per ton. 25. $2.67	26. $4.46
	$	$	$	$	$	$	$	$	$	$	$	$	$
Simple high speed non-condensing	1 83	0 70	0 52	0 38	2 98	7 05	6 01	14 26	4 02	25 16	37 48	44 25	56 98
" low " "	1 65	1 00	0 52	0 37	2 55	6 03	5 72	13 57	4 24	25 26	36 59	42 48	53 95
Compound high speed condensing	1 11	0 98	0 52	0 30	2 98	7 05	4 86	11 52	3 64	22 28	33 01	33 85	42 07
" low " "	1 03	1 20	0 52	0 30	2 55	6 03	4 72	11 20	3 86	22 55	32 51	33 24	41 84

Style of Engine.	Total Cost per net H.P. per Year. On basis of 365 days of 20 hours. Cost of Coal per ton. 27. $2.67	28. $4.46	Total Cost per K.W. Hour at Switchboard. On basis of 308 days of 10 hours. Cost of Coal per ton. 29. $2.67	30. $4.46	On basis of 365 days of 20 hours. Cost of Coal per ton. 31. $2.67	32. $4.46	Ratios of Costs of Coal per H.P. per Year. On basis of 308 days of 10 hours. Cost of Coal per ton. 33. $2.67	34. $4.46	On basis of 365 days of 20 hours. Cost of Coal per ton. 35. $2.67	36. $4.46	Ratio of total costs except Coal. On basis of 37. 308 days of 10 hours.	38. 365 days of 20 hours.
	$	$	Cents.	Cents.	Cents.	Cents.	$	$	$	$	$	$
Simple high speed non-condensing	80 68	109 48	2·138	2·757	1·645	2·232	100 00	100 00	100 00	100 00	100 00	100 40
" low " "	75 54	101 51	2·053	2·607	1·540	2·069	90 20	90 00	90 20	90 20	97 63	100 00
Compound high speed condensing	59 19	76 65	1·636	2·032	1·207	1·562	60 60	61 90	60 60	60 60	88 07	88 46
" low " "	56 69	72 81	1·606	2·022	1·156	1·484	56 00	60 60	56 00	56 00	86 74	89 64

Style of Engine.	Ratios of Total Costs. On basis of 308 days of 10 hours. Cost of Coal per ton. 39. $2.67	40. $4.46	On basis of 365 days of 20 hours. Cost of Coal per ton. 41. $2.67	42. $4.46	Costs per K.W. Hour at Switchboard with a Variable Load. 43. Cost per K.W. hour of all expenses but Coal.	Cost per K.W. hour of Coal. Cost of Coal per ton. 44. $2.67	45. $4.46	Total cost per K.W. hour. Cost of Coal per ton. 46. $2.67	47. $4.46	Ratios of total costs per K.W. hour at Switchboard with variable load. Cost of Coal per ton. 48. $2.67	49. $4 46	50. Watt hours per lb. of Coal not counting Coal wasted.
	$	$	$	$	Cents.	Cents.	Cents.	Cents.	Cents.	$	$	$
Simple high speed non-condensing	100 00	100 00	100 00	100 00	1·528	1·409	2·348	3·018	3.876	100 00	100 00	159 7
" low " "	96 00	94 68	93 63	92 71	1·492	1·271	2·118	2·763	3.610	91 53	93 14	177 1
Compound high speed condensing	76 50	73 84	73 37	70 00	1·346	0·854	1·424	2·200	2·770	72 90	71 47	263 5
" low " "	75 12	73 44	70 28	66 50	1·326	0·789	1·314	2·115	2·640	70 09	68 12	285 4

the coal burned, the more likely will it be that the inefficient engine is the most economical when all things are considered.

of any that this station has to carry, but even with the load line shown the average production will probably be brought down to nearly 200. The efficiency of the whole plant is even further reduced by light and variable loads. Curves No. 1 and 2, plate 4, are plotted from the monthly report sheet of this station. One shows the

engine, the efficiency, considering all the costs, will be in the ratio of 16 to 15. The machinery in the more economical plant is of a better type than in the other, and would cost more. If the dividend to

THE COST OF ELECTRIC POWER. 127

be paid on the capital invested is 10%, it would only require a difference in the cost of such an amount that 10% of it would represent a saving of 6½% of the fuel in order that the cost of production should be the same in both cases.

On referring to table No. 1, columns 38-49, where the costs are
given in a percentage form, the cost of production by the least efficient

the coal burned, the more likely will it be that the inefficient engine is the most economical when all things are considered.

of any that this station has to carry, but even with the load line shown the average production will probably be brought down to nearly 200. The efficiency of the whole plant is even further reduced by light and variable loads. Curves No. 1 and 2, plate 4, are plotted from the monthly report sheet of this station. One shows the daily
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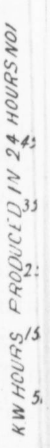

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engine, the efficiency, considering all the costs, will be in the ratio of 16 to 15. The machinery in the more economical plant is of a better type than in the other, and would cost more. If the dividend to

be paid on the capital invested is 10%, it would only require a difference in the cost of such an amount that 10% of it would represent a saving of $6\frac{1}{4}$% of the fuel in order that the cost of production should be the same in both cases.

On referring to table No. 1, columns 38-49, where the costs are given in a percentage form, the cost of production by the least efficient being taken at 100, we see that under all conditions the power is produced most economically by the most efficient engine. This is due to the fact that there is almost the same amount of capital invested in each case (column 12). The efficient engines cost the most (column 8), but they require less steam, and therefore less boiler power. The cost of boilers (column 7) will be much less for the high priced engine on this account. Moreover, with the smaller and more efficient units, less capital is required for buildings, so upon the whole the total capital invested (column 12) and therefore the interest, depreciation, etc., will be about the same in each case. It can easily be seen, however, that in some cases it would pay to put in an inefficient but cheap engine to carry a certain part of the load in a power house. Many stations have several large units which are very efficient with fair loads. For a part of the day, however, the load intended for one of these units becomes so small that there is a very great waste, and it pays to install a small engine and dynamo to carry the load for a short time. No other capital or expenditure is entailed, and in many cases an inefficient engine would be found economical when all things are considered. Besides this a power house will often supply the incandescent lights from large alternators belted to high speed engines, and use different dynamos and different engines to supply the power for motors, etc. In most cases the power required for motors, added to that required for lighting, is, in the day time, much less than that required for the lights alone at night, when the motors are not running. Here the amount of boiler horse power to be installed depends on the greatest load and the efficiency of the engines carrying it, and if the motor load is small compared to the power required for lighting, no additional boilers will have to be installed, even if we use inefficient engines for the day loads. In this case also it may pay to use inefficient engines. In any case, it can be seen that the shorter the time the engine runs, or the smaller its load factor, and the cheaper the coal burned, the more likely will it be that the inefficient engine is the most economical when all things are considered.

TABLE NO. 2.

No. of load line.	No. of watt hours per day.	Average power in watts.	Maximum power in watts.	Load factor.	Watt hours per lb. of coal.
1	26402256	1100094	1800000	61	212
2	16149010	672875	1525000	44	133
3	5401900	225079	575000	31	129
4	2683008	111792	218000	51	124
5	6794304	283090	525000	54	109
7	1818780	75782	175000	43	178
8	1444931	60205	112000	53	69
9	203555	8481	15000	56	33
10	3207392	133641	193000	69	237
12	903720	37655	111000	34	92

TABLE NO. 3.

Water per I.H.P. lbs.	I.H.P.	Effective H.P.	Lbs. of water per hour.	Effective H.P. per lb. of water.	Comparative efficiency.
18.000	556	500.0	10008.00	0.0500	100.0
19.044	450	397.2	8569.80	0.0463	92.6
19.679	400	348.6	7871.60	0.0443	88.6
20.440	350	300.0	7154.00	0.0418	83.6
21.389	300	251.4	6416.70	0.0393	78.4
22.608	250	202.9	5652.00	0.0359	71.8
24.260	200	154.3	4852.00	0.0318	63.6
30.737	100	57.2	3073.70	0.0186	37.2
18.511	500	445.7	9255.50	0.0481	96.2

TABLE NO. 4.

Per cent. of full load	Total efficiency.	K.W. produced.	K.W. produced per H.P. at shaft	H.P. supplied at dynamo.	H.P. to shaft.	Efficiency in % of that at full load.
.875	93.57	294	698.32	421.4	438.7	99.76
.750	93.30	252	696.02	362.1	377.2	99.46
.675	92.51	210	690.12	304.2	316.9	98.62
.500	91.20	168	680.35	246.8	257.1	97.23
.375	89.00	126	663.94	189.7	197.6	94.89
.250	85.00	84	634.10	132.5	138.0	90.63

GOLD MINING IN NOVA SCOTIA.

By Fred W. Christie, C.E.

Gold mining is now recognized as one of the established industries of Nova Scotia, and any considerable review of the items of scientific and industrial information, so far gathered, as relating to it, inevitably leads to the conclusion that when carried on with due regard to local conditions and distribution of pay ore, the profits are large. The gold miner occupies a distinct place and type, and belonging to an industry that dates back to 1860, he is looked upon by most people as the kind of man one ought to know. There is a pleasing harmony between the scientist and miner, and the blood-curdling story of the "Anticlinals" strikes a responsive chord in both their breasts, and their mutual enjoyment is only broken by the jarring notes of the quarrels of the promoter and mining shark. An old gold miner, tested mentally, is a combination of facts and propositions not yet reduced to definite terms, with an instinct that is attracted to, or warned away from a particular locality where gold may be looked for. This knowledge, gained by natural observation, and held in "refractory" combination, is necessary to the young engineer going into mining, and can be extracted by acquiring the ability to think in the miner's peculiar mental language.

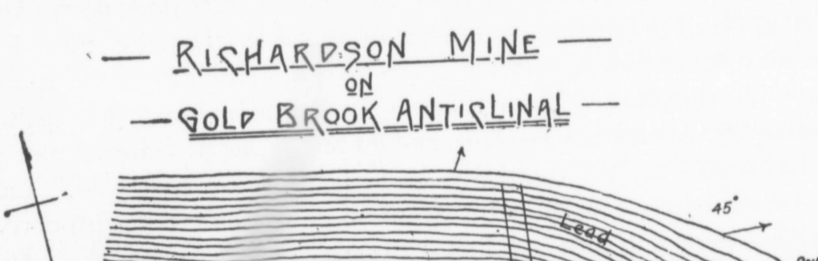
Gold mining in Nova Scotia has gone through one period of experiment, lottery and development, with the result that a large amount of mining experience and knowledge of ore deposits has been acquired by those connected with the industry. With the mining lottery promoters very largely exiled, the second period has opened with the demonstrated successes of low-grade mining, proofs of large deposits of ore, re-working of old mines, the finding of many lost pay-streaks, and the making of records for economical handling of ores, making gold mining a legitimate business, and an attractive means of investment. In spite of the prejudice toward the man with "a hole in the ground," it has gone through its apprenticeship, paying its bills and taxes without getting any exemptions or bounties. A careful observer can always notice that the farming localities and villages which lie near the Nova Scotian gold mines, have an extra appearance of stir and thrift, caused by the oftener repeated pay days, as compared with some other industries. In spite of the misadventure and waste of investment caused by promoters and mining sharks, the slavish dependence on theories, and the mistakes of men attempting to copy particular mines in other countries, when these plans did not suit their properties, legitimate and properly directed work has accomplished marked success, and in many cases the prizes have been very valuable. It is found that the success of a mine depends on properly applied knowledge of the low grade ores, when the original rich pay streak has come to a point where it breaks off and hides in some part of the rock, leaving behind some "indications" known only to the initiated, and by which it may be found.

The gold country may be considered as being in two sections: the Atlantic coast section of Nova Scotia proper, where free gold quartz is abundant, and the mountainous interior of Cape Breton, where copper and lead ores abound and free gold is the exception. The Nova Scotia section is usually estimated to have an area of 6,000 square miles, and takes in the counties of Yarmouth, Shelburne, Queen's, Lunenburg, Halifax, Guysboro, with parts of Digby, Hants and Colchester, and perhaps parts of Annapolis and King's will be prospected into this section. The Cape Breton section comprises the greater part of the mountainous interior of Inverness and Victoria counties, with some sections underlying parts of the carboniferous formation of Richmond and Cape Breton counties.

The study of the geological age of these sections has called out a divergence of opinions from authorities, particularly as to details, but these authorities are agreed as to the structure of the gold districts. Looking at the three general classes of rocks in these sections; granites (so-called), including syenites, felsites, gneiss, etc.; quartzites, sometimes called "whin," and slates; and considering the granites as the bottom of the series, and the slates as the top, authorities have placed them as Laurentian, Cambrian, or Silurian, with pieces of Devonian, according as their opinions were swayed by the evidence of the characteristics shown by the different varieties of these three general classes, or of their resemblance to the rocks of such ages in other localities.

The overlying of the varieties of the rocks is seldom clearly presented, the effect produced being often of the appearance of "contacts" or "intrusions" with attendant faulting and fissuring. A similar effect is noticed where the later carboniferous strata abut against these older measures. These "contacts" or "intrusions," on account of the almost certain occurrence with them of metalliferous deposits, suggest a relation of cause and effect that has not been crystallized in any rules. Miners have a kind of instinct that leads them to pay attention to changes in the slates and quartzites, and hunt from lead to lead to find the direction in which the gold has been deposited. The deposits of "metals" (so-called) are not confined to the quartz of the veins, but are found throughout extensive zones of the leads and measures; showing in the partings of the layers, and in the fissures or cracks, in whatever direction these fissures may run, and often in the grain of the country rock for a distance from the veins. Thus, the free gold and sulphide metals, by following the fissuring, have enriched large blocks of material alongside the rich veins as originally worked for the "free gold," and thus have greatly enlarged the possibilities of the mining property. For this reason old pits are being eagerly searched for these lower grade bodies of ore and lost pay-streaks. The study of the characteristics of "contacts" of different formations, and the system of fissuring in a district, promises important results in the workings of mines in the future. Among results that may be looked for, are well-founded and decided opinions as to the depth of profitable ore, and the right places to sink shafts for "deep mining." The subject of experimental deep shafts has often been urged upon the Government of the Province; but the great difficulty has been to lay down any rules as to "expectations" as to the depth and direction of the deposits of gold in the vein or lead-filling. The impossibility of changing the line of a shaft to follow the irregular deposits of gold that are seen in all gold-mining countries would necessitate provision being made for extra underground work. Individual leads have been mined on their dip to 800 and 900 feet, as in Caribou; but the extension of these inclines another 1,000 feet might only develop the quartz deposit to that depth, without deciding the gold bearing ability of the lead, as a whole, to the depth attained. Many desirable points as to the general structure of a district, the probable direction of the course of vein-filling, the system and kinds of fissures and beds, and the rich pieces of ground, are liable to be missed by an experimental shaft. The deep shaft question is not at all a pressing one to the ordinary mining concern, the quantity of gold-bearing ore within a depth of 500 feet being sufficient in any district to supply the largest range of business for a long period. The occurrence of regular leads, known by working to 1,000 feet, is good evidence of there being beyond that a good depth to the deposit, and is conclusive to the general mining man. The "deep" question is usually brought up by the man who worries over the likely scarcity of coal in Great Britain or Nova Scotia.

The structure of all the gold districts, and the arrangement of the rocks, follow the same general rule, being more or less a tilting and folding of the rock measures, presenting variations in each that



the wise miner requires to study with care, in order that he may have decided views as to the whereabouts of his best ore, and good judgment in selecting his working ground. Each district is called a quartzite or slate district according as the one or other is present in the larger quantity, although belts of slate, belts of quartzite and quartz lodes are all present in nearly every case. The ideal arrangement is seen when the main quartz lodes and the belts of slate and quartzite are tilted up and folded in an anticlinal giving outcrops of two or three dips of the quartz lodes, so close to each other that two or more inclines can run into one breaker by direct lines of haulage. A good example of this is shown in the Richardson mine at Gold Brook, Guysboro County, where the quartz lode has a loop with the ore dipping north, east and south, so that three shafts can hoist directly to one breaker, while their underground workings are hundreds of feet apart. In many instances, however, the anticlinal formation is only partial or merely suggested, with complications of faulting, capping or overturn. A common example is of a group of leads running roughly parallel to each other and the country rock, and dipping in the one general direction, without showing the loop or opposite dip. In some cases where the known quartz lodes all dip in the same general direction, the two sides of the anticlinal formation are suggested by the more acute angle of dip one group has when compared with some adjoining group with a flatter dip. This form is liable to cause "weak ground" when adjoining veins are worked out, and the pressures from the weight of the intervening rocks and the different dips are unequal. Some cases of "caving" are supposed to have happened from such conditions, showing that ordinary timbering was inadequate to support the walls.

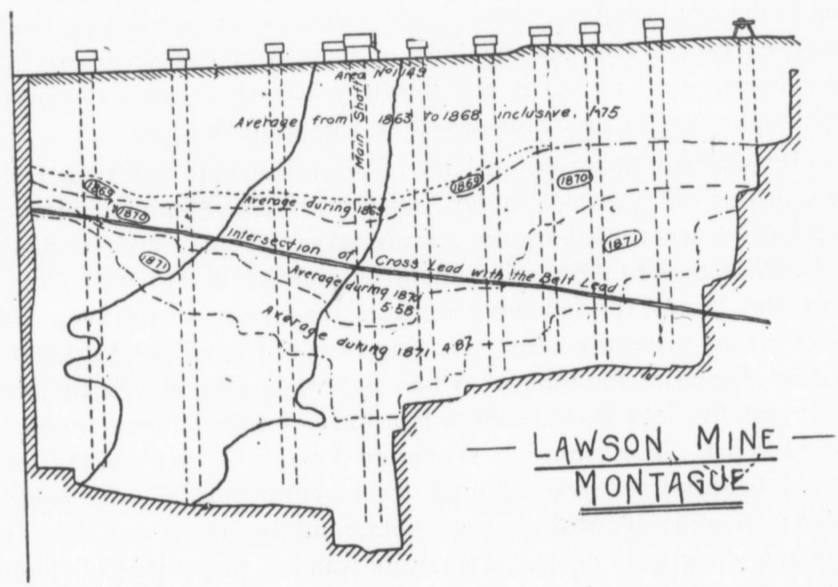
A mineral map shows the gold districts scattered irregularly about the country, and the localities have been discovered by accident either by finding the outcrops of the quartz leads, or the boulders which indicated the near locality of gold bearing quartz in lodes of some kind. When, as in the majority of cases, the boulders are the evidence of ore deposits, the prospector trenches in a northerly direction from the boulders to find the outcrop. This is done according to the accepted law of glacial drift moving always from north to south. This direction of the glacial drift is often clearly shown by the "striae" or ice scratches on the bed rock. A cross lead, of course, is found by trenching east and west and angles, usually by finding main lead belts and stripping them until the intersection of the angler is found.

Gold mining in Nova Scotia comes under the description of "quartz mining," and the different form of quartz deposits present a great variety of problems in carrying on underground work. These quartz deposits are classed by miners as "main leads" when they appear to be interstratified with the main rock measures, "cross leads" when they traverse the measures very nearly at right angles, and "anglers" when they intersect the measures and other leads at acute angles in dip and strike. The miner naturally looks in the quartz for the free gold, and so the term "lead" is applied in general usage to the quartz deposit, whether in the filling of the vein or in a bedded form.

The term "vein" is seldom used in actual work, there being usually a selection of the material to go to the mill, and in following the mill ore in the direction of the best pay ore the bulk of the stopes will, as a rule, be in the so-called "main leads" of quartz on account of their working easily from the seams or partings of the layers, and the walls of the "true" vein or fissure will be seldom used. In some cases, however, in working main leads, one fissure wall is followed to blow from, or timber to, and the other wall is seldom stripped. Cross leads and anglers exhibit in many ways the characteristics of fissure veins, and main leads often some characteristics of both fissure veins and bedded deposits. Fissures will often carry in a main lead belt for long distances, and sometimes appear to deflect considerable portions of main rock measures. In the early days of gold mining in Nova Scotia, the opinion was general that the gold belonged to the "main leads," and when seen in "cross leads" or "anglers," the idea was that these cross leads or anglers were robbing the main lead or helping in some way to make a good piece of ground. On this account the mining work was always carried on with reference to the main leads.

Of late years profitable mining has been carried on in leads classified by miners as anglers and cross leads, and in some districts the principal work has been confined to them and the main leads are neglected. The width required for tunnels and stopes is generally provided for by "making" another wall, or using some seam of slate or quartzite at a convenient distance for taking the timber and scaffolding. Main leads have the great advantage in mining work, of being enclosed in bands of quartzite and slate, parting at the seams and thus making the sinking of main shafts and the driving of levels very cheap. From this fact a shaft can be made very roomy and the stopes and levels may be left narrower and economize in the amount of material moved. Cow Bay is an example of a cross lead district, and North Brookfield of an angler district, although in both these cases the leads are parts of a regular vein with decided fissure walls. In neither of these cases, however, is the probability lost sight of that the other classes of leads may develop, carrying profitable quantities of ore. The point practically demonstrated by these instances, is, that in any given district, some or all of these classes of quartz deposits present special advantages in reaching the pay ore.

A good knowledge of the number, size and relative importance of the leads present in the veins or bedded deposits is important to the success of the business carried on. To this should be added a well founded opinion of the system of fissuring and the effect of the faulting in the leads, veins and country rock. By not making as general use of the word vein as miners in other countries, an unfounded prejudice has arisen against the quartz deposits of Nova Scotia, that they are narrow and hard to work. Although individual fissure veins are often narrow, wide belts of material may carry a great number of quartz leads. No matter how many leads there are in a belt the Nova Scotia miner has the habit of naming and describing the separate quartz leads. The frequency of bodies of gold-bearing



slate, and mixed material occurring with the rich lead, would justify the use of the term vein, and the finding of all these in belts up to 100 and 200 feet wide, would also justify the use of the term ledge, in describing a mining property. The Lawson lead mentioned below might be described as five feet wide, although the rich pay streak mined was four inches of quartz.

The development of the ideas in the preceding statements is illustrated by the change that is seen in comparing the working of a mine of the earlier seventies with the workings that are now to be seen of later design. As the rich coarse gold was formerly found in narrow main leads, this class was hunted for, and the nature and value of the other deposits alongside were overlooked. A good example is illustrated by the sketch of the Lawson mine at Montague, where the workings were carried on in a lead four to six inches in thickness, giving as high as forty ounces to the ton, and making an average of as high as five and one half ounces to the ton for the work of a season. The workings were carried down on the lead for 300 feet and along its course about 500 feet. This narrow width of quartz in this extent of workings yielded about $200,000, giving about $100,000 profit. On account of the difficulty of working such a narrow width of rock at the prices charged at the time for supplies and crushing, the work was stopped when the quantity of rich ore at the bottom decreased. Many mines worked in the earlier days on similar narrow leads can now be worked by wider work reaching across several leads, or the quarry system of working by "open cuts" makes it possible to select the quartz at a profit, and handle very large quantities of so-called black rock.

Instances or examples of some of the prices paid in former times for crushing ore, mining supplies, machinery and mining work show how hard it was to make gold mining pay. From a letter from Mr. B. C. Wilson, Esq., Superintendent of the Acadia Powder Co., at Waverley Mines, the following extracts are taken :—"Between 1860 and 1870 the price of custom crushing ranged from $8 to $4 per ton. I know of a lot of 200 tons for which $16 was paid; it was frequent to pay $8 for lots from three to ten tons. I was the first person to bring the price of crushing in Waverley down to $4 per ton. In 1862 I paid twenty-five cents a pound for blasting powder with freight added; in 1863 it was twenty cents, and by 1870 was down to sixteen and eighteen cents. Dynamite was not introduced till about 1880. The first used was Nobels, at about fifty and sixty cents a pound; just before the Acadia Powder Co. started in 1883, it sold in Waverley for $1 per pound. Fuse cost a cent a foot, getting down gradually to seventy-five cents per 100 feet, and finally to half a cent per foot. Detonators for dynamite sold for two cents each or $1.50 per 100. The retail price now is seventy-five cents per 100. Tallow candles were twelve cents a pound, steel fifteen cents a pound, while shovels were double the present price." These figures show how expensive it was to mine or to develop. The change in the cost, weight and efficiency of machinery alone would make many properties to-day paying investments. The change in the cost of machinery means a difference of at least fifty per cent. on boilers and engines. Pumps of all kinds have come down fifty per cent. These remarks apply equally well of course to any mining country and go to show that the past should be forgiven and no mining proposition be prejudiced by former misadventure when modern machinery and improvements, and years of experience, are available for the working of a property.

As the prices of work, machinery and supplies came down in cost, the amount of work done by miners in a given time and place became greater on account of local mining experience of the country; the efficiency of steam engines, mining tools and milling machinery were increased; and more leads and ore deposits came into the class that would pay to handle. Unfortunately, capital was scarce and promoters looked for the properties that had the very rich streaks of quartz. These rich streaks gave in different cases from $50,000 to $200,000 of rich free gold, and mining companies too often paid excessive dividends beyond a safe period and neglected to develop their properties for future supplies. By neglecting to put in concentrating and reduction plants, a heavy falling off in revenues came when the sulphide metals took the place of the coarse gold in the mineralization of the ore; these sulphide metals not being fit for amalgamation carried away the gold of the ore. The majority of tests showed the sulphides to be valuable gold carriers, but the great estimated expense of concentration and reduction plants frightened shareholders from paying back the required funds.

Mine managers and foremen were then often called upon to work marvels with scanty funds. One beneficial result of this penurious policy has been to drive miners to do some wonders in the way of making records for the amount of mining work done for a small amount of money, and reducing the cost of crushing. The search for free gold has led to the mining and crushing of blocks of material of all sorts that was thought to carry gold, and brought into existence a great variety of gold mining operations. One condition that was favorable to such variety was, that in all gold mining in Nova Scotia, the shafts and levels could be confined to working in the quartz ore, or stripping the ore foot by foot as the workings advanced. An occasional cross-tunnel in the country rock would connect two or more adjoining quartz leads or deposits. By always working in the ore, evidence of the system of mineralization was continually accumulating, and the production of ore for the mill was always going on. Thus, there was always some return for the work done. For example, to mine a number of inclined leads lying close to each other, the ore would be most economically and quickly developed by an incline shaft on the lead most suited to shaft sinking, making main working shaft on such lead, and connecting the workings on adjoining leads and intersecting veins by tunnels.

In the case of an anticlinal formation, with the leads and veins dipping in different directions from a "divide," or "anticlinal line," the deposits are more profitably, quickly and cheaply developed by two inclines, each in opposite directions, than by a perpendicular shaft with cross tunnels to the various ore leads and veins. This is easily demonstrated by the following statements: The cost of sinking through country-rock is much more expensive than through ore deposits where the seams or partings afford favorable means of bringing away a large quantity of rock by a shot. As the incline shaft is carried down, the extent of ore face is extended, and ore levels may be started at any favorable point. As the ore faces are extended quantities of ore may be mined and crushed, affording revenue during the carrying on of the development work. In sinking a perpendicular shaft, as the depth gets farther away from the point of intersection of the line of the shaft and the plane of the lead, the cross-tunnel to reach said lead increases in length and cost. When the perpendicular shaft is put down to the intended depth, and the cross tunnels driven into the ore deposits, the mining of ore below the attained depth is only carried on at greatly increased expense by sinking winzes with extra hoisting, or by deepening the shaft and driving longer tunnels.

In the case of an incline shaft, a lower level may be made with a small additional proportionate cost, and without any extra cost for tunnels. The general "parallel" rule of ore deposits indicates the amount of cross tunnel work needed to connect them, and is as much in favor of incline as the perpendicular. Haulage by the line of the incline to turn into intersecting ore deposits is generally likely to be more favorable to the incline, because the deflection from the dip of a main lead to the dip of an angler is caused by some relation of the directions of fissuring of the two kinds of ore bodies. Many main leads are very nearly perpendicular, and thus all the advantages of two systems are blended.

Following the change of prices affecting the cost of work, came the mining of wider leads giving a lower rate per ton, but a larger volume of business. On account of the niggardly policy of many companies in not providing enough funds to carry underground work along to a satisfactory extent, the ore in sight was barely sufficient to supply the mill, often of very moderate capacity, and managers and miners were called upon to practice great self-denial, and tax their ingenuity to produce record-making results. As might be expected from the varying local conditions ruling at the same time, and the different kinds of ore deposits found by experiment to pay, there are as a result a great variety in the gold mining operations classed as quartz mining. As a general rule the rock in the selected working belts breaks from a blast in one direction much easier than in another, so that sometimes miners on one side of a shaft are progressing nearly twice as fast as those on the opposite side.

The underhand system of carrying down stopes is more usual than the overhand. Sometimes the underhand system of working is adopted in preference to the overhand, for the reason that in a wide mining belt the underhand method permits of much narrower working and less removal of rock. This often gives greater safety and saves timbering. Often a combination of sinking and breaking down is carried on when the underground haulage can be conveniently arranged. The style of working first adopted is shown in the sketch of the Lawson mine. A number of shafts were sunk at right angles to the direction of the course of the lead, the stopes were sunk and worked east and west from the shafts, as the rich ore was followed, and the ore was hoisted out of the nearest shaft. For the sake of convenience in getting at rich ore some distance from a shaft, a new shaft was generally made. These narrow inclined workings generally prevented the use of handy trolleys, and the quartz was shovelled several times to reach the skip. Scaffolds were made to store as much waste rock as possible. As the prices of work cheapened, and the miner's ingenuity designed handy contrivances, wider workings were made, more rock taken out of the mine, and more poor grade material went to the mill. Sometimes, as at Moose river, large quantities of loose rock and gravel lying on the outcrops of gold-bearing leads were put through the mill at a profit, and all such deposits of material are particularly noted.

By confining the mining to narrow leads it often cost $20 per tonf o ore. After a time a six-inch lead with a good belt and giving $10 per ton was considered in the range of profit. Still later leads of three and four feet wide, giving $6 per ton with large mill capacity, became profitable. When the mining of wide belts of material was undertaken to supply larger and heavier mills, $2 per ton was found to leave a good margin of profit.

The most successful example of mining a wide belt is that of the Richardson mine at Gold Brook, Guysboro county. The belt consisted of several leads in a belt, and from 11 to 18 feet in width was stoped out. The ore mines and crushes easily. The mill has forty stamps each of 850 pounds weight, with ninety drops to the minute. The success of this mine is largely due to the remarkable situation of the main shaft where the three dips show in outcrop. The north and south outcrops were only separated by an intervening body of quartzite of 48 feet in width, and the hoisting works and breaker were placed over them. A third incline from the east is now being built to the hoisting works. The ore-belt is wide and mines easily. The haulage being directly by the incline to the breaker permits of very cheap handling. The ore from the breaker is loaded by a shute into the mill cars and hauled to the crusher by an endless cable system. Mr. C. F. Andrews, the former manager of this property, in a paper read before the Federated Canadian Mining Institute, stated that the cost of mining and milling this material, including all charges, was $1.65 per ton. This is a remarkable showing, especially as it was done by hand drilling, and the hoisting, pumping and crushing done by steam power. In a letter of Mr. Andrews, he states that the real cost per gross ton or actual weight handled was actually $1.37½, he having deducted 20 per cent. from the weights for moisture. Some of the monthly mill runs for the 40-stamp mill have been returned, as from 2,200 to 2,300 tons actual weight going through the batteries.

Many other belts in the province may be mined after the style of the Richardson, and the problems to be worked out in such cases are the best location for good pay ore in quantities, the lowest possible cost of mining, the largest regular supply of mill ore with a minimum cost of haulage, and the most comprehensive and economical milling plant. Several open-cut operations at Mount Uniacke, Sherbrooke and Moose River have demonstrated their feasibility, and quarries up to 200 feet in width are likely to become features of Nova Scotia mining. This will call for large mills, from 50 to 100 stamps, running on selected ore, and will give a very large volume of business. Although this wide belt mining has come into vogue the finding of rich deposits of gold in the quartz has not ceased. In fact this comprehensive mining finds all the good streaks in the piece of mining ground, and gives large supplies of concentrating material.

The presence of so many varieties of minerals associated in gold ores is popularly looked upon as evidence of richness of the ores in the vicinity, and the large quantity of metal existing in the source from which vein filling came. The list includes galena, zinc blende, copper pyrites, arsenical iron pyrites, oxide of iron, copper glanc, molybdenite, native copper, sulphur, chlorite, feldspar, garnet, mica, calcite, felsite, etc. By deduction founded on circumstantial evidence, the miner looks with favor on the presence of what he calls the blue jack, black jack, mispickel, rust, white iron, copper pyrites, iron pyrites, spar, clay hulk, etc., as these have appeared more or less along with his good finds of gold. A consideration of these minerals brings up the subject of concentration. The bulk of the gold, however, recovered in the stamp mills, has been "free" gold, and the concentration at first attempted was by blankets and sluices. The limited amount of heavy metals caught in this way was put outside to the weather to be "rusted" so as to free a further percentage of gold to be caught either by panning or re-working in the mill when a lot of ore was going through. Sometimes this was done by washing the rusted sands on the amalgamated table plate of the mill. As the concentration of the finely crushed tailings of a gold mill is difficult on account of the great variation of specific gravity and size of grains, only a few vanner tables have been set up in the province.

The want of regular percentage of concentrates in the ores crushed has prevented the arranging of business regular in amount. The handling of large belts of material makes it feasible now to collect sufficient quantities of concentrates to supply chlorination works. At North Brookfield in Queen's Co., the Brookfield Mining Co. have a very complete and efficient chlorination plant of large capacity for handling the product of their mining and concentrating works. The cost of chlorination plants has largely prevented their installation in mining districts, but the demands for compactness and efficiency are likely to be met by designers, and estimates have lately been given for a complete roasting and chlorination outfit for $2,500. Plants of such moderate price can be put up in connection with single mining operations, and when concentration plants of sufficient capacity can be built at proportionate prices, a very great stimulus will be given to low grade ore mining and dressing

The adoption or development of the system of concentration, or adaptations of several methods of concentration for the securing of the best values in the metals of a property will, for a considerable time, supply vexing problems to Nova Scotia gold miners. In the "free gold" section of the Province the large quantities of free milling quartz material are of such extent that the reduction of refractory ores is not a necessity to an ordinary mining concern. The neglect of the concentrates in this region is generally estimated as causing a loss equal to the gold obtained by amalgamation. In the Cape Breton section the great preponderance of concentrating material in fine-grained sizes, shows the necessity of designing concentration and reduction works at the beginning. The existence of large quantities of galena will probably lead to the erection of numbers of reducing plants, as lead plants for smelting can be put up at estimated figures of $2,500. The crushing of these ores would be easily done and the connecting process of concentration will likely be solved with a limited amount of experiment in the case of galena occurring in the forms that it does.

Reference to the kinds of hoisting appliances used in the gold districts is interesting, so much of the work in former days being done by the expensive means of man power and horse power. The prospector starts with the windlass made of a piece of round spruce log with a crank in each end, and mounted on wooden standards. This rig has been worked to a depth of eighty feet when the water is very light. Two men on a windlass can lift out a lot of water, using as a baler an ordinary flour barrel. If there is considerable water in the ground, and usually after thirty feet, some use the horse "whip" for its quickness, and others the "whim" for its power. The use of the whip is restricted to the pit at which it is placed, but the whim by changing the rope can be used for hoisting from several shafts by running the ropes in by guide pulleys. When steam is adopted on a small mine the usual style of hoist is the wooden friction, made to work automatically and thrown into friction by the deckman and having a back balance post brake. Howell's automatic friction hoist and engine is an adaptation from mining practice in Nova Scotia, and as an automatic engine is made to do about all the necessary work at many mines. Thus it may be seen to be hoisting rock out of the shaft, pumping the pit water, working the stamp mill and turning the grindstone. This machine stands on a floor space five feet square, and any or all of the above operations may be going on without interfering with the others. With these friction and automatic engines, a fireman and deckman are all that are necessary on the deck or surface crew. The larger mines with large quantities of material to move have the double drum types of hoists.

Although wonderful results have been obtained by hand drilling, the value and size of properties as worked at present necessitate the general introduction of compressed air machinery.

In the milling of gold ores there are no special features in practice. The point to be noted in favor of Nova Scotia is the cheapness at which ores may be milled, being doubtless, under similar conditions lower than in any other part of the Dominion. This is largely due to the use of water power when obtainable, or in the case of steam mills, due to the cheapness of both coal and wood for fuel. Supplies are also reasonable. It must be remembered that milling in the past has been confined to amalgamation of "free gold" ores, and although this partial process has been unsparingly condemned by metallurgical engineers, miners have on their properties, near the surface, sufficient quantities of free gold ores to keep them employed for many years. From the miner's point of view in a "free gold" country he can do without chlorination and cyanide plants, but the majority of business men prefer a proposition with a large volume and variety of working departments, and less risk in supply and yield.

Although Nova Scotia is still turning up finds of rich nuggets of gold, investors generally are looking at the combined facts of the large quantities of free gold quartz and concentrating material occurring together, the many natural advantages for reaching these properties, the very reasonable prices at which machinery and supplies can be landed at the mines, and the remarkably cheap rates attained in the Nova Scotia mining country for mining and milling. The returns of the Mines Department shows the gold product up to September, 1896, to have been $12,000,000. It is estimated by the Department that this product has been accompanied by a waste in tailings of as many millions as were saved. The average yield of the quartz returned from 1862 to 1896, has been thirteen dwts. per ton. If the estimate of the waste is correct, and it also amounts to 13 dwts. per ton, then the value of 26 dwts., or say $25 per ton for everything, is extraordinarily rich. When the limits of paying value can be so closely estimated, as past experience warrants, the value of any property examined can be very closely arrived at. These considerations, combined with the facts of the discoveries of good pyritic ores, and the alternating of recurrence of free ores and refractory ores in any vein, indicate a large expansion in other processes following amalgamation. The variety in gold-bearing ores is very great, material yielding $1.50 per ton yielding a profit when occurring in large quantity in quarry, and rich quartz of 50 to 100 ozs. per ton having been obtained in several instances, and single nuggets of $1,000 in value.

The cost of mining work varies considerably, from the various causes affecting any particular pit. The use of air-drills has been carried into the workings of narrow leads with good results. It is interesting to note the results in the Lawson lead taken in this paper as a typical illustration of the narrow quartz vein. This vein is now being worked, and the shaft sinking and stoping carried on by air-drills. As before stated the Lawson might be styled a 5-foot vein with a 4-inch pay streak. The new shaft now going down is fifteen feet by five feet, and the stopes are three feet wide. The shaft was put down eighty-six feet in twenty-six days by two No. 2 Little Giant Rand Drills. The time occupied by the drilling work was eight hours out of twenty-four, and the other sixteen hours was occupied in hoisting, other work, and meal hours. Each drill bored eight holes of four and a half feet depth in its eight hours running time. The boring belt was the hard quartzite strip shown in the sketch of the vein.

NOTES ON THE RELATIONS OF THE MECHANICAL PROPERTIES OF STEEL TO THE CHEMICAL COMPOSITION.

By William Minty, B.A.Sc.

Steel Making Processes.—Mild steel is made from pig iron by the Bessemer acid, the Bessemer basic, the open hearth acid, and the open hearth basic processes. In the two acid processes, any phosphorus contained in the metals operated on is found in the finished product. These processes are, therefore, only applicable when pig and scrap iron can be obtained virtually free from these injurious metals. The two basic processes, by aid of the basic lined vessels which are alone employed, are able to remove phosphorus, and can, therefore, make use of the more impure and cheaper qualities of pig iron.

In the United States, the first, third and fourth methods are used for plates, and the first for structural shapes and rails. The Bessemer basic has scarcely obtained a footing. On the continent of Europe the Bessemer basic is largely used for section bars. In Great Britain the open hearth acid process is practically the only one in general use for the manufacture of ingots for plates. For rails and bars, the first three processes are largely used.

In the basic Bessemer steel process, the pig is run into the converter from the cupola, and subjected from below to a strong blast, which first removes the silicon and carbon, and then attacks the phosphorus.

Phosphorus like silicon adds much heat to the blast, but will not burn until practically all the silicon and carbon have been consumed. It has been found that phosphorus cannot be consumed unless lime is present, and as this would attack and melt the ganister (silicic acid) lining, it is necessary to give the converter a basic lining (dolomite) from which the steel made by the process derives its name, basic steel. Only very little silicon may be tolerated in steel made by this process, as it attacks the dolomite. In order to obtain a sufficiently high temperature, which cannot be done by burning carbon, phosphorus must be present in large proportions, namely, two to three per cent. When the phosphorus is consumed spiegel and ferro-manganese are added in the right proportions, and the charge is ready for casting.

The acid Bessemer process differs from the one just described, in so far as the lining of the converter is acid (silicic acid = ganister), and that no lime is added. The slag is generated by the combustion of the silicon and the iron. Not a trace of phosphorus is removed in the process, and, therefore, the pig used must contain less than that to be allowed in the finished product. It must also contain a large percentage—two to three per cent.—of silicon for the production of sufficient heat.

Spiegel and ferro-manganese are then added either to the converter or in the ladle. To obtain the necessary carbon, either the blowing may be interrupted before all the carbon is burnt, or the carbon may first be all burnt and then reintroduce it with the necessary manganese. The latter is the more reliable method.

An incidental difference between the acid and basic process is that in the basic process during the afterblow for removing the phosphorus every trace of carbon disappears, while in the acid process this is not the case. So that by adding just sufficient ferro-manganese to remove all red shortness one would obtain a far weaker—but also a more ductile—material by the basic than by the acid process. The former is therefore used almost exclusively for producing steel for soft wire.

In the open hearth acid process pig iron is placed in the furnace, and when melted iron ore and about twenty-five per cent. scrap iron are added until all the carbon and silicon are consumed, and the ore reduced to iron; then spiegel and ferro-manganese are added, and the charge run.

The acid open hearth process does not remove either phosphorus or sulphur, and the pig and scrap used should therefore contain only traces of those impurities.

The basic open hearth furnace is identical with the above, except that on account of using a basic slag almost every trace of phosphorus disappears. It is also found that both the carbon and manganese are vigorously attacked by the flames, and after spiegel has been added it is difficult to hit the proper moment for running the charge.

It is unnecessary to enter into the details of other processes for steel production, as none of them have ever been used for the manufacture of boiler plate to which this paper specially refers.

INFLUENCE OF IMPURITIES.—It may be remarked that pure iron or steel has never been produced, and my remarks only apply to the effects on additions made to average qualities.

CARBON.—Increases tensile strength and tempering qualities, but reduces ductility, weldability and melting temperature.

PHOSPHORUS.—Has ascribed to it the chief blame for cold-shortness and all the general bad qualities of steel, but this seems only to be true if much carbon or sulphur is present. It also increases its liability to be burnt and reduces melting temperature.

SULPHUR.—Accentuates the bad effects of phosphorus, produces red-shortness and greasiness as regards welding.

SILICON.—Reduces elongation and melting temperatures, prevents blow holes, increases tenacity in presence of carbon, but does not accentuate the effects of phosphorus.

MANGANESE.—Intensifies the influence of carbon except as regards tempering properties and neutralizes red and cold-shortness of phosphorus, sulphur, etc.

NICKEL.—Reduces tensile strength, but increases ductility, particularly as regards impact. It neutralizes the influence of carbon and perhaps phosphorus and reduces corrosion.

More detailed information as to the effects which these and other impurities have on the good qualities of steel can, unfortunately, only be obtained from conflicting authorities whose experimental results are often impracticable and exceedingly vague. This is due to the difficulty of obtaining pure metals, as the various impurities often accentuate or neutralize one another, to neglect and difficulty experienced in analysis, and of universal ignorance as to the effects of various mechanical treatments.

INTRODUCTION TO METHODS.—In this paper I propose giving a summary of the results of tests made on the mechanical properties, with special reference to tensile strength and the effect of each addition of some impurity, and from this attempt to measure the effect of each element singly and combined with other impurities.

As a specific basis for the determination of these various effects an accurate chemical analysis forms an essential part. The mechanical properties are altered more or less by the various percentages of the different elements entering into the composition of the iron or steel. The presence of an element which alone would be objectionable may not be so when a number of others are present, also some elements modify the influence of others, while others, themselves objectionable, act as antidotes for more harmful impurities. I have noticed in the curves I have plotted that the relative amounts and the sums of the various elements vary slightly according to the slight variations in the process of manufacture. For this reason the results laid before you were made entirely from basic open hearth steel rolled into plates varying from 3/8 inch to 9/16 inch in thickness, and cut

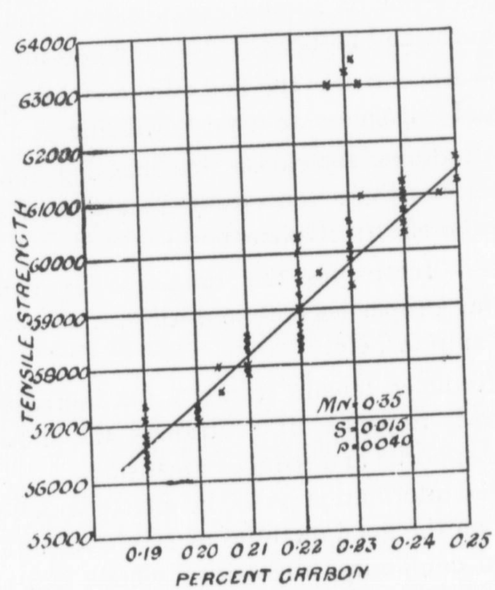
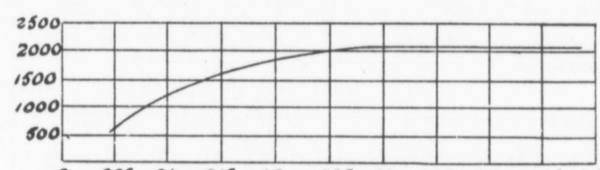


Fig. 1.

from sheets, all of which I have reason to believe were rolled from approximately the same size of ingot. As a single analysis may not indicate an average quality, it is necessary to have access to a great number of tests and analyses before arriving at anything like an approximate result as to the effect of any one element.

The conclusions given in this paper as to the effect of various elements, when acting with the other elements generally present, were arrived at by determining the analysis and various mechanical properties when the amounts of all but one of the principal elements remained constant. The results so obtained were then compared with another series of tests in which one of the elements which had remained constant in the first series of tests varied in proportion, and the other elements remaining constant. It will be seen that the method is only approximate since variations of the amounts of any element may modify the inter-actions between the other elements. To eliminate as far as possible this error I have plotted a series of curves extending over the complete range of boiler steel plate. I have compared the results with other tests on steel made by the basic process, and have found them correct within the limits of stee used for all kinds of boiler construction. The purer the iron the more difficult it is to predict mechanical properties from chemical analysis alone. All the plates from which test pieces were taken were "normal" steel, that is steel as received from the rolls, heated in a muffle and air covered. Figure 1 is an example of a carbon curve. Steels were selected of approximately the same thickness, cut from the same sizes of plate and having the same percentage of manganese,



THE VERTICAL ORDINATES GIVE THE INCREMENT OF TENSILE STRENGTH FOR EACH INCREASE OF 0·01 PERCENT PHOSPHORUS

FIG. 2.

phosphorus and sulphur, but with a varied percentage of carbon. A curve of tensile strength for the different percentages of carbon was plotted. Another curve was then plotted under the same conditions, excepting, say, 0.30 per cent. instead of 0.35 per cent. manganese. In this way curves were plotted in which the percentage carbon varied from 0.12 to 0.40; manganese 0.10 to 0.50; phosphorus 0.005 to 0.055, and sulphur 0.01 to 0.025.

With these curves once plotted it is easy to determine the effect on the tensile strength for each increment of any particular element. All the steels tested were mild carbon steels and the increase for each 0.1 per cent. carbon was found to be constant throughout the range of carbons.

The effect of equal increments of manganese was found to be constant in increasing the tensile strength and in hardening the iron as carbon did, except that larger quantities were required to obtain the same effect. The effects of phosphorus on the tensile strength are shown in Fig. 2. In the only tests ever before made on the action of phosphorus on steels the mechanical effects were credited with being constant, as in carbon and manganese. My conclusions were obtained by plotting curves with all the elements excepting phosphorus constant, and also by making carbon variable as shown in Fig. 3, where it will be noticed that for a given increase in the higher percentage of phosphorus the increase of tensile strength is

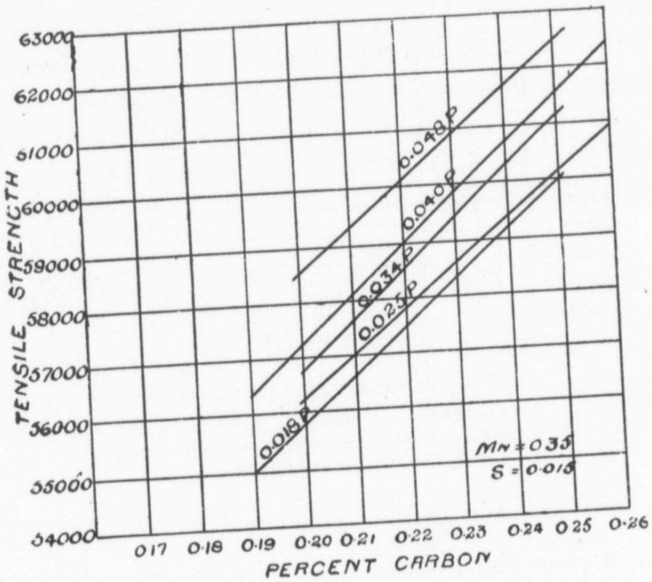


FIG. 3.

greater than it otherwise would be for a low phosphorus steel. Phosphorus, however, does show a constant effect the same as carbon and manganese when the steel contains more than 0.25 per cent. of this element.

The quantity of sulphur in the specimens examined was small and hence negligible. The injurious effect of large quantities, as 0.05 per cent., is not due to sulphur as such, but to its formation into a sulphide of iron, utterly destroying cohesion between the steel crystals. The presence of silicon may accentuate the effect of sulphide of iron. Referring to Fig. 1, the isolated points in the steels with 0.23 per cent. carbon contained sulphides of iron. It is evident that traces of sulphide of iron increases the tensile strength. Its action on the steel is of the same nature as phosphorus, but much more active. Tests show that the addition of manganese tends to prevent sulphur steels from passing into the more treacherous sulphides. When the sulphur is low a high silicon will not affect the steel.

I have found for all the basic steels examined that by extending the carbon curves, such as Fig. 1, to zero carbon and then subtracting the increase of tensile strength due to manganese, phosphorus, sulphur and silicon, the resulting tensile strength is constant at 34,600 pounds, with a probable error of fifty pounds. Hence to find the tensile strength of any basic steel (normal) from $\frac{3}{8}$-inch to $\frac{9}{16}$-inch thick, rolled from ingots similar to the above, and having given the analysis of the steel, proceed as follows: To the base 34,600 pounds add 80 pounds for each 0·01 per cent. manganese. Multiply the ordinate of Fig. 2 by the percentage phosphorus and add to the above. Then add 850 pounds for each 0.01 per cent. carbon. As boiler steels only contain traces of silicon and sulphur their action on tensile strength may be neglected. Referring to the steels of Fig. 1 with composition, manganese, 0.35; sulphur, 0.015; phosphorus, 0.020, and carbon (say) 0·20, its tensile strength would be as follows:

		34,600
Manganese,	35 × 80 =	2,800
Phosphorus,	1,500 × 2 =	3,000
Carbon,	850 × 20 =	17,000
		57,400

The results thus far shown have been entirely on the tensile strength. An attempt has been made to plot results on elastic limit but so far no satisfactory or reliable results have been obtained as the elastic limit is not obtained with the same accuracy as the ultimate tensile strength, similar steels giving conflicting results. The results of tests on the relations of chemical composition to the elastic limit are of more importance to the engineer than those on tensile strength, but nothing has been done on this line.

Several curves have been plotted with the elastic limits for different percentages of phosphorus and carbon. Figure 4 is one with phosphorus as the variable. The elongation increases with a decrease of phosphorus and of carbon, but it is yet impossible to give definite values. It is even possible the figures given for tensile strength may be revised. Howe says phosphoric steels are liable to break under very slight tensile strength if suddenly or vibratorily applied. Much as phosphorus diminishes ductility it diminishes toughness to shock steel more, thus rendering it unfit for all purposes. "The effect on ductility seems to be very capricious, for we find many cases of high phosphoric steel which show excellent elongation, contraction and even fair elastic ratio, while side by side with them are others produced under apparently similar conditions, but brittle." The excessive variation of the points plotted in Figure 4 bears out this statement. It is also possible that the chemical analysis may often be returned incorrectly when dealing with such small traces of phosphorus.

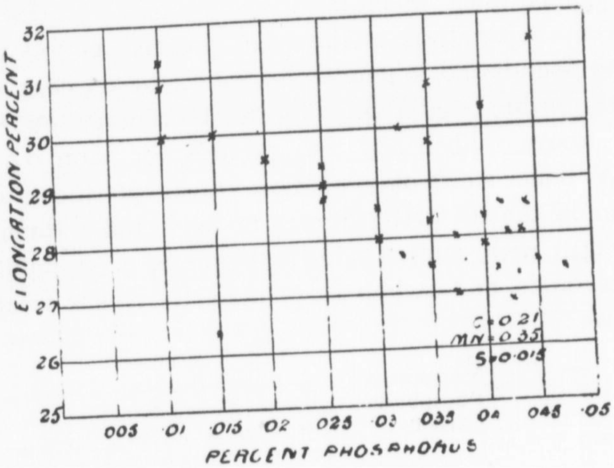


FIG 4.

So far as at present determined the ratio of elastic limit to tensile strength appears to depend principally on the amount of carbon present on the steel. The milder the steel the higher the ratio. Its value for the mildest steel is approximately seventy per cent., and decreases until it reaches a minimum in cast steels.

From certain tests made by several authorities on tortion the behavior of the metal appears to be similar to that in tension. The twist is proportional to the stress until the limit of elasticity of the outer fibres is reacted. After this point the twist increases faster than the stress.

For Bessemer steels the ratio of the limit of elasticity to the maximum torsion stress and tension stress are both equal.

The resistance to shear is always less than the tensile strength, the higher the tensile strength the lower the ratio. For Bessemer and crucible steels this ratio is 70 per cent., for open hearth steels 80 per cent., and for rivet iron 85 per cent. The shearing strength of ordinary steels is greater in torsion than in a direct shear. The limit of elasticity in torsion to the limit in tension is about one half the ratio of the breaking load in torsion to fracture in tension, the ratio decreasing with decrease in carbon from $\frac{5}{8}$ to $\frac{9}{16}$. Comparing this ratio with that in tension it will be noticed that there is here an opposite effect, the ratio of limit of elasticity to limit of fracture decreasing with the increase of carbon or of tensile strength.

TESTS OF ONTARIO BUILDING STONES.

The following is the report presented January 31st, 1893, by a committee of the council of the Ontario Association of Architects. The committee consisted of Messrs. W. G. Storm, D. B. Dick, E. Burke, S. G. Curry, S. H. Townsend (chairman), and was assisted by Profs. Galbraith and Coleman, and Messrs. Wright and Rosebrugh of the School of Practical Science.

The stone to be tested was first sawn into cubes, then set in plaster of Paris in a steel frame made for the purpose and brought to perfectly parallel faces by rubbing.

In placing the specimens in the machines care was taken that the pressure was always applied at right angles to the natural quarry bed of the stone.

Specimens of the stones tested have been preserved and a complete set of these, together with all data connected with the tests, including in most cases photographs of the stones broken, have been kept at the School of Practical Science, and also at the office of the above Association.

No. 1.

Connecticut Brown Sandstone from the Middlesex Quarry Co., Portland, Connecticut. Supplied by Messrs. Brown & Love.

SANDSTONE, medium grained, purplish brown, consists of quartz grains, with some grains of felspar, and many scales of white mica. Very slight effervescence with acid.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3⅛ x 3	2⅞	89,000	9,493	
B	3⅛ x 3	2⅞	92,000	9,813	
C	2⅞ x 3 1/16	2⅞	78,000	8,858	8,896
D	2⅞ x 3	2⅞	64,000	7,420	

No. 2.

Stone from the "Mono" Quarry, owned by the Owen Sound Stone Co., and situated two and a half miles north of Orangeville, Ont.

Sandstone, very fine grained, pale greenish gray. Effervesces slightly with acid, showing presence of carbonates. The stone is in beds of from one to four feet thick and can be got in large sizes. It has to be teamed to Orangeville Station.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	$3\frac{1}{16}$ x 3	$2\frac{7}{8}$	139,600	15,194	
B	$3\frac{1}{8}$ x 3	$2\frac{7}{8}$	130,000	13,866	
C	$2\frac{13}{16}$ x 3	$2\frac{7}{8}$	111,000	13,155	
D	3 x 3	$2\frac{7}{8}$	131,000	14,555	14,192

No. 3.

Credit Forks Brown Stone, from Carroll & Vick's No. 2 Quarry, Credit Forks, Ont.

Sandstone, fine grained, reddish brown. Contains quartz, and a little felspar and mica. The stone is in beds of four feet and under, and can be handled in pieces up to five tons. Quarry 300 yards from Railway.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	....	....	....	....	
B	$2\frac{7}{8}$ x 3	$2\frac{7}{8}$	131,000	15,188	
C	$2\frac{15}{16}$ x 3	$2\frac{7}{8}$	130,000	14,751	
D	3 x 3	$2\frac{7}{8}$	133,000	14,777	14,905

No. 4.

Stone from the St. John's Quarries, Longford Mills, owned by Messrs. A. McPherson & Co.

Limestone, compact, with some fissures, pale gray, effervesces strongly with acid. The stone is in beds of from 6 inches to 2 feet, and may be loaded upon cars in quarry.

Specimen.	Section under Pressure.	Height.	Crushing. Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	$2\frac{7}{8}$	200,000	22,222	
B	$3\frac{1}{16}$ x $3\frac{1}{16}$	$2\frac{7}{8}$	210,000	22,390	
C	....	....	....	....	
D	3 x 3	$2\frac{7}{8}$	175,000	19,444	21,352

No. 5.

Stone from the Hope Bay Quarries, owned by Messrs. Wm. Hooson & Sons, Plumper Pass, B.C.

SANDSTONE, medium grained, with some larger grains, bluish gray. Contains quartz, with some felspar and mica, and probably hornblende. Effervesces slightly with acid. The stone is in parallel beds of from two to ten feet, and can be loaded on scows or light draught vessels at the Quarry, which is situated on the North shore of Pender Island, about midway between Victoria, Vancouver, and New Westminster.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	$3\frac{1}{16}$ x 3	$2\frac{7}{8}$	200,000	21,768	
B	....	....	....	....	
C	$2\frac{15}{16}$ x 3	$2\frac{7}{8}$	186,000	21,106	
D	$3\frac{1}{16}$ x $3\frac{1}{16}$	$2\frac{7}{8}$	176,000	18,765	20,546

No. 6.

Stone from the Parrsboro Marble Quarry, owned by Mr. Thos. Kirkpatrick, and situated two miles west of Parrsboro, N.S.

LIMESTONE, very fine grained, dark gray (almost black), on fresh surface, bituminous odor when struck, effervesces strongly with acid. The stone is in beds of from four inches to two feet, and must be teamed to Parrsboro, where it can be shipped either by rail or water.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	$2\frac{15}{16}$ x $3\frac{1}{16}$	$2\frac{7}{8}$	199,000	22,120	
B	3 x $3\frac{1}{16}$	$2\frac{7}{8}$	200,000	21,768	
C	$2\frac{15}{16}$ x 3	$2\frac{7}{8}$	189,000	21,446	
D	3 x 3	$2\frac{7}{8}$	195,000	21,666	21,750

No. 7.

Rain Drop Stone, from Fuerst, New & Co., Chicago. Supplied by Messrs. Brown & Love.

SANDSTONE, medium grained, with a few pebbles ¼ inch in length, pale purplish brown with many darker colored spots. Contains quartz, felspar, and a green mineral, also some scales of mica. A whitish cement in the lighter portions. Arigillaceous odor.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	2⅞	52.000	5.777	
B	$2\frac{15}{16}$ x 3	2⅞	51,000	5,787	
C	3 x 3	2⅞	53.000	5,888	
D	3 x 3	2⅞	52,500	5,833	5,821

No. 8.

Corsehill (Scotland) Red Sandstone. Supplied by Messrs. Brown & Love.

SANDSTONE, very fined grained, brick red. Components hardly distinguishable with lens Argillaceous odor, effervesces very slightly with acid.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	2¾	136,000	15.111	
B	$2\frac{7}{8}$ x 3	2¾	150,600	17,461	
C	3 x 3	2¾	143,000	15,888	
D	$3\frac{1}{16}$ x 3	2¾	144,000	15.674	16.033

No. 9.

Ohio Blue Sandstone "BEREA." From the Cleveland Stone Co, Cleveland, Ohio. Supplied by Messrs. Brown & Love.

SANDSTONE, fine grained, pale gray, consists of quartz grains with yellow and black particles, probably oxides of iron. Argillaceous odor.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	$3\frac{1}{16}$ x 3	2¾	79,000	8,598	
B	$3\frac{1}{32}$ x $3\frac{3}{32}$	2¾	92,000	9,810	
C	3 x $3\frac{1}{32}$	2¾	92,000	10,116	
D	3 x 3	2¾	99.000	11,000	9.881

No. 10.

Ohio Buff Sandstone. From the Cleveland Stone Co., Cleveland, Ohio. Supplied by Messrs. Brown & Love.

SANDSTONE, fine grained, pale brownish gray or buff Contains quartz, some felspar, and mica, and a rusty iron oxide in small particles.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	$2\frac{3}{4}$	68,000	7,555	
B	$3\frac{1}{8}$ x $2\frac{15}{16}$	$2\frac{3}{4}$	68,000	7,406	
C	$3\frac{1}{16}$ x $3\frac{1}{16}$	$2\frac{3}{4}$	77,000	8,209	
D	3 x 3	$2\frac{3}{4}$	80,000	8.888	8,014

No. 11.

Stone from the Dorchester Union Freestone Co., Wallace, N.S. Supplied by Messrs. Brown & Love.

SANDSTONE, fine grained, greenish gray. Grains of quartz, with a good deal of felspar and other decomposed silicates, clayey cement.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	$2\frac{3}{4}$	96,000	10,666	
B	$3\frac{1}{16}$ x $3\frac{1}{16}$	$2\frac{3}{4}$	100,000	10,662	
C	3 x 3	$2\frac{3}{4}$	95,000	10,555	10,628
D	....	....	....	....	

No. 12.

New Brunswick Brown Sandstone. Supplied by Messrs. Brown & Love.

SANDSTONE, rather coarse grained, purplish brown. Contains quartz with much felspar, and a green silicate; also a few scales of mica. Strongly argillaceous odor, effervesces with acid in spots.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x $2\frac{15}{16}$	$2\frac{3}{4}$	68,000	7,716	
B	3 x $3\frac{1}{16}$	$2\frac{3}{4}$	62,000	6,748	
C	3 x $3\frac{1}{16}$	$2\frac{3}{4}$	68,000	7,401	
D	$3\frac{1}{16}$ x $3\frac{1}{16}$	$2\frac{3}{4}$	65,000	6,930	7,198

No. 13.

Stone from the Anderson Quarries, owned by Mr. Thos. B. White, Gordon, P.O., and situated two miles from Amherstburg on the M.C.R.R., and three-quarters of a mile from the Detroit River.

Dolomitic Limestone, very fine grained, pale buff or yellowish brown, contains grains of quartz, effervesces somewhat with cold dilute acid. The stone is in beds of from three inches to five feet, and can be handled in large sizes.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	$2\frac{7}{8}$	86,000	9,555	
B	3 x $2\frac{13}{16}$	$2\frac{7}{8}$	89,000	10,548	
C	$3\frac{1}{16}$ x $3\frac{1}{16}$	$2\frac{7}{8}$	85,000	9,062	
D	3 x $2\frac{15}{16}$	$2\frac{7}{8}$	78,000	8,851	9,504

No. 14.

Stone from Six Mile Brook Quarry, owned by Mr. Robert L. Elliott, Stillman, P.O.

Sandstone, very fine grained, greenish gray. Contains grains of quartz with a large amount of green and brown silicates, a little mica, and a few fragments of garnet. The stone is in beds of from two to six feet thick, and has to be teamed ten miles to ship by rail or water.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x $2\frac{15}{16}$	$2\frac{3}{4}$	76,000	8,624	
B	$3\frac{1}{16}$ x 3	$2\frac{3}{4}$	89,000	9,687	
C	$3\frac{1}{16}$ x 3	$2\frac{3}{4}$	96,000	10,448	
D	3 x 3	$2\frac{3}{4}$	92,000	10,222	9,745

No. 15.

Stone from the Montreal and Russel Street Quarry, Kingston, owned by Mr. W. S. Shufflebotham, 466 Montreal Street, Kingston, Ont.

LIMESTONE, compact, pale bluish gray, effervesces strongly with acid. Stone in beds of from two to twelve inches, and can be obtained in large sizes.

Specimen.	Section under Pressure.	Height.	
	Inches.	Inches.	These stones all stood a pressure of 200,000 lbs. without crushing. B was cracked.
A	$2\frac{7}{8} \times 3\frac{1}{16}$	$2\frac{7}{8}$	
B	$2\frac{15}{16} \times 3$	$2\frac{7}{8}$	
C	$3\frac{1}{8} \times 3$	$2\frac{7}{8}$	
D	$3 \times 3\frac{1}{16}$	$2\frac{7}{8}$	

No. 16.

Stone from the Rama Limestone Quarry, owned by Mr. W. R. Scadding, Longford Mills, Ont.

LIMESTONE, compact, with some crystals of calcite, pale bluish gray, somewha flawed with veins of calcite; effervesces strongly with acid. Stone in beds of 4, 6, 8, 9, 11, 12, 14, 16, 18, 24, 27 and 28 inches, and can be obtained any size up to six feet square. The Grand Trunk Railway runs through the quarry. Specimens prepared by owner.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches	Pounds.	Pounds.	Pounds.
A	3×3	3	78,000	8,666	
B	$3\frac{1}{32} \times 3\frac{1}{16}$	3	132,000	14,219	
C	3×3	3	116,000	12,889	
D	$3\frac{1}{32} \times 3\frac{1}{32}$	$3\frac{1}{32}$	64,000	6,967	10,685

No. 17.

Pelee Island Limestone. Supplied by Messrs. Brown & Love.

LIMESTONE (probably somewhat dolomitic), very fine grained, pale buff or yellowish brown, effervesces somewhat with cold dilute acid.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds
A	3×3	$2\frac{13}{16}$	61,000	6,777	
B	$3\frac{1}{16} \times 3\frac{1}{16}$	$2\frac{7}{8}$	72,000	7,676	7,226

No. 18.

Stone from the Queenston Limestone Quarries, owned by Messrs. P. A. Johnson & Co., Queenston.

LIMESTONE, rather fine grained, pale gray, effervesces strongly with acid. The stone is in beds of from one to five feet, and can be handled in large sizes. May be shipped on G.T.R. cars in quarry, or by water, $1\frac{1}{2}$ miles from quarry.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	$2\frac{13}{16}$	134,000	14,888	
B	3 x 3	$2\frac{7}{8}$	122,000	13,555	
C	3 x $3\frac{1}{16}$	$2\frac{7}{8}$	135,000	14,694	
D	3 x 3	$2\frac{7}{8}$	130,000	14,444	14,395

No. 19.

Stone from the Victoria Limestone Quarry, Cookston, North Hastings, owned by Messrs. Brodigan & Co., Madoc, Ont.

LIMESTONE, very fine grained, with some crystals of calcite, rather dark gray, slightly brownish on fresh surface, bluish grey on dressed face, slightly bituminous, effervesces strongly with acid. Stone in beds of 6, 10, 12, 14, 16, 18, 20, 24, 28, 38 and 48 inches, and can be got out in any sizes possible to handle. Quarry on main line G.T.R.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	$2\frac{13}{16}$	183,000	20,333	
B	3 x $3\frac{1}{16}$	$2\frac{7}{8}$	147,000	16,000	
C	....	....	....	....	
D	$2\frac{3}{4}$ x 3	$2\frac{7}{8}$	181,000	21,939	19,424

No. 20.

Red Granite, from the New Brunswick Red Granite Co., Mr. F. T. C. Burpee, Manager, St. John, N.B.

GRANITE (var. Granitite or Biotite Granite), coarse grained, strong flesh red, appears fresh and takes a good polish. Contains quartz (gray), orthoclase (red), plagioclase (yellow), and biotite (black). Specimens prepared by owner.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	$2\frac{1}{8}$ x $2\frac{5}{16}$	2	68,000	13,837	
B	$2\frac{1}{8}$ x $2\frac{1}{4}$	$2\frac{1}{8}$	84,000	17,568	
C	$2\frac{1}{4}$ x $2\frac{1}{4}$	$2\frac{1}{8}$	69,000	13,629	
D	$2\frac{5}{16}$ x $2\frac{5}{16}$	2	79,000	15,617	15,162

No. 21.

Gray Granite, from the New Brunswick Red Granite Co., Mr. F. T. C. Burpee, Manager, St. John, N.B.

GRANITE (var. Granitite), medium grained, gray, fresh looking. Contains quartz, orthoclase in grayish white, slightly porphyritic crystals, black biotite, and some reddish brown titanite. Specimens prepared by owner.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	$2\frac{1}{8} \times 2$	$2\frac{1}{8}$	69,000	16,235	
B	$2\frac{1}{8} \times 2\frac{1}{8}$	$2\frac{1}{4}$	68,000	15,059	
C	$2 \times 2\frac{1}{8}$	2	79,200	18,635	
D	$2\frac{1}{8} \times 2\frac{1}{16}$	2	61,000	13,917	15,961

No. 22.

Gray Granite, from the Standstead Quarry, owned by Mr. James Brodie, Lineboro, P.Q.

GRANITE (var. Granitite), rather coarse grained, gray, fresh looking. Contains quartz, white orthoclase, white plagioclase, greenish black biotite, and a trace of pyrite. Stone in beds of from one to ten feet, and can be obtained any size up to 40 feet long. Specimens prepared by owner.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A					
B	$2\frac{1}{4} \times 2\frac{1}{4}$	2	32,000	6,321	
C	$2\frac{1}{8} \times 2\frac{1}{16}$	2	30,000	6,844	
D	$2\frac{1}{16} \times 2\frac{3}{16}$	2	40,000	8,865	7,343

No. 23.

Stone from the Cumberland Basin Brown Stone Quarry, owned by the Atlantic Brown Stone Co., Box 118, Sackville, N.B.

SANDSTONE, medium grained, purplish brown. Contains quartz, much reddish felspar, and a few grains of green silicate, and scales of mica. Strong argillaceous odor, effervesces slightly with acid. Stone in beds of from four to ten feet, and can be got out up to twelve tons.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches	Pounds.	Pounds.	Pounds
A	3×3	3	30,000	3,333	
B	$2\frac{15}{16} \times 3\frac{1}{16}$	$3\frac{1}{16}$	42,600	4,735	
C	$2\frac{31}{32} \times 2\frac{31}{32}$	$3\frac{1}{16}$	38,000	4,311	
D	$3 \times 3\frac{1}{16}$	3	41,500	4,517	4,224

No. 24.

Indiana Oolitic Limestone, from Perry Bros.' Quarry, Ellettsville, Monroe Co., Indiana. Supplied by Messrs. Brown & Love.

LIMESTONE, very fine grained, oolitic globules easily seen with a lens, pale yellowish-brown or buff, effervesces strongly with acid.

Specimen.	Section under Pressure.	Height.	Crushing Stress.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches	Inches.	Pounds.	Pounds.	Pounds.
A	....	....	....	....	
B	....	....	....	....	
C	3 x 3	3	56,900	6,322	
D	....	....	....	....	6,322

No. 25.

Miramichi Stone from the "French Fort" Quarries on the Miramichi River, owned by Mr. C. E. Fish.

SANDSTONE, somewhat fine grained, yellowish brown. Contains quartz, a good deal of felspar, and a few scales of mica. Stone in beds of from one to eight feet, and can be quarried up to ninety cubic feet.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds.	Pounds.
A	3 x 3	3	41,000	4,555	
B	2⅞ x 3	3	36,000	4,174	
C	2⅞ x 3	3	42,000	4,870	
D	3 x 3	3	47,500	5,277	4,719

No. 26.

Sandstone from Thomas Gatelawbridge Quarries, supplied by Messrs. W. M. Knowles & Co., Agents, Montreal.

SANDSTONE, rather fine grained, brick red, reddish quartz grains, with a few whitish grains of felspar. Argillaceous odor. No effervescence with acid. Specimens prepared by owner.

Specimen.	Section under Pressure.	Height.	Crushing Load.	Crushing Stress per Square Inch.	Average Crushing Stress per Square Inch.
	Inches.	Inches.	Pounds.	Pounds	Pounds.
A	3 x 3	2⅞	37,500	4,166	
B	2⅞ x 2⅞	3	40,000	4,839	
C	2¾ x 2⅞	3	46,000	5,818	
D	2⅞ x 3	3	37,000	4,289	4,778

PAPERS AND BOOKS RECEIVED.

In addition to the papers herein presented the Society also received one on "Gold Mining on the Witwatersrand," by Mr. Edgar J. Laschinger, B.A. Sc., a graduate of 1892, who is now chief draftsman for the Consolidated Gold Fields Co. of the South African Republic.

The paper is illustrated by forty excellent photographs, and we deeply regret that the expense of engraving prevents us publishing the matter of which the photographs are a necessary part.

They are, however, being neatly mounted and bound, and are to be kept in the library for reference.

We are very pleased to acknowledge Mr. Laschinger's kindness.

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NOTES AND FORMULES.—Through the kindness of the publishers, Messrs. E. Bernard & Co., 53ter Quai des Grands-Augustins, Paris, the School has been presented with a copy of "Notes et Formules de l' Ingénieur du Constructeur-Mécanicien du Métallurgiste and de l' Electricien," edited by L. A. Barre and Ch. Vigreux, assisted by ten other specialists in the branch on which they have written.

As its title indicates the book is a valuable collection of rules and formulas on mechanical, electrical and metallurgical work of some 1,300 pages. The book is now in its 11th edition, having been originally published sixteen years ago. This edition has been thoroughly revised and brought fully up to date.

It is a matter of regret that it is too large for the pocket, but it nevertheless is a very valuable book of reference.

The price in France is 10 francs, and in other countries 11 francs.