THE TRANSACTIONS

OF THE

Canadian Mining Institute

(With which is incorporated the Transactions
of the Mining Society of Nova Scotia)

1920

EDITED BY THE SECRETARY-EMERITUS

VOLUME XXIII

"The Institute as a body shall not be responsible
for the statements and opinions advanced in the
papers which may be read or in the discussions
which may take place at its meetings."

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MR. O. E. S. WHITESIDE,
President of the Institute, 1920.

REPORT OF COUNCIL FOR THE YEAR ENDING DECEMBER 31st, 1919.

In submitting to the members of the Institute the report for the year ending December 31st, 1919, the Council desires to express its satisfaction that notwithstanding the greatly increased expenditure during the past year, the financial position has been such that all obligations have been met by the usual revenue, and although there is not a surplus similar to that of last year, there is nevertheless a small one.

As was to have been expected, affairs not only in Canada but in all countries, were most unsettled industrially; this is attributable to the transition from war conditions to those of peace. This transition period was reflected in many ways in the activities of the Institute. It is gratifying to record that nearly all our members who were serving overseas have already returned and are being gradually absorbed again into the various branches of work in connection with mining and metallurgy.

National Prospecting

In April the Council appointed Mr. J. A. Dresser as a Committee of one to take immediate steps to interest the Federal Government in the scheme for employing suitably qualified returned soldiers as prospectors. A memorandum was prepared for presentation to the Ministers of Mines and of Soldiers' Civil Re-establishment, with whom the Committee had several interviews between April 16th and May 14th. Although these Ministers expressed sympathy with the plan, they failed to secure the interest of the Cabinet in the matter. An appeal was then made to the Prime Minister and the plan submitted to him for consideration; although the matter received his courteous attention, the deputation judged by his attitude that no further steps would be taken.

The plan, although rejected by the Cabinet, had in the meantime been endorsed by the Council of the Engineering Institute of Canada, the Royal Society of Canada, the Research

Council, the Great War Veterans' Association, the Committee of the House of Commons on Natural Resources, etc. Although it is to be greatly regretted that a plan such as this, which would be of the utmost assistance in helping to develop the mineral resources of Canada, has not been put into operation by the government, the untiring efforts of the Committee may yet, it is hoped, bear fruit. A strong feeling in favour of some form of public aid to prospectors was disclosed during the time of the Committee's activities and this justly warrants the renewed efforts of the Institute along these lines. It is possible that, in spite of the Federal Government's refusal to entertain the scheme, certain of the provincial governments will be induced to apply it to prospecting in their respective provinces. At the Western General Meeting held in Vancouver in November, a resolution was adopted to the effect that the Institute should again urge the government to adopt some suitable plan for giving aid to returned soldiers in prospecting, and that the Council should take such steps as might be necessary to ensure that the plan be operative during the season of 1920.[1]

The Council desires to convey to Mr. J. A. Dresser its thanks for the able manner in which he presented the case to the authorities and for his untiring efforts to obtain for the plan a favourable and sympathetic reception. Valuable assistance was also given by others.

NATIONAL INDUSTRIAL CONFERENCE

In the latter part of July the Minister of Labour requested the Institute to nominate five members whom he could invite as representatives of the mining industry in various parts of the country to attend the National Industrial Conference in Ottawa. The members nominated were Mr. D. H. McDougall, the Hon. George R. Smith, Mr. B. Neilly, Mr. O. E. S. Whiteside, Mr. W. R. Wilson. The Institute was also officially represented by the Secretary. The Conference was held during the week of September 14-20, and about two hundred representatives were present; these included federal and provincial Ministers, and delegates representing employer and employee in almost every industry throughout the country, as well as other representatives from technical societies, public organizations, etc. That such a conference was justified is seen in the spirit of compromise that was evident in nearly all the matters introduced for consideration. It appeared also to be generally realized that only by the united effort of capital and labour will Canada escape further industrial upheavals during the present unsettled period.

[1] It is to be noted that since this report was originally issued a plan on very similar lines to that advocated by the Institute has been put into effect by the Government of British Columbia.—Editor.

SALARIES OF TECHNICAL OFFICERS IN THE CIVIL SERVICE

At the Annual Meeting in March it was resolved that the Institute respectfully urge upon the Dominion and Provincial Governments the pressing need of paying better salaries to those civil servants who are technical officers of the Department of Mines and similar services. This resolution was duly forwarded to the authorities, whose acknowledgement was of an encouraging nature. Later in the year the Civil Service Commission's report appeared, but the classification and scale of salaries as dealt with in this report were so obviously unfair to the technical officers of the Department of Mines, etc., that the Secretary despatched a telegram of protest to the Prime Minister before action would be taken by Parliament. As a result of the general condemnation by various technical societies and individuals, the bill was withdrawn for further consideration, and later, in a completely revised form, which has had the general approval of the heads of departments, it was passed at the next session of Parliament. Naturally, in a few individual cases there is still cause for criticism, but this is more or less unavoidable, and provision has been made in the bill for the formation of a board of appeal to consider such cases. The Council of the Institute has already appointed a committee to investigate and report on what steps it might be advisable for the Council to take to secure the most satisfactory arrangements of the salaries of technical officers in the Mines Department under the new Act.

LEGISLATION FOR ENGINEERS

The question of legislation for engineers was brought to the attention of the Institute last year, and a committee was appointed to confer with a committee of the Engineering Institute of Canada in December, 1918. The result of this was that the Engineering Institute agreed not to apply for legislation until the bill as affecting mining and the profession of mining, had been approved of by the Canadian Mining Institute. The bill, as drawn by a special committee of the Engineering Institute, was published in the June BULLETIN for the information of our members. Later the Institute was notified that the Engineering Institute was not, as a body, applying for legislation, but, as the matter was one for the engineers in each province to decide, it was proposed to leave the initiative to them. In view of this the Council has notified the branches and requested them to consider any proposed legislation and to submit their reports as soon as possible, in order that the Council might be guided by the wishes of the branches in dealing with this subject. Reports from all the branches have not as yet been received, but it is hoped that at an early date the Council will be in a position to decide upon the best course to be adopted.

UNIFORMITY IN DRILL STEEL AND SOCKETS

In September the Engineering Standards Association asked for the Council's opinion on the desirability of obtaining uniformity in sections of drill steel and in the dimensions of chucks or sockets for such steel as used by different Canadian manufacturers. As the Council was of the opinion that such a step was desirable, the matter of securing representatives from the large manufacturers and purchasers who would be available to form a committee was next undertaken, and the branches were asked to nominate these. The names have been forwarded to the Secretary of the Canadian Engineering Standards Association with the suggestion that the nominees be constituted a committee to consider the subject.

COAL REPORTS FOR BRITISH ADMIRALTY

In August the Canadian Trade Commission informed the Institute that some years ago the British Admiralty had made numerous tests on Eastern Canadian coal to determine its value as fuel for the navy, and had recently taken up the matter again with a view to overcoming any objections against its use. The matter was referred to the Institute with the request that the Secretary assist the Commission in dealing with the matter. As a result, the Secretary communicated with all the operators whose coal had been tested and their suggestions respecting the adoption of grates, flues, etc., to suit Canadian coal have been duly forwarded to the Canadian Trade Commission for transmission to the proper authorities.

PROPOSAL TO FORM A COAL SECTION

In June a Committee was appointed to report on the feasibility of forming a Coal Section of the Institute. The formation of such a section was proposed in view of the fact that approximately twenty-five per cent of the members of the Institute are identified with coal mining, and a section would permit of co-operation between eastern and western coal miners for the exchange of papers and in dealing with all subjects of common interest. The Committee reported unanimously in favour of the organization of a Coal Section, and it is hoped that this will be done at the next Annual Meeting.

CHANGE OF NAME OF THE INSTITUTE

A proposal to change the name of the Institute to "Canadian Institute of Mining and Metallurgy" was submitted at the Annual Meeting, and it was decided to refer the matter to the members, to be decided by letter ballot. The result was in favour of the change; and the necessary legal steps to have this effected are at present being taken.

AMENDMENTS TO BY-LAWS

As a result of a letter ballot on the proposed amendments to the By-laws, which were submitted at the Annual Meeting, there is now a Professional Membership section of the Institute. A Committee has been appointed to pass on the qualifications of members who wish to become Professional members, but, although the work has been begun, it will be some time before the list is complete.

The By-laws as amended also provide that each province shall be represented on the Council by one Vice-President and five Councillors. This will come into effect at the next election.

EMPLOYMENT BUREAU

To assist in the re-establishment of members returning from overseas, the Secretary, early in the year, communicated with all such members asking them to forward statements of their needs and qualifications in order that they could be brought into touch with possible employers. As a result of this the 'Employment Bureau' was inaugurated, and the qualifications of members (mostly returned from overseas) available for employment have been published under this heading in the Monthly BULLETINS. Practically all the mining companies throughout the country promised co-operation. By this means employment has been found for a number of members seeking positions.

MEETINGS

The Twenty-First Annual General Meeting was held in Montreal on March 5th, 6th and 7th. The attendance was well over two hundred. The meeting was noteworthy in many respects. The fact that it was the first meeting of the Institute following the cessation of hostilities was responsible for the interest taken in the discussions of the broader problems of reconstruction, etc. It was also notable in that a joint session was held with the members of the American Institute of Mining and Metallurgical Engineers, forming a continuation of a

similar joint session that had been held in New York during the previous month.

The papers and discussions were of a high order, a military flavour being imparted to the proceedings by the presentation of two papers on tunnelling operations at the Front, a lecture on British operations in Palestine, moving-picture films of Canadian operations at the Front, and the unveiling of the War Memorial. The guests of honour at the annual dinner included His Excellency, the Governor-General; Mr. H. V. Winchell, President, and Mr. Bradley Stoughton, Secretary, of the American Institute of Mining and Metallurgical Engineers; Dr. H. M. Payne, of the American Mining Congress; Mr. F. S. Keith, Secretary of the Engineering Institute of Canada, and Mr. A. D. MacTier, Vice-President of the Canadian Pacific Railway.

The first Western General Meeting was held in Vancouver on November 26th 27th, and 28th, and was comparable in every respect with the General Meetings held in the East. The attendance was over two hundred. The programme was an eminently interesting one, and reflected the greatest credit on the committee of Vancouver members responsible for the arrangements. The success of this inaugural meeting is highly gratifying to the Council since it indicates that the decision to hold regular general meetings each year in the West as well as in the East was well justified.

Numerous successful meetings have been held by the various Branches throughout the country, the attendance being as high as 90 at some of these.

BRANCHES

The past year has been a record one for activity in the formation of new branches and the revival of older ones. The following new branches have been established: Hastings District Branch, Deloro, Ontario; Sudbury Branch, Sudbury, Ontario; Nanaimo Coal Section, Nanaimo, B.C.; Kootenay-Boundary Branch, Nelson, B.C.; North Coast Local Branch, Prince Rupert, B.C.; Vancouver Branch, Vancouver, B.C. The name of the

Western Branch has been changed to "British Columbia Division," the other branches in the province being under its jurisdiction.

PUBLICATIONS

The publications of the Institute during the year comprised twelve numbers of the Monthly BULLETIN and Volume XXII of the TRANSACTIONS. The latter will be ready for distribution in January, 1920. Thanks are due to those members who contributed papers and articles to the TRANSACTIONS and the BULLETIN, and to the following who acted as special contributors to the latter; Mr. T. C. Denis, Hon. Robert Drummond, Mr. R. P. D. Graham, Mr. W. G. Mitchell, Mr. W. R. Rogers, Mr. J. S. DeLury, and Mr. C. Camsell, as well as to Mr. R. A. Elliott, Mr R. H. Hutchison, Mr. J. A. Richards, and other branch secretaries. Acknowledgments are also due to the *Engineering and Mining Journal* for the continued kindness in contributing the monthly reports on the metal market; and to the American Institute of Mining and Metallurgical Engineers by whose courtesy we have been able to publish, monthly, their mining and metallurgical index.

LIBRARY

During 1919, except for the usual purchase of a few annual publications, no books have been bought for the Library.

The Institute's War Memorial now occupies a prominent position in the Library. With the consent of the contributors to the Tobacco Fund, the cash balance on hand when hostilities ceased helped defray the cost of this memorial.

Sets of TRANSACTIONS were presented to various public and reference libraries situated in the neighbourhood of mines, smelters, etc., where they will be available for the use of technical men. These donations have been gratefully accepted by the various library boards. Sets were also presented to the Mining Departments of Sudbury High School and the Provincial Institute of Technology and Art in Calgary.

LEAVE OF ABSENCE FOR THE SECRETARY-TREASURER

Owing to ill-health, Mr. H. Mortimer-Lamb was granted six months' leave of absence by the Council, and is at present recuperating in the milder climate of the Pacific Coast. The duties of the Secretary-Treasurer have been assumed by the Assistant Secretary.

MEMBERSHIP

A large number of accessions to the membership of the Institute has been made during the year. This is attributable to the activity of the Branches in British Columbia, the Manitoba Branch, the Sudbury Branch, the Hastings District Branch, etc.; and to numerous members, including, in particular, Mr. E. E. Campbell, Mr. E. J. Conway, Dr. E. T. Hodge, Professor J. M. Turnbull and Mr. E. J. Carlyle. The accessions are classified as follows:

Ex-officio members	1
Members	106
Associate members	40
Student members	5
Affiliated student members	3
Honorary members (ex-officio)	2
	157

The losses by death, resignation and removal were as follows:

Deaths	12
Resignations	33
Removals	83
	128

The loss by death of the following members is recorded with deep regret:—

A. H. Brown, Hon. Frank Cochrane, Geo. E. Drummond, Frederick Hobart, Walton Kellett, L. M. Lambe, Sir Wilfrid Laurier, J. A. MacDonald, A. O. Norton, C. A. O'Connell, R. W. Raymond and N. M. Thornton.

The following have resigned their membership during the year:—

A. K. Anderson, D. A. Brebner, F. J. Burle, J. N. Bulkley, I. C. Callander, Henry Clark, J. W. Collis, J. R. Cowans, Frank H. Crockard, B. Crowell, G. H. Eaton, Hon. G. H. Ferguson, F. N. Flynn, Homer L. Gibson, E. N. Howell, A. G. Kirby, O. O. Laudig; A. D. Little, J. E. McAllister, John MacDonald, R. B. McGinnis, K. F. Mather, Hon. H. Mercier, M. Nordegg, W. L. Paterson, A. C. Ross, W. J. Spain, N. A. Stockett, E. B. Thornhill, L. T. Walls, E. Waterman, B. C. McCrodan (resigned as student, elected member), and H. M. Roscoe (resigned as student, elected member). In addition, three affiliated student memberships were not renewed.

The following table gives a comparison of the membership for the years 1918 and 1919:

	Dec. 31, 1918	Dec. 31, 1919
Associate members	182	210
Patrons	4	2
Corresponding members	15	14
Honorary members	7	7
Fx-Officio members	50	50
Life members	12	16
Members	1014	1006
Student members	7	12
Affiliated student members	11	14
	1302	1331

D. H. McDougall, *President.*
H. Mortimer-Lamb, *Secretary,*
per R. R. Rose, *Asst. Secretary.*

Treasurer's Report

In spite of the facts that costs of printing, publishing, and supplies have continued to advance and that administrative expenses have been increased by the appointment of an assistant secretary, the granting of financial aid to branches, and the inauguration of the Western Annual Meeting, the year's revenue

has been sufficient to meet all expenses without increasing the membership subscription. At the same time it must be realised by all members that the present financial condition of the Institute can only be maintained by their co-operation in securing new members and by the prompt payment of annual subscriptions.

Receipts and disbursements during the year ending December 31st, 1919, were as follows:

Receipts

Cash in bank 1st Jan., 1919			$ 6,118.78
Subscriptions, annual		$9,763.00	
Less: Paid in advance	$925.00		
Rebates	275.55		
		1,200.55	
		8,562.45	
Life Membership		400.00	
Students		47.80	
Arrears		1.025.00	
Advance		45.00	
			10,080.25
Publication Sales		495.40	
Advertisements		1,648.37	
Binding		219.45	
Index		12.00	
			2,375.22
Miscellaneous:			
Dom. Government Grant		$3,000.00	
Sale of Anglo-French Bonds	2,869.37		
Profit	125.31		
		2,994.68	
Sale of Buttons		22.50	
Members' Subs., Tobacco Fund		276.51	
Revenue from Investments		1,252.50	
Donation Western Branch		200.00	
Bank Interest		173.17	
			7,919.36
			$26,493.61

DISBURSEMENTS:

Publication, Printing and Distributing	$5,733.43		
Editor's Salary	2,000.00		
		$ 7,753.43	
Rents, Rates and Taxes		1,126.75	
Telephones and Telegrams		261.23	
Secretary, Office Salaries	6,330.00		
Less: Charged to Pub.	2,000.00		
	4,330.00		
H. M. Lamb, 5 mos. in advance	1,500.00		
		5,830.00	
Postages		475.59	
Express		66.36	
Printing and Stationery		658.25	
Travelling Expenses		1,365.45	
General Expenses		319.99	
Meetings, annual		439.56	
		18,276.61	
Permanent:			
War Memorial Tablet	564.00		
Furniture	5.25		
Library	108.09		
Victory Bonds	5,091.94		
Balance in bank	2,447.72		
		8,217.00	
			$26,493.61

H. MORTIMER-LAMB, *Treasurer*,
per R. R. ROSE, *Assistant Secretary*.

Audited and verified,
(Signed) P. S. ROSS & SONS,
Chartered Accountants.

Montreal, 15th Jan., 1920.

PRESIDENTIAL ADDRESS

By D. H. McDOUGALL

Annual General Meeting, Toronto, March, 1920.

I propose on this occasion to ask your consideration of some circumstances connected with the economic utilization of our national resources

First of all, I would propound two statements as the text of my remarks, which, made from this chair, may strike you as unusual, and possibly heretical. The statements that I would make are that the natural resources of Canada are:

a. Very specialized;

b. Not inexhaustible.

I am aware that it has been the custom for many years to refer to our national resources by such terms as 'illimitable,' 'immense' and 'boundless,' and, while these phrases may have been warranted when our people numbered a few millions, is it not perhaps time to take stock of our national assets in the light of future population, which may within the lifetime of those present here be doubled, or maybe trebled?

Canadian views have possibly been coloured by reflection of those of our friends in the United States, whose prodigal natural resources have been a revealed phenomenon of modern times. The world has not seen before, and cannot see again, such a treasure house as the territory now known as the United States of America was when the white man first commenced to mine those resources, without much thought of the future.

In coal, iron, gold, silver, copper, zinc, lead; in the great forests of oak, pine, cypress, and walnut that formerly existed; in agricultural possibilities and variety of climate, there never was so fortunate a land seen by white man, nor ever can be seen again in its unspoiled and intact pristine wealth.

But, in Canada, have we so great an accumulation of potential wealth? It is much to be doubted. Our wealth is truly vast, but relatively, we have certain distinct limitations, which, if we will admit them now, and proceed in that wise and prudent manner which has become a proverb. to 'cut our coat according to our cloth,' we may to a large extent offset.

Three basic requirements of our national life are (to refer to them in their natural order of importance), coal, iron and wood; and if we enquire carefully into the extent of our national resources of these three essential materials, their limitations, so far, of course, as our prospecting and the progress of scientific research today have led us, will be disclosed.

To deal in order, and very briefly, with these three basic materials:

COAL

Our coal deposits do not include anthracite, barring some anthracite coals of small tonnage quantity in the West.

Our bituminous coals are concentrated in widely scattered localities—not, as yet, the most populous parts of Canada—and the gap between is wide and important.

With the exception of the great coalfield of Alberta and eastern British Columbia, which is of course, essentially one deposit, we have remaining only the coal deposits of Vancouver Island and those of New Brunswick and Nova Scotia.

The great reserves of the two coalfields of, respectively, Vancouver Island and Cape Breton Island are under the sea, and in regard to both these widely separated localities, the extent of the coal reserve depends on the progress that science will make in providing the means to transport light, air, and motive-power over long tracts of underground passages.

The limits to the mining of coal imposed by great depth of cover, and by distance from shore, are not known, because they have not been tested but it may be confidently presumed that these limits will be widened as human knowledge is increased by experience, and I merely wish to emphasize that the limitation exists.

In the same manner the removal of the present obstacles to the utilization of the low-grade lignites of the West rests with

the advances of applied science, as does also the means of making Canada independent—as far as may be— of importations of anthracite from the United States.

IRON

Canada's position in regard to iron ore is deducible from the definition of commercial iron ore which is contained in the Encyclopedia Brittanica article on 'Iron and Steel' by Dr. Henry Marion Howe, of Columbia University, who writes:

'Whether a ferruginous rock is or is not ore is purely a question of current demand and supply. That is ore from which there is a hope that metal can be extracted with profit, if not today, then within a reasonable time.'

Our iron ores, so far as we know them, consist of large occurrences of ore of relatively low iron content. Their utilization will depend on the progress made in concentrating these lean ores to compete economically with richer ores. This again is a matter for practical scientists, and some progress has already been made in this direction.

WOOD

The inroads that are being made upon our forests are a matter of notoriety, nor are these inroads accompanied by anything approaching adequate reforestation. Indeed it is doubtful whether in some instances reforestation is practical.

Enquiry will prove that timber lands are daily increasing in scarcity, and therefore in cost, and our limitations in Canada in this regard are not only distinct, but actually alarming.

What I have said respecting these three basic materials is quite probably true of other essential products, but these latter do not so properly concern us as a mining institute.

One general conclusion we may draw, namely, that efficient and full use of our resources is dependent upon the progress of science, which, applied to their limitations, and supplementing their deficiencies, will have the effect of increasing their quantity and duration.

Dealing now with my second statement, and with those natural resources that I have called 'specialized,' I would direct your attention to the fact that Canada contains almost the world's reserve of nickel, asbestos, and cobalt.

These minerals are chiefly important in being necessary in certain large industries, and, if this word is permissible, they are minerals possessing a 'strategic' value, inasmuch as our almost exclusive possession of these minerals should enable us to drive a fair bargain with those nations that possess essential natural resources with which Canada is somewhat meagrely or unevenly supplied.

The members of the Iron and Steel Section of our Institute have not been slow to comprehend the growing importance of alloy steels, but this is an industry that Canada should strive to make a national specialty. We have large water-powers and, consequently, the means of generating with comparative cheapness large quantities of electricity. Canada possesses, as mentioned, a preponderance of the world's nickel and cobalt, and in addition she is supplied with smaller quantities of chrome, molybdenum, and magnesite—from which magnesium is obtainable. As a producer of synthetic steels, the variety of which is now immense, Canada only requires for success the development of trained metallurgists, for she has all the natural resources necessary.

The dominance of Canada in asbestos production and her important contribution of amber mica, should enable us to take an overshadowing position in the electrical industry, in which these two products find such varied and indispensable employment.

With reference to many of the ores of precious metals found in Canada it is well-known that their complexity of composition has been a deterrent to earlier profitable development of many deposits, but Canada today can boast of great strides

in the processes of recovering the precious metals. So marked is this feature of Canadian mining, that during the past year the members of the Canadian Mining Institute decided by a preponderating vote to change the name of the Institute as a recognition of the important status of the metallurgist.

To what conclusions do these considerations lead us? Inevitably, I think, and quite unmistakeably, to a recognition of the importance of applied science to our young nation, and if this conclusion be admitted, then I think it will be necessary for the Canadian people entirely to revise their valuation of the scientific worker.

The Canadian Mining Institute in common with our sister societies, has for years urged greater recognition of students of science, of the universities and institutions of scientific learning, and of those civil servants charged with the development of the resources of the mines, the forest, the farm, and the sea.

The salaries paid to those engaged in demonstrating in our Universities, and to civil servants engaged in scientific work, are so inadequate as compared with the rewards available by accepting employment with any of the large industrial companies in the country that the average scientific worker in our universities and in Government service is compelled to choose between carrying on at a great personal sacrifice the work in which he is engaged or relinquishing it and accepting employment outside his present duties which will at least be sufficiently remunerative to provide the necessities of life.

Among others, there is one branch of the Civil Service—I refer to the Geological Survey—which has never been appreciated at its true worth. No Canadian Government has yet voted a worthy appropriation for the purposes of the Survey. It has always been hampered in its work by an inadequate number of workers, inadequately paid; and this deplorable, but I think undeniable fact, arises from a fundamental misconception of its importance. The Canadian people do not know what advantages flow from applied geological research and, largely for this reason, they do not care. I suggest that here is a direction in which the Canadian Mining Institute can do useful work. We should fully consider the work of the Geological Survey and should present to the Government a memorandum of how we conceive its usefulness can be maintained and enlarged, and then back our recommendations by the entire influence of the Institute. Certain steps have already been taken and these steps should be supplemented to the fullest possible extent. Unless we ask for what we want, and ask plainly and urgently, we shall go wanting.

THE MINING AND SMELTING OPERATIONS OF THE INTERNATIONAL NICKEL COMPANY OF CANADA, LIMITED.

BY THE STAFF

Annual General Meeting, Toronto, 1920

All the ore treated at the present time by the International Nickel Co. of Canada, Ltd., is derived from the Creighton mine. The mine is situated six miles west of Copper Cliff—the site of the smelter and general offices of the Mining and Smelting Division—in the Sudbury district, Ontario.

Mining was commenced in 1901. The result of operations was such that within a few years from that date the pre-eminence of the deposit in point of magnitude and in the value of its metallic content had been clearly demonstrated, and this distinction of being the greatest nickel mine in the world is still retained by the Creighton.

GEOLOGY OF THE CREIGHTON MINE

The earliest records relating to the mine date back to 1856, when a surveyor noticed a strong deflection of the needle in the vicinity of the Creighton deposit. The locality was examined during the same season by Alexander Murray. Murray reported the occurrence of an "immense mass of magnetic trap" which he found to contain "magnetic iron ore and magnetic iron pyrites generally disseminated through the rock, the former in small grains; titaniferous iron was found in association with the magnetic ore, and a small quantity of nickel and copper with the pyrites."

Murray's description evidently refers to the norite, containing disseminated sulphides; and probably he did not observe the gossan covering the orebody at the foot of the ridge. Twenty-seven years passed without further discoveries, but immediately following the construction of the Canadian Pacific Railway in 1883, many finds were made throughout the district and within a few years practically every deposit exposed at the surface had

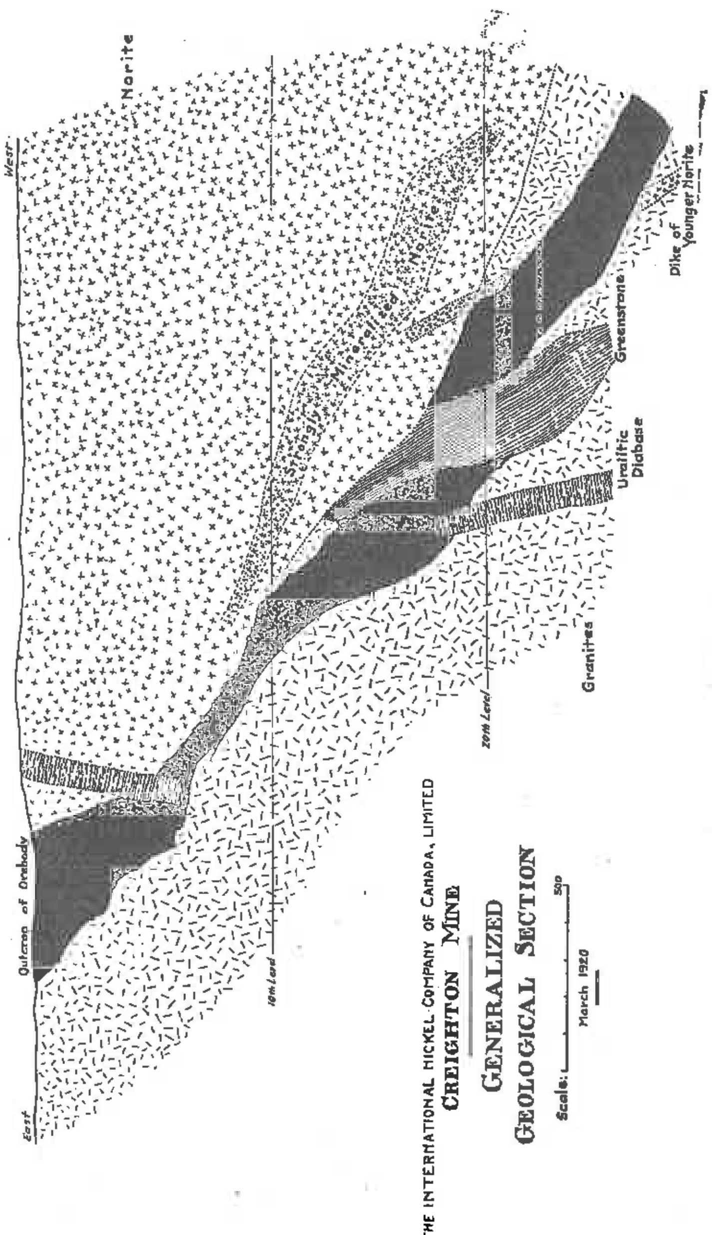


Figure 1.

been staked. The Creighton deposit was re-discovered in the fall of 1886, and in 1887 a Crown granc thereto was issued to the Canadian Copper Company, which had been incorporated in January of the previous year. In 1900, when the Algoma Eastern Railway was extended to the mine, stripping was commenced, and in August, 1901, the first shipment was made to the smelter at Copper Cliff. At the close of 1901 the rate of production was from 500 to 600 tons a day; during the war period it, at times, exceeded 5,000 tons a day. The total production to the close of 1919 was 8,874,780 tons.

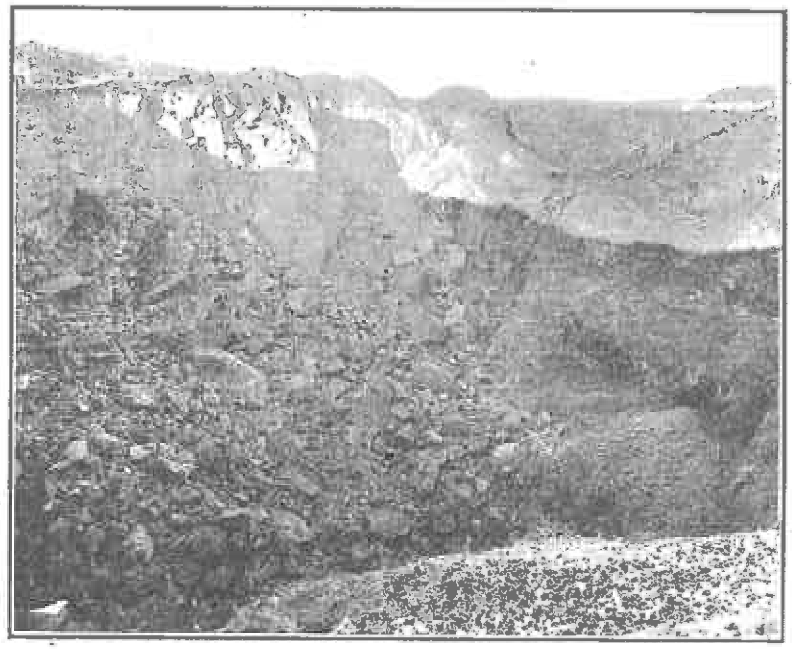


Photo by British & Colonial Press, Toronto.

Plate 1.—The open pit, Creighton mine.

The greatest extension of the main orebody is along the dip, the angle of which is about 45°. The margin of the orebody is usually sharply defined against the country rocks. The cross-section varies in form from oval to narrow lenticular, with wavy outlines, which, in some places, are strongly marked and irregular. The pitch length developed is over 2,500 feet. The level length varies from 1,000 to 400 feet. The horizontal width reaches the maximum of 375 feet with an average of over 100 feet. There is no gangue in the ordinary sense of the word, but inclusions of country rock in the orebody are abundant in many places. The proportion of rock to the sulphides, constituting the ore, is very uniform. The sulphides comprise pyrrhotite, chalcopyrite, and pentlandite, with minute grains of magnetite disseminated through them. The orebody is cut by several diabase dikes, and the largest of these, as the geological section shows (Fig. 1), exhibits a striking feature. The breaks in its

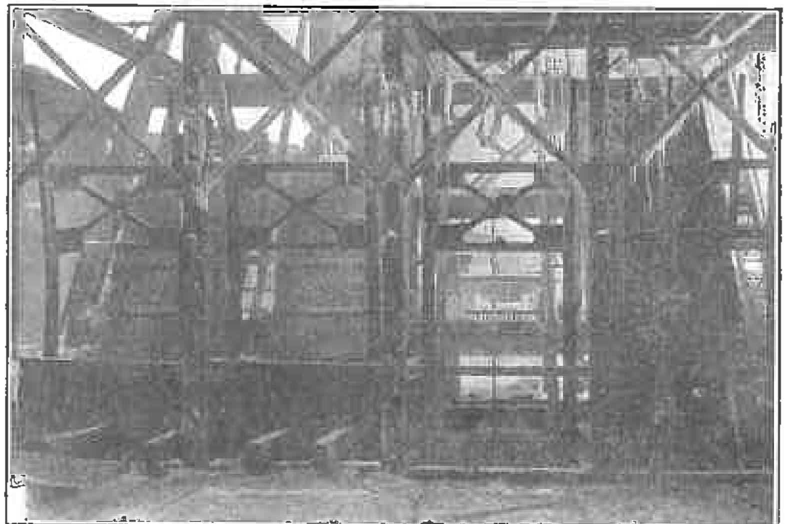


Photo by British & Colonial Press, Toronto.

Plate II.—Collar of No. 3 shaft, showing skips and man-cage.

continuity are not caused by faulting, but are due to the form assumed at the time of intrusion. This curious phenomenon is sometimes seen in small dikes that offset a short distance on coming to a soft bed lying between tougher ones.

Many writers have discussed the origin of the ore. The various theories may be summarized under two heads: (1) magmatic segregation; and (2) deposition from solution along zones of crushing and faulting. Both hypotheses, in their various modifications, have found ardent protagonists among both the early and recent investigators. Geological work at the mine has disclosed facts that indicate the origin of the ore by its intrusion in molten condition along a plane of shearing in the footwall rocks adjacent to the norite, after the latter had solidified. The most interesting evidence is afforded by a dike of a comparatively fresh younger norite that intrudes the main body of norite and the footwall rocks. The dike itself is intruded and altered by the ore. The alteration is an unusual variety of contact metamorphism. It appears as a dark margin, varying in width from a quarter of an inch to six inches, against the ore. It is also found in every rock with which the ore comes in contact, except in the diabase dikes which are younger.

METHODS OF MINING AT THE CREIGHTON MINE

The mine was first worked as an open pit, or glory-hole, from the surface. Later, when increasing depth made this system dangerous, underground methods were adopted.

From the open pit method, which was continued to a depth of 300 feet, the work has progressed through the various stages of underground quarrying, shrinkage stoping with dry-walls, and shrinkage stoping with under-drifts mainly in ore (in both of which round pillars were left supporting the hanging wall), to the present method of shrinkage stoping with main drifts in the footwall, crosscuts in ore, and the use of rib pillars. This system has been employed in all work below the 6th level.

A synopsis of the development and stoping 'layout' will show that the outline of the orebody has been approximately determined by diamond drilling and divided transversely into alternate stopes 60 feet, and of rib pillars of 15 feet wide. The shafts are in the footwall. Main haulage levels at intervals of 120 feet vertically, with drifts north and south from the shaft stations, are in the footwall at a convenient distance from the orebody. Crosscuts are driven through the orebody from footwall to hangingwall along the centre-lines of pillars, with box-holes at intervals of 15 feet on alternate sides. Intermediate drifts, with box-holes in the footwall, are driven as required. A continuous system of ore-passes carries the ore to central underground crushing-stations.

Three shafts have been sunk. No. 1, a three-compartment shaft, sunk at an incline of 59°, was used to handle ore during the earlier operations in the open pit and from dry-wall stoping; it extended to the 5th level only. This shaft has since been dismantled.

No. 2 shaft, of four compartments, was sunk at an incline of 47°; it extends to the 12th level. It was in full operation until 1917, but is now used for men and supplies only.

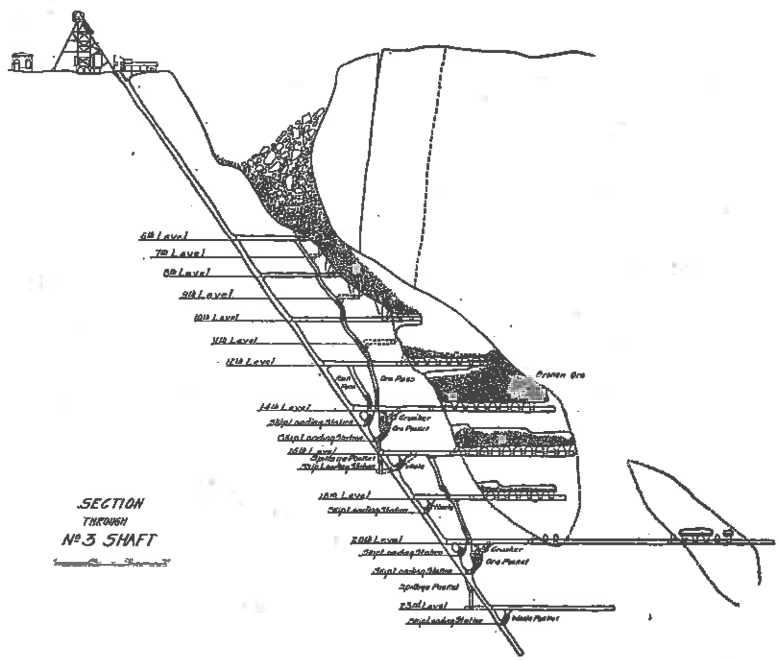


Figure 2.

No. 3 shaft was sunk at an angle of 55°. It is 33 ft. 2 in. x 7 ft. 6 in., outside timbers, and is divided into five compartments which are 5 ft. 10 in. x 6 ft. 6 in. in the clear. Sinking was commenced in April, 1915; and ore was first hoisted from the 14th level crusher-station in April, 1917. Two compartments are equipped for handling ore, two for men, waste rock and supplies, and one for ladder, pipes and electric cables. The shaft has since been extended 380 feet below the 20th level, a total depth, on the incline, of 1,941 feet. In sinking the upper part of the shaft 3⅛-in. piston drills were used. The centre V-cut was used, and rounds averaging seven feet were drawn. Below the 16th level, sinking was continued with Sullivan (D.R.6) machines, completing sections of approximately 200 feet by the use of a small auxiliary hoist and a rock pentice as protection from the operations above.

The sections through No. 3 shaft (Figs. 2 and 6) show in detail the method of shaft-timbering, pocket construction, and loading-station equipment. British Columbia fir is used in timbering. The wall-plates are 8 by 10 inches, dividers 8 by 8 inches, and studdles 4 by 10 inches. Concrete piers are placed across the shaft beneath every fifth wall-plate, and also beneath station levellers.

Ore-skips of 9-ton capacity are operated in balance. Rock-skips, cages and supply trucks are interchangeable and are operated in balance. Thirty men are hauled in each cage. Waste-rock skips are of 5-ton capacity.

Skip and cage tracks are 4 ft. 6 in. gauge. Rails (standard C.P.R., 85 lb. to the yard), are connected with angle splice-bars and Harvey grip-thread track-bolts with spring washers. Lundy tie-plates are placed on each sill and the rails are held by $\frac{9}{16}$ x 5½ in. spikes. One anti-creeper is attached to each rail. The entire absence of rail-creeping under conditions of heavy production has proven this construction most satisfactory. A 24-in. gauge track of 40-lb. rails is placed between the heavier rails in one of the cage compartments and is used to transfer haulage locomotives and cars.

Shaft rollers are made of 6-inch tubing 12 in. long. The spindle turns in brass bearings held in an iron frame. All parts are interchangeable. The rollers are placed 60 feet apart and are offset alternately two inches on either side of the compartment centre; this allows the turning of the rollers end for end in the frame as grooves are worn by the rope.

An air-main, 16 in. in diameter, is continued without change in size to the 20th level, and is held at its lower end, and again at the 16th level, by yokes resting on 18-in. I-beam bearers. Grooved blocks (6 in. x 8 in. x 3 ft. 10 in.), placed at convenient intervals on the footwall plates in the manway compartment, support the pipelines. The bend in the pipe at the shaft collar is held rigidly in concrete, and an expansion joint is provided near the surface. A 6-in. pump column and 4-in. fresh-water main are held by clamps through the wall-plates. Two 4-in. pressure reducers are placed in the water main, and a 2-in. reducer in each branch line.

Submarine power- and signal-cables are held on the end dividers. The power-cable is of 3-in. diameter, three-conductor, 400,000 circular mill, with 2,600-volt insulation, the average load being 175 amperes at 550 volts. The signal-cable contains 15 wires. Charging-cables for electric storage-battery locomotives are two-conductor, 400,000 circular mills. Mine lighting is obtained by using from three to five K.W. transformers on each station, fed from the power circuit. Stations are equipped with return electric-bell system, call buzzers, signal lights and telephone.

Ore is drawn from two loading stations, one below the 14th level, and the other below the 20th level. The stations are equipped with measuring boxes having capacities of 7½ and 9 tons respectively. The spillage pockets are cut below the loading pockets as shown on the sections through No. 3 shaft (Figs. 2 and 6). Shaft rails are carried across these pockets on 18-in. steel I-beams 32 ft. long.

Shaft stations are cut the same width as the shaft for a distance of 60 feet except on the crusher-station levels, where

the full width is maintained beyond the crusher. Waste-rock pockets of 100-ton capacity are placed with centres 52 ft. horizontally from the grade-line of the shaft.

To control ventilation and as a means of fire protection, a reinforced concrete wall 8 inches thick is placed 38 feet from the shaft across each station. Openings are left for protection against concussion due to blasting. These openings are covered with ⅜-in. steel plates swung on hinges. Double doors in the centre are made of ¼-in. steel plate lined with 2-in. planks. These doors are automatically opened and closed by means of a 4-in. by 5-ft. air-cylinder, the valve being conveniently placed for operation by the locomotive driver. A smaller door similarly constructed is placed near one end of the wall for use of workmen. Fire-hose and extinguishers are provided at convenient places behind the wall.

All timbering on stations, and for a distance of 25 feet above and below the station in the shaft, is fireproofed with a covering of expanded metal and gunite.

Air-mains, water-mains, and drainage launder are carried in a covered concrete conduit beneath and along one side of the station. Latrines are built into the reinforced concrete wall, against one side of the station, and are of the same construction as the main wall. They have concrete floors and drains, and are ventilated by a 12-in. pipe, which extends through the wall and a few feet into the upcast shaft.

Ore Passes.—A series of raises, 8 x 8 ft. in the footwall are driven between main haulage levels (Fig. 2). The raises are commenced from the sides of the station crosscuts, a brow point for the control gates is carefully determined and from this point continued to the level above, at an angle of 65°, forming a continuous ore-pass into which ore is dumped from the haulage levels and delivered to the crusher below. The ore is controlled on each haulage level by bent-finger rails and at the crusher by a baffle gate. All control gates are operated by air-cylinders.

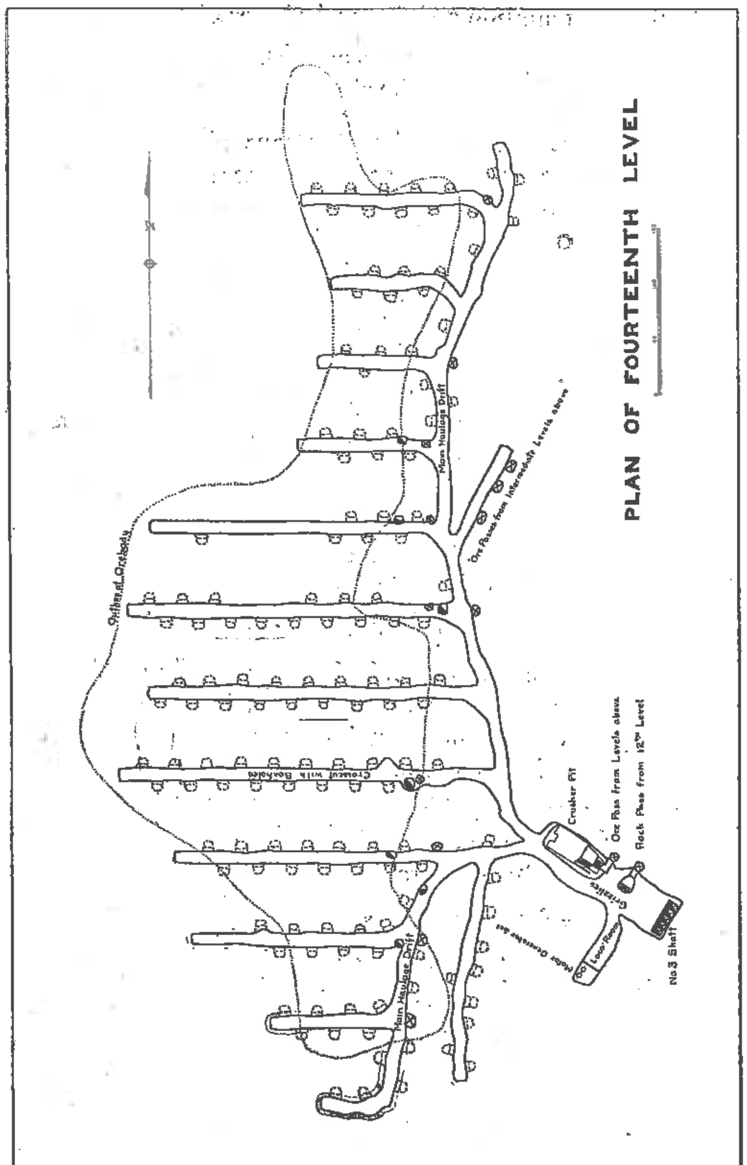


Figure 3.

Level Development.—Footwall drifts, or main haulage drifts, are run at a convenient distance from the orebody (from 25 to 35 ft.) with as few curves as possible. Crosscuts are turned off every 75 feet on the centre-lines of pillars and are driven to within a few feet of the hangingwall. Box-holes are raised from the sides of the crosscuts on alternate sides every 15 feet. In this manner a crosscut serves two stopes, with box-holes at 30-foot centres along the sides of each stope. Box-holes are also placed along the main drifts where necessary. Main haulage drifts and crosscuts are driven 11 ft. wide and 10 ft. high and give a clearance of three feet on each side of the tramcars. (See Fig. 3.)

In each pillar a manway raise is driven in the ore along the footwall contact from level to level, and equipped with ladder-way, pipe lines, and steel chute. These raises have the following functions:

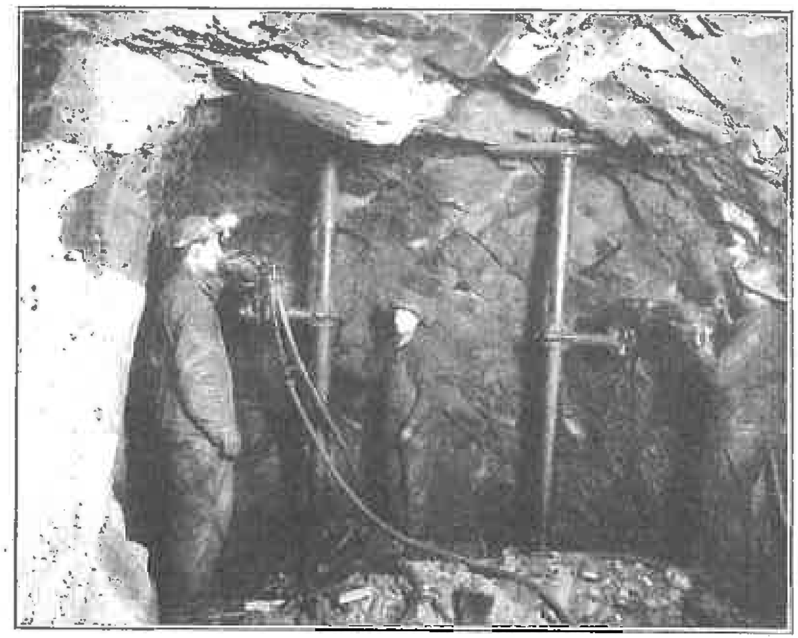


Photo by British & Colonial Press, Toronto.

Plate III.—Drilling in the face of a footwall drift.

(a) They form a systematic means of access to the stopes. As backstoping progresses, successive small openings are broken through from the stope on each side. Sharp steel is delivered through the chutes from the level above and dulled steel passed to the level below.

(b) They serve as a ready passageway between levels and are useful as an independent travelling way out of the mine.

(c) They form a simple and efficient means of ventilating the stopes and are of ample capacity for air currents to the lower levels.

(d) They assist in recovering the ore in pillars after the stopes are exhausted.

(e) They are used for compressed-air and water pipes from which branches are taken into the stopes.

In some parts of the mine intermediate levels are necessary owing to the flat dip of the footwall. These drifts are 9 x 9 ft. in section and are driven in the footwall, but nearer the orebody than are the main haulage drifts, the work being guided as to actual location of footwall by the manway raises previously driven. There may be as many as three intermediate drifts at intervals of 30 feet vertically between haulage levels, the length of each being governed by the local dip of the footwall. Boxhole raises are driven beneath each stope and also to the manway raises. The latter are not equipped with chutes until such time as the pillar is to be drawn. Ore- and rock-passes are raised from the main haulage level to connect with the intermediate drifts above, the rock-passes being used later as manways during the pillar-drawing stage. On the intermediate levels, tramming is done only at such times as the stoping machines are cutting back along the footwall, and at the end of the drawing-off stage, after the bulk of the ore has been drawn through the main-haulage-level chutes.

One of the lower levels has been equipped with a system of intermediate footwall drifts and branching ore-passes which are directly fed by boxholes in the drifts. By this method it is hoped to handle the ore without intermediate tramming.

Rock Drills and Steel.—The footwall granite, in which a large part of the development work is done, is hard and of rather coarse texture; it contains almost no fracture planes The greenstones and associated granites in the lower section of the mine are somewhat easier to drill and to break. The ore is easily drilled, but being rather difficult to break in the comparatively small development openings, requires as many drill holes as the rock demands.

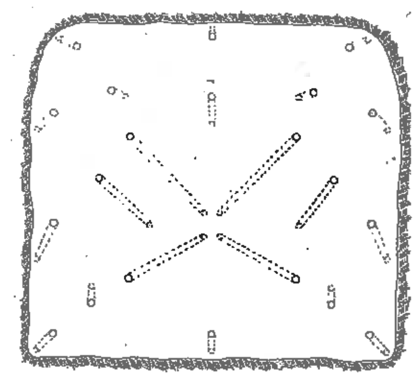


DIAGRAM OF STANDARD ROUND FOR 11'x10' DRIFT IN GRANITE

Figure 4.

The same rounds are drilled in all drift and crosscut headings, varying only in the number of 'easers.' The four-hole centre-pyramid cut is used, and from 17 to 22 holes, usually 19, are drilled in a round. Great care is taken to bring the cut-holes near the point of intersection, and to place properly the 'easers.' Cuts are blasted and enlarged separately, before the 'square-up' is blasted. Rounds varying in length from 6½ to 7½ ft. are broken. Polar 'forcite' of 40% strength is used for all classes of work. Experiments with different types and lengths of rounds have been conducted. Rounds averaging ten feet in length were broken during a period of 30 days with the same speed per machine-shift and less powder per foot driven, but the large amount of broken rock to be handled interfered with the cycle of operations.

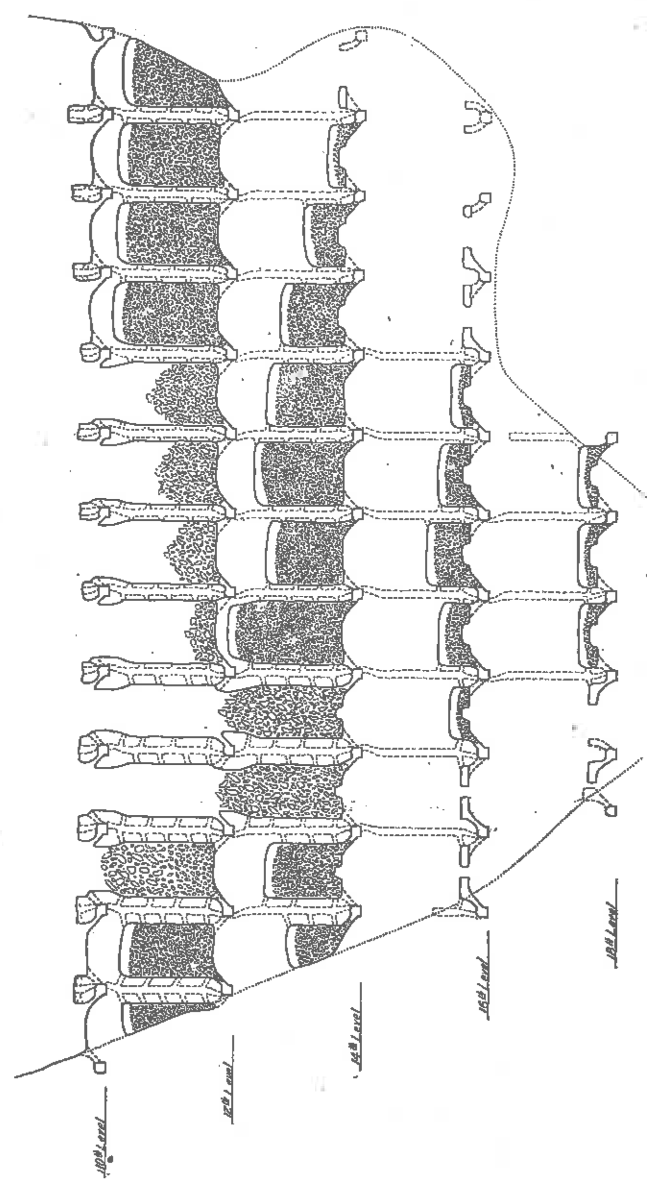
Figure 4 shows the positions of holes as drilled in tough granite to break a round of seven feet. By placing the cut-holes slightly lower and doing away with the two lower 'easers,' the standard 19-hole round in greenstone will clear $7\frac{1}{2}$ ft., to the bottom of the holes.

Two Sullivan (DR6) drills are used in each heading. The air pressure is 100 lb. Hollow hexagon steel $1\frac{1}{8}$ inches in diameter is used. Cross-bits, with 14° and 5° taper and reaming edge, are gauged $2\frac{1}{2}$ in. on 2-ft. 'starters' and decrease $\frac{1}{8}$ in. per foot to a length of 8 ft.; thereafter the decrease is $\frac{1}{16}$ in. per foot to 14 ft. and there is a difference of $\frac{1}{8}$ in. between the gauge of 14-ft. steel and that of 20-ft. As it is necessary to drill 16-ft. holes in the stopes, and as the bits of 16-ft. steel are $1\frac{5}{16}$ in. in diameter, this bit gauge is used throughout the mine in order to avoid confusion and to standardize shop work. Experiments are being made with small-gauge bits on development work, commencing with 2-in. 'starter' bits and finishing with a $1\frac{3}{8}$-in. bit at 10 ft.

A $7\frac{1}{2}$-ft. round in granite is drilled in one and one-half shifts, or three drill-shifts. During the period from July to December, 1919, the advance per drill-shift in drifts and cross-cuts was 2.3 ft., the consumption of powder averaging 19.4 lb. per foot driven.

Drills are tested on a granite block in the repair shop and must cut 3 in. per minute using a $2\frac{1}{2}$-in. bit.

Stoping.—The longitudinal section through the orebody (Fig. 5) shows the system of pillar-and-stope arrangement. The pillars extend throughout the mine from footwall to hanging-wall and are at right angles to the long axis of the orebody. In a section of the mine, from the 12th to the 16th levels, near the south end of the orebody, the pillars were increased in width to 25 ft. around the manway raises, with a corresponding de-



LONGITUDINAL SECTION THROUGH OREBODY

Figure 5.

crease in the width of the stopes. This change was considered necessary on account of the more friable nature of the ore, but it has not affected the essential feature of the narrow pillars, namely, the cheap recovery of the ore in the pillars more or less concurrently with the finishing of the stopes.

Floor Cuttings.—Headings 20 feet wide are first carried along each side of the stope at an elevation of 18 feet above the grade line of crosscuts. These headings are connected across the stope along the footwall and are extended toward the hangingwall by ordinary stoping methods from one boxhole to the next, the benches being so arranged that broken material is thrown by the blast into the nearest boxhole. No shovelling of broken ore is necessary. With the stope-floor heading seven feet high, the ore broken per drill-shift amounts to about 37 tons.

The next stage in the cutting of floors is accomplished by raising the elevation of the stope along the footwall and carrying a breast stope toward the hangingwall. As the under side of this breast is the roof of each of the two headings just described, the result is a clear space 60 feet wide with a ridge in the centre of the floor. All the broken ore possible is drawn through the chutes in advance of the breast, in order to keep sufficient head room for the next bench above. In stopes in which this stage of floor cutting is carried on, the duty per drill-shift averages 110 tons, making an average drill-shift duty of 80 tons for floor cutting in these stopes.

Back stoping progresses from footwall to hangingwall, carrying a bench from 10 to 12 feet high. Eighteen holes drilled horizontally from four separate set-ups, are usually sufficient to bring down a bench. Sixteen-foot steel is used. Broken ore is drawn in advance of the benches and a height of from six to eight feet below the back is maintained from the footwall to the working face. This gives comfortable head room and affords an easy opportunity for the inspection and scaling of the roof. As succeeding benches are carried forward it is necessary to cut upward along the footwall, and if this should be quite flat a somewhat lower duty per drill-shift is obtained.

Ore not broken fine enough to pass through the chutes below is blockholed. Rand (DDR13 and BC26) machines, fitted with chucks to take 1⅛-in. hollow hexagon steel, are used in this work. Incidentally, short ends of broken steel are utilized here. It is inevitable that a certain number of pieces of ore, too large to pass the chutes, are either buried in the blast or carried beyond reach of the blockholder, and it is necessary to blast these in the chutes later.

Photo by British & Colonial Press, Toronto.

Plate IV.—Back stoping.

Floor Removal.—When the ore has been broken to within a convenient distance from the floor of the stope above, the floor can be shot out any time after stoping above the next level is finished. A thickness of 25 feet is usually sufficient to maintain safe working conditions. To avoid the constant attention of scalers, which otherwise would be necessary in that section of the mine where the ore is softer or where a pillar shows signs of weakness, a few stulls of round timber resting on the broken ore are used to indicate possible movement in any part of the roof. These stulls are recovered before blasting. A section near the hangingwall is now thinned to about 18 feet, in which vertical holes are drilled and an opening blasted through. Careful attention is given to the placing of the vertical holes to ensure a clear break-through and its subsequent enlargement. The entire floor is then removed by retreating in stages towards the footwall.

Duty in Stopes.—Sullivan (DR6) drills are used. Holes are drilled to a depth of 10 feet on the floor headings. In breast stoping, 16-ft. steel is used, and for breaking down floors with vertical holes, 20-ft. steel is often required.

The drill-shift efficiency taken over six months, with drills about equally distributed between floor headings and back stoping, is 83 tons per shift of eight hours; material broken per man in stopes is 24 tons; powder per ton broken (which includes blockholing powder) is 0.4 lb. All drilling in stopes is done during the single day shift, the blasting crew coming in on the afternoon shift.

Pillar Removal.—Pillars may be broken after the floors have been cut through and the ore has been drawn sufficiently to allow the pillar to fall when blasted.

Holes are drilled in the back and sides of the footwall manway between two levels and, if necessary to free the pillar from the hangingwall, in the floor and roof of the crosscut above. These holes are blasted with ordinary fuse, and usually at one time. Two hundred and fifty may be taken as the average number of holes necessary to bring down a section of fairly solid pillar.

The probable caving of the hangingwall on the removal of pillars was foreseen, and as shown in the section through No. 3 shaft (Fig. 2), part of the stope floor was left along the footwall of the orebody at the 10th level. The outer edge of this floor leaves a safe margin beyond the angle of repose of

broken rock that might come from the area of flat-dipping hangingwall above. The last of the pillars above this point are in process of removal. Some of the pillars below the 10th level have been purposely weakened by cutting into them near the hangingwall. Some of these show signs of weakness above, but it is hoped to remove them without much trouble when the time comes, as the dip of the orebody is considerably steeper in this section.

Tramming.—On intermediate levels, and during the early stage of level development, side-dump cars of 16-cubic feet capacity are used, and are trammed by hand. All other tramming is done by storage-battery locomotives with side-tipping cars of 56 cubic feet capacity.

Ten Baldwin-Westinghouse 5-ton storage-battery locomotives are in use, each equipped with 66 Edison Type A-10 cells, having a capacity of 375 ampere-hours.

A 50-K.W. D.D. generator placed on the 14th level will charge four locomotives at one time at a normal rate of 75 amperes each; this generator serves the charging stations from the 14th to the 20th level. A generator set of equal capacity is placed on the surface and is used to charge the locomotives above the 12th level.

An example of the life of locomotive storage-batteries is afforded by that of a set after 2½ years' service. This locomotive is delivering 210 cars of ore to the tipple over an average tramming distance of 600 feet on one charge. The length of time necessary to re-charge is five hours. The average service of the cells is about three years.

Trains of seven cars are handled easily. Main-line tracks are uniformly graded 0.5 per cent., and branch lines in crosscuts 0.75 per cent. Haulage cars weigh 7,000 lb.

Car tipples are constructed of timber fitted with steel wheel-tread, cast in sections six feet long, the curved surface of the tread being so developed that the dump wheel of the car is at all times at right angles to the plane of the tread. (See Plate V.)

No grizzlies are used at the dumping points into the ore-pass that feeds the crusher on the 14th level, but as the largest pieces of ore that can be drawn through the stope chutes sometimes give trouble at the crusher, the lower ore-pass has been equipped with grizzlies. The grizzly rails are constructed of 12 x 16-in. timber 17½ ft. long, lined with 12-in. channel and 1¼ x 5-in. wearing plates. They are placed at an angle of 22° from the horizontal and spaced 28 in. at the upper end and

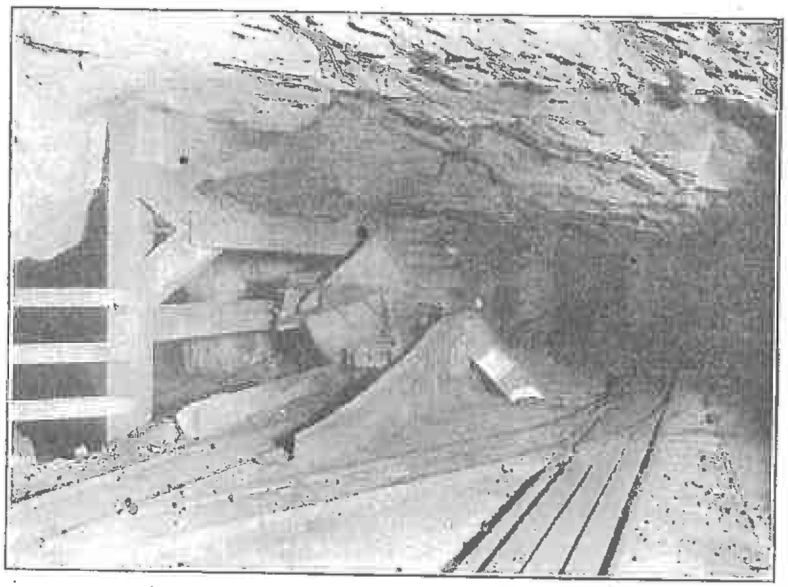


Photo by British & Colonial Press, Toronto.

Plate V.—Power tram-car dumping into an ore-pass.

30 in. at the lower end. A row of discarded crusher plates, placed on a 30° slope at the upper end of the grizzly, takes the shock of the dumping load. Oversize is delivered to a blasting station at the lower end, where the large pieces are blockholed.

Sledging grizzlies over the rock pockets are made of 85-lb. rails turned base upward and bent about four feet from one end at an angle of 27°. This forms a slope on which the broken rock slides away from the car track, the sledging station being about three feet below the grade of the track. The rails are spaced 10 inches apart and are held in place by wooden blocks made to fit the rails. Blocks and rails are held in place by a cover strip of ⅜ in. x 6 in. bolted through the bearer beneath.

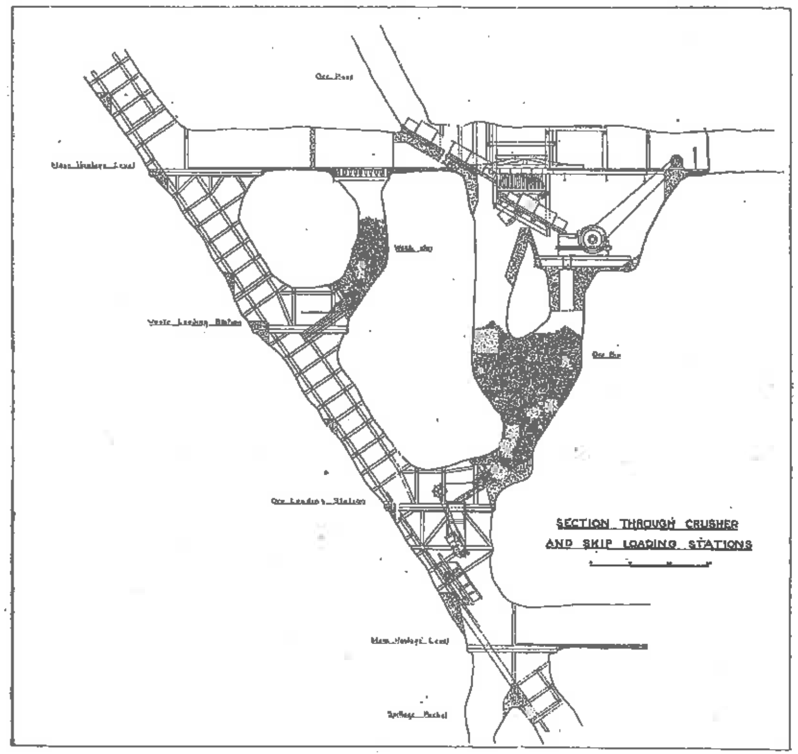


Figure 6.

The gauge of the track is 24 in.; 25-lb. rails are used on intermediate levels and 40-lb. rails for locomotive haulage. The minimum curve of 25-ft. radius is maintained on haulage levels. Curves at the entrances to crosscuts are accurately laid out by the foreman by the use of a chart giving the points for different angles of intersection. All ties are placed at two-foot centres.

The position of underground crushers is shown on the sections through No. 3 shaft (Figs. 2 and 6). Broken ore from the ore-pass and from the level on which the crusher is situated, is dumped over separate grizzlies (five-inch spaces) into a 30 x 42-in. jaw-crusher set at 6 inches, and the product falls into an ore pocket from which it is loaded into the skips. Two of these crushers are in use, one on the 14th and one on the 20th level; they handle all the ore above their respective levels.

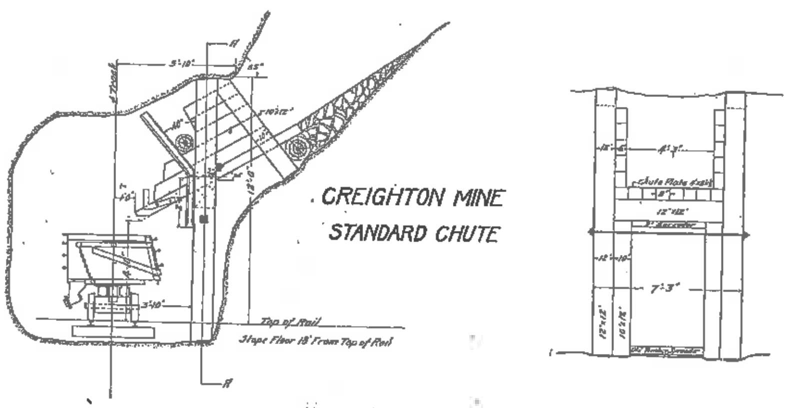


Figure 7.

The crushers weigh 65 tons, and base and fly-wheels are sectionized for passing through the shaft. One of these crushers has handled 5,000 tons in two shifts of eight hours each. Water and grease are used to lubricate the pitman and journals, and cup grease is used for the jaw shaft. Waste water from the lubrication is collected in a sump and pumped to the drainage launder above. The crushers run at a speed of 235 R.P.M., and are connected to two 100-H.P. induction motors. One motor would furnish sufficient power to drive the crusher under normal load, but owing to the large starting torque required

two motors are used; these also ensure constant speed under different conditions.

Stope Chutes.—Chutes are shown in the drawing of the standard chute (Fig. 7). The essential features of the chute are height and location of the brow point upward to the stope floor, forming a funnel in which the larger pieces of ore can

Photo by British & Colonial Press, Toronto.

Plate VI.—Loading tram-car from a stope chute.

jamb near the lower end only. Careful attention to these details has lessened accidents from chute blasting, and has improved the tramming efficiency.

Extending over a period of six months the amount of material trammed, including chute drawing and development shovelling, is 38 tons per trammer shift for power tramming, and 20 tons per shift for hand tramming. Material hoisted per man underground is 7 tons. Powder used in chute blasting is from 0.2 lb. to 0.3 lb. per ton trammed.

Pumps.—Seepage water from the surface is practically all that is encountered in the mine, aside from the fresh water brought down for rock-drills and drinking. During the rains and thawing periods, however, this becomes considerably more.

Two main pumping stations are provided, one on the 6th level, No. 2 shaft, and the other on the 16th level, No. 3 shaft, both pumping directly to the surface.

Equipment on the 6th level consists of a Gould vertical 3-plunger pump geared to a 40-H.P. motor. The plungers are six inches in diameter and have a 12-in. stroke, the capacity of the pump being 250 gallons per minute, against a head of 457 feet. This is supplemented by a three-stage centrifugal pump and an air-driven plunger-pump, which are held in readiness in case of accident or during the wet season. The working time of this pump varies from three hours per day in the winter to full time during the first rains in the spring.

The equipment on the 16th level consists of a Gould pump of similar type, but of heavier construction. It is direct-connected to a 100-H.P. motor; the plungers are 6½ in. in diameter and have a stroke of 16 in. Suction and discharge diameters are 8 in. and 6 in. respectively; the height over all is 10 ft. 3 in., and the capacity is 250 gallons per minute under a working head of 1,035 ft. The working time varies from six hours in the winter to 18 hours in the spring. The mine water passes through settling tanks placed between the launder discharge and the suction pump. These tanks are concrete lined and provided with by-passes to facilitate cleaning. The capacity of the sump is 17,000 gallons.

A small motor-driven vertical-plunger pump will be installed on the 23rd level. Small air-driven Cameron pumps are used in the various parts of the mine.

Ventilation.—Natural ventilation supplies ample air for the deepest workings. The two shafts and a winze that connects with the stopes at the south end are up-cast; and the difference in the elevation between these openings on the surface and the bottom of the open pit, which is 300 feet deep, maintains a strong upward current of heated air. Short circuits to the shaft and between the north and south ends of the mine, above the 12th level, are prevented by doors in drifts and crosscuts.

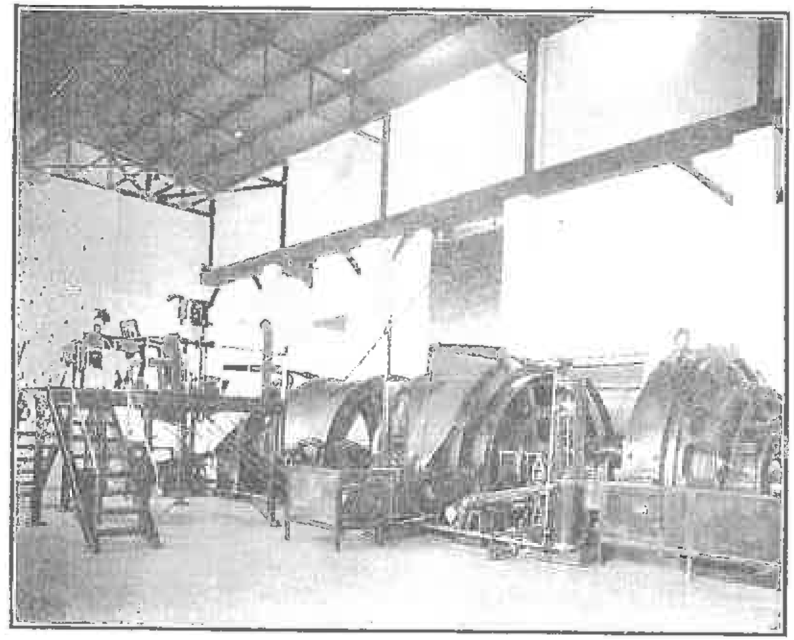


Photo by British & Colonial Press, Toronto.

Plate VII.—The ore hoist.

Below the 12th level the footwall manways become down-cast, owing to the bulkhead walls on the stations, the air current passing to No. 3 shaft through the lowest level. Propeller fans were placed temporarily in the station bulkhead walls on the 18th and 20th levels, but as development proceeded and more manways were opened, they became unnecessary.

Hoisting.—As all the mine product comes through the No. 3 shaft, every effort has been made to effect rapid and safe handling of ore, rock, materials and men. The ore is hoisted to the rock-house in 9-ton skips, working in balance, through the first two compartments of the shaft, and the waste rock is hoisted through the third and fourth compartments to a rock-storage bin standing in front of, and lower down than, the ore dump. Men and materials are handled in steel man-cages, working in balance in the third and fourth compartments, the skips being replaced by cages or *vice versa* as required. The changing of cages and skips is done quickly by storing the skips on a balcony above the collar and the cages at the collar and transferring from either collar or balcony track to shaft-track by jump-rails which are swung into position by small air-hoists.

The ore hoist is of the Ilgner type and was built by the Wellman-Seaver-Morgan Company. The drums are 12 ft. in diameter and 7 ft. 6 in. wide, having a capacity of 2,100 feet of 1½-in. rope in one layer. One drum is keyed to the shaft, whereas the other is loose on the shaft and is provided with a multiple-tooth clutch, operated by an oil cylinder, the oil for which is supplied by the accumulator by means of a lever on the operator's platform.

Each drum is fitted with a post brake, set by weights and lifted by a brake engine operated by oil which is supplied by the accumulator. The clutch and brake are interlocked so that the clutch cannot be thrown out until the brake is applied, or the brake released until the clutch has been thrown in.

The hoist is provided with a mechanism for automatic acceleration, and an automatic slowing and stopping device.

The accumulator equipment consists of a weighted accumulator having a capacity of 16 cubic feet, duplicate three-throw oil-circulating pumps with direct-current motors, also the necessary return tank and piping, connecting the brake cylinders with accumulators and return tank.

The hoist motor has a normal rating of 1,800 H.P. continuous at 550 volts and 40° rise, operating at a speed of 66½ R.P.M., and having a maximum rating of 3,600 H.P.

The motor generator set consists of a direct-current generator, a wound-rotor induction-motor, a steel-plate fly-wheel and a direct-connected shunt-wound exciter, all mounted on a common bed plate.

The separately excited direct-current generator has a capacity of 1,500 K.W. based on a 40° rise.

The three-phase alternating current, wound-rotor induction-motor has a normal continuous capacity of 1,400 H.P., and is designed for a frequency of 25 cycles and 2,400 volts. It is wound for six poles and has a full normal-load speed of 487 R.P.M.

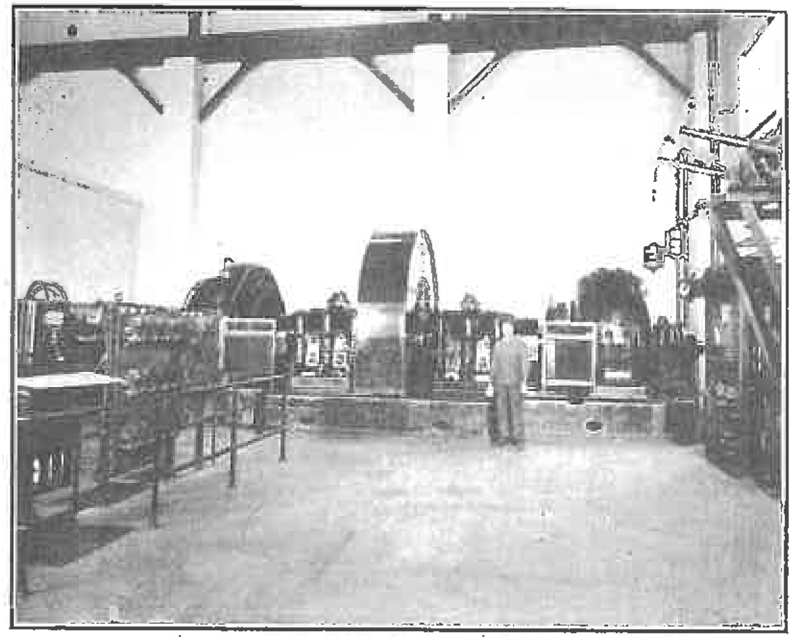


Photo by British & Colonial Press, Toronto.

Plate VIII.—The motor generator set.

The steel-plate fly-wheel is 12 ft. in diameter, 21½ in. wide, and weighs 100,000 lb.

The direct-current inner-pole, shunt-wound exciter has a normal rating of 30 K.W. 40° rise, operating at a speed of 500–415 R.P.M. at 250 volts.

The hoist is designed to operate under the following conditions:—

Net weight of ore hoisted	Lb.	18,000
Weight of skip	Lb.	11,000
Weight of rope, per foot	Lb.	3.5
Inclination of shaft		55°
Depth of shaft	Feet	1,800
Output in 7 hours	Tons	3,500
Output per hour	Tons	500
Hoisting speed	Feet per minute	2,500

Cycle—10 seconds for loading, 15 seconds for acceleration, and 10 seconds retardation.

The cables in use on this hoist are of 1½-in. diameter, six-strand, 16 wires each, 9, round 6, round 1, plough-steel, Lang's lay with a breaking strain of 212,000 lb. Those now in use have hoisted over 606,000 tons per rope.

Rock and Man-Cage Hoist.—This is a Nordberg, two-drum hoist, and is driven through one reduction by a 350 H.P. Allis-Chalmers slip-ring variable-speed motor (alternating-current) rated at 480 R.P.M. at full load. The drums are 7 ft. in diameter with a 4-ft. face and each is fitted with parallel-motion post-brakes. The brakes are set by means of weights and released by oil cylinders. There is an automatic cut-off which operates in case of over winding or lack of current. A liquid rheostat, which gives a smooth and certain control of acceleration to the hoist is used.

The clutch is of the axial friction type, set radially, and the motion of the movable parts of the clutch is always parallel to the axis of the drum. The hoising speed is about 1,100 feet per minute with a 5-ton load of rock, and the hoist is designed for a rope pull of 21,700 lb.

The cables are 1¼-in., 6-strand, 16-wire, plough-steel, Lang's lay with a breaking load of 138,500 lb. All cables are supported by 6-in. idlers throughout the shaft, and by fleet wheels between the sheaves and the hoist, which is 140 feet from the shaft. The head sheaves are of the bicycle type, 12 ft. in diameter with 4⅜-in. rope-groove, and are carried on 12-in. journals 142 feet above the collar of the shaft.

The head frame is combined with the rock-house, which stands 71 ft. from the collar of the shaft.

The four shaft-tracks of 85-lb. C.P.R. standard rails and 4½-ft. gauge, extend from the collar of the shaft to the ore-dump in the rock-house and are supported by 20-in. I-beams throughout. The back legs are constructed of angles and plates and are anchored to concrete piers.

All four tracks from the mine pass over the rock-dump which is placed over a waste-bin situated about 20 feet in front of the rock-house and at a lower level than the ore-dump. This bridging of the rock-dump opening is arranged by sliding-rails which move in and out of line with the main track rails by means of a shuttle worked through a lever operated at the collar of the shaft. There is 47 feet of head room between the ore-dump and the sheave.

Ore-Skips.—Nine-ton, all-steel skips, with a capacity of 168 cubic feet and weighing 12,200 lb., are used for ore. These are 13 ft. 4 in. in length over-all, 3 ft. 11 in. in cross-section and are made up of ⅝-in. outside plate, ½-in. lining plate, and the whole is braced by ample angles and channels. The end and bottom are interlined by 2⅝-in. hardwood to absorb the shock, and ½-in. x 6-in. manganese-steel bars are used to take up the wear of the sliding ore. The 16-in. wheels are of manganese steel, the front wheels having a 3-in. tread and the rear 6-in. The axles are of 4-in. forged steel and are carried in tight housings, which extend the full length and ensure ample space for lubricants. The bail is 15 ft. 2 in. long, pivoted 3⅞ in. behind the end of, and 12 in. above the bottom of, the skip: it swings on a 3-in. shafting with ample bearings and reservoir for lubricants. The bridle bar is of 6 x 1-in. steel bars while the yoke is made up of heavy plate.

Rock-Skips.—These are of the same sturdy all-steel construction, being 11 ft. long, with a 6-ft. 6-in. wheel base; they have a capacity of 85 cub. ft. and weigh 9,000 lb. The bail is pivoted 3 in. behind; and 10-in. above, the botton of the skip-body, ensuring complete emptying, as in the case of the ore-skips. The wheels are also of manganese steel and of the same size as those on the ore-skips.

Man-Cage.—This is of ⅛-in. steel plate riveted to trucks and weighs 8,800 lb. empty. The wheels are of manganese steel and the draw-bar is of Lowmoor iron annealed at frequent intervals. A screen door slides up and down, the weight being balanced by a spring roller, thereby entirely enclosing the men in a steel car.

Signals, Safety Appliances and Inspection.—The shaft, which is inclined at an angle of 55°, has no back-runners, and no chairs are used with cage or skips. The hoist, sheaves, cable, and shaft are inspected daily. A warning bell is installed to indicate to the hoistman when the skip is approaching the surface and he accordingly slows down. The load of the cage when carrying men does not exceed 85% of the maximum weight of the other loads. The hoists have the ordinary safety appliances. The bell signals for hoisting and lowering are on the electric return-bell system. The switches are pull-type and the bells are rung simultaneously on all levels, collar of shaft, and hoist-house, the same signal being repeated by the hoistman for verification.

Operation of Hoists.—A Johnston and Johnston trip-recorder gives a continuous record of the trips of each hoist and some wonderful charts have been obtained from the ore hoist during 16 hours' operation. The best record has been 412 skips in 8 hours from the 14th level ore-pocket, 1,350 feet below the dump, with a load of 7½ ton of ore. The average hoisting rate, however, is about 46 skips per hour witb a 7½-ton load

from the 14th level, or 5,520 tons in 16 hours. This hoist has never been operated to its capacity.

Waste-Bin.—The waste is hoisted in the third and fourth compartments to a 22-ft. x 36-ft. (diameter) steel receiving-bin, holding 1,100 tons. This is either discharged through arc-gates directly into railway cars below, or else through finger-gates, on the side, into 56-cu.ft. cars to be hauled by electric locomotives to the open pit for disposal.

Rock-House.—This has been designed to handle a large tonnage and provide plenty of room for the required number of pickers. The different-sized particles are screened out and treated separately on different floors. The building, which has three picking floors, is of steel and brick, with cement floors, and stands on concrete pillars directly over the railway track on which the receiving cars draw the sorted product from the storage bins. The hopper-bottomed receiving-bin, into which the entire ore product of the mine is dumped, is placed at the top of the building. From this the ore is drawn off through a gate controlled by rack and pinion; it passes over two feeding rolls of 5-ft. diameter, which turn at a constant speed of about one revolution per four minutes undercutting the stream of ore. The ore then goes by gravity directly into the trommels (60 in. x 8 ft.) on the upper floor, which are inclined at an angle of 8° and are revolved at the rate of 10 R.P.M. by 25-H.P. back-geared Westinghouse motors. The under-size from the 6-in. openings in these screens goes directly to the second screen on the next lower floor. The over-size on the upper floor is discharged over cast-steel plates, inclined at an angle of 27°, onto 36-inch Robin's rubber conveyor-belts travelling past the pickers at a rate of 35 ft. per minute. The waste rock is taken out by the pickers and dropped down conveniently situated steel chutes which pass through the building to a sorted-rock storage-tank above the railway tracks to be drawn off into railway cars; or the rock can be collected in cars on an intermediate floor for disposal in the open pit. The sorted ore remaining on the belt is carried to the end where it drops into crushers, 48 in. x 15 in. set at 3½ in. The product from the crusher is delivered into a screen, 48 in. in diameter, inclined 8°, and revolving at 12 R.P.M. The fines, which pass through its ¾-in. openings, are separated from the coarse by this screen. Both products are now finished and drop into a 2-compartment steel storage-bin 50 ft. in diameter and 23 ft. high.

2nd Floor.—The under-size, which passed through the 6-in. openings of the trommel on the upper floor, drops through hoppers into trommels (48 in. in diameter and 10 ft. long) on the second floor which are also inclined at 8° and revolved at 12 R.P.M. by 25-H.P. motors. The openings in this trommel are 2½ in. in diameter and the under-size goes direct to the floor below, to be sorted, while the over-size goes to the sorting belts where the sorted ore is carried over the end as a finished

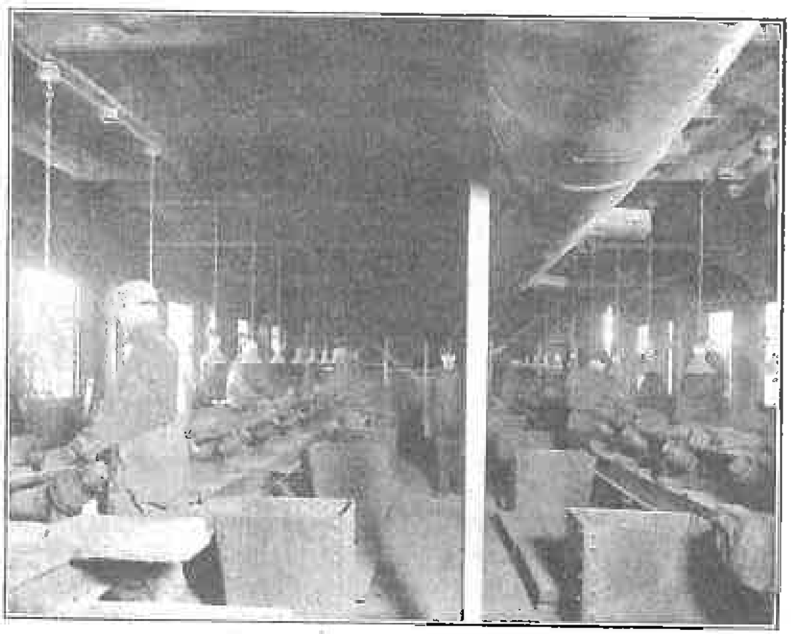


Photo by British & Colonial Press, Toronto.

Plate IX.—Upper floor of rock-house.

product and drops into the same steel storage-bins into which the ore from the upper floor was discharged.

1st Floor.—The under-size from the second floor is again screened in a trommel (48 in. in diameter and 12 ft. long) driven by a 10-H.P. motor at 11 R.P.M.; this, however, is a double screen with holes in the main section 1½ in. in diameter. The over-size goes to sorting belts and the sorted ore goes to the coarse-ore storage-bins, while the under-size is again sized by another screen outside of the main one. This outside section is 7 ft. long, has ⅞-in. holes, and the over-size and under-size (which are the middlings and fines) are finished products and are stored separately.

As the above is a general description of the sequence of operations, attention may be drawn to a few matters touching design and efficiency:

As the crushing is done underground, the sizing performed in the rock-house is to aid picking and to ensure proper sizes and the proper separation of these sizes for reverberatory and blast-furnace products and for building the roast-heaps. The products shipped are termed 'coarse,' 'middlings' and 'fines,' and the proportion of each is 55%, 10% and 20%, respectively, of the hoisted ore; the remaining 15% constitutes the sorted rock. The underground crusher is set at a 6-in. opening, while the rock-house crushers are set at 3½ inches. The rock-house comprises two units, either of which can be operated independently as the output demands. The efficiency in sorting is rated by tons of rock picked per picker shift of eight hours and averages about 12 tons.

All trommels are inclined at an angle of 8° from the horizontal, and the discharge chutes to the belts are at from 27° to 29°, depending upon the fineness of the product. To allow for the 'riding up' of the over-size, all trommels are set 14 in. off the centre-line of the conveyor belts on to which they feed. All conveyor belts are 36 in., 5 x 7-ply Robin's, supported on 5-section rollers set at 4-ft. centres; they have about 60 ft. of picking length on each. These are driven at a speed of 35 ft.

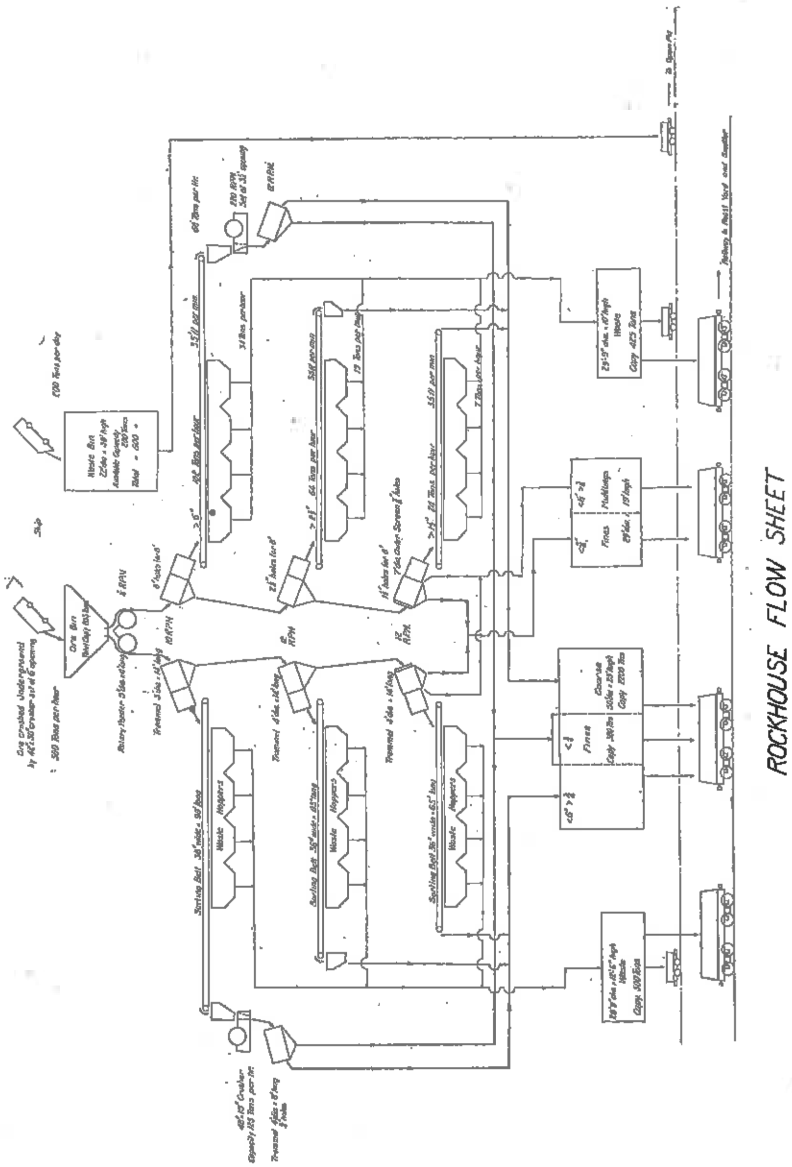


Figure 8

per minute by 5-H.P. constant-speed motors through James speed-reducers having a ratio of 705 to 5.6.

The belts on the upper floor have each carried about 500,-000 tons and are still in service. It might be added that the design of the rock-house has been so satisfactory in every detail that nothing has been changed in it since it was put into operation two and a half years ago. The flow sheet (Fig. 8) shows the 'layout' of the rock-house.

Steel Sharpening.—As about 1,200 pieces of steel are handled daily in and out of the shop, convenience in working therein has been carefully planned. It will be seen by referring to the plan of the blacksmith shop (Fig. 9) that the tracks are laid through the building so that the dull steel is brought into one side of the building on trucks, and deposited on racks in front of the five heating-furnaces. The heated steel is carried by the furnace operator to his punch. He punches the hole and hands it to the sharpener to form the bit, gauge the steel, and deposit it in the waiting truck on the other side of the building. These trucks are conveyed back to the tempering and grinding room where the bit is tempered, shanks are ground, and the finished steel is again placed on trucks ready to be sent underground for storage on the various levels until needed. There are five sharpening machines in operation, and each has its own rack, furnace, and punch, thereby making a complete sharpening unit. Each sharpener handles only certain lengths of steel, and thus minimizes the changing of dollies and gauging-blocks. All of the steel is re-heated in a single furnace and tempered by plunging into a tank supplied with running water. The shanks are tempered in oil, the oil being cooled by pumping it through a multiple-tube inter-cooler and also by jacketing the oil tank in running water. Three Sullivan and two Leyner types of sharpeners are in use, and a third Leyner is used when the amount of steel exceeds the capacity of the other five. A sharpener and his furnace-man average 250 pieces of steel for the eight-hour shift and all the tempering can be done quite readily by two men. At present, 1,200 pieces of steel are handled in an 8-hour shift.

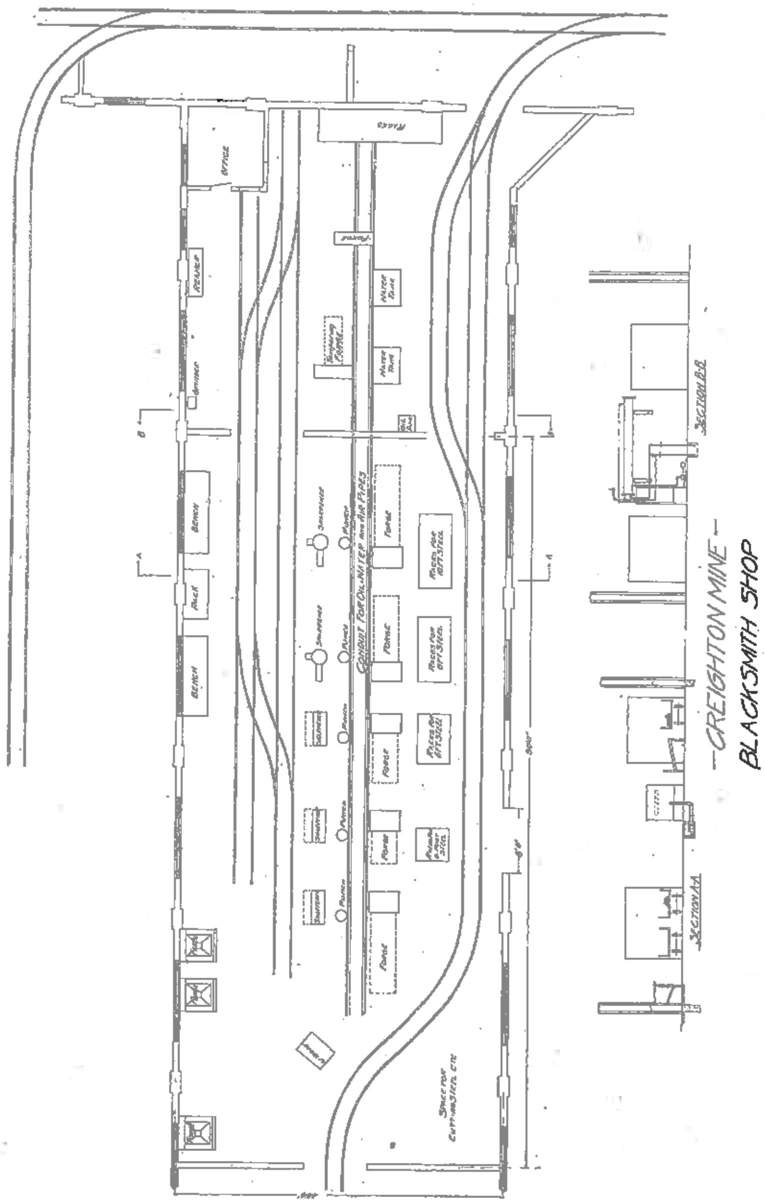


Figure 9.

The crude-oil furnaces are of local design, and are made of brick and steel; they have the usual needle valve, with ample air supply and compressed-air inlet. The ignited gas traverses the combustion chamber and, striking the opposite end of the furnace, is deflected up into the reverberatory chamber immediately above, and through the side of which the steel is thrust for heating. The waste gas passes out of the furnace of the top, and at the same end at which the oil entered. The amount of oil consumed is about one gallon for every eleven pieces of steel put through the shop; this includes the two heats per steel.

These furnaces have proven very satisfactory, and the repairs only amount to re-bricking from time to time as some slag forms in the combustion chamber and the fire-bricks are eaten through. To a furnace running steadily eight hours per day, this happens about once in every two or three months.

The wear on the bits, a cross-type witb 14° and 5° taper, is heavy in the machines drilling in the granite; hence, in the shop, a great deal of dollying is required to form the bit. The aim of the shop is to produce the greatest amount of steel that can be well formed and tempered, and to make each operator responsible for certain steel put through.

General Blacksmithing.—Three forges are in use and are served by a steam-hammer, power-shears and punch. Another small shop off the collar-house for skip repairs, etc., is in operation. The mine cars are repaired in a shop at the end of the collar-house.

Other Shops.—The situation of these important service departments can be seen by referring to the accompanying plan showing the general surface arrangement of the buildings (Fig. 10). The machine shop is conveniently situated and has modern equipment comprising lathes, radial-dril press, shaper and planer, bolt machine, pipe threader, etc.

The carpenter shop is small but is sufficiently well equipped with machines and tools for the proper handling of the work required of it.

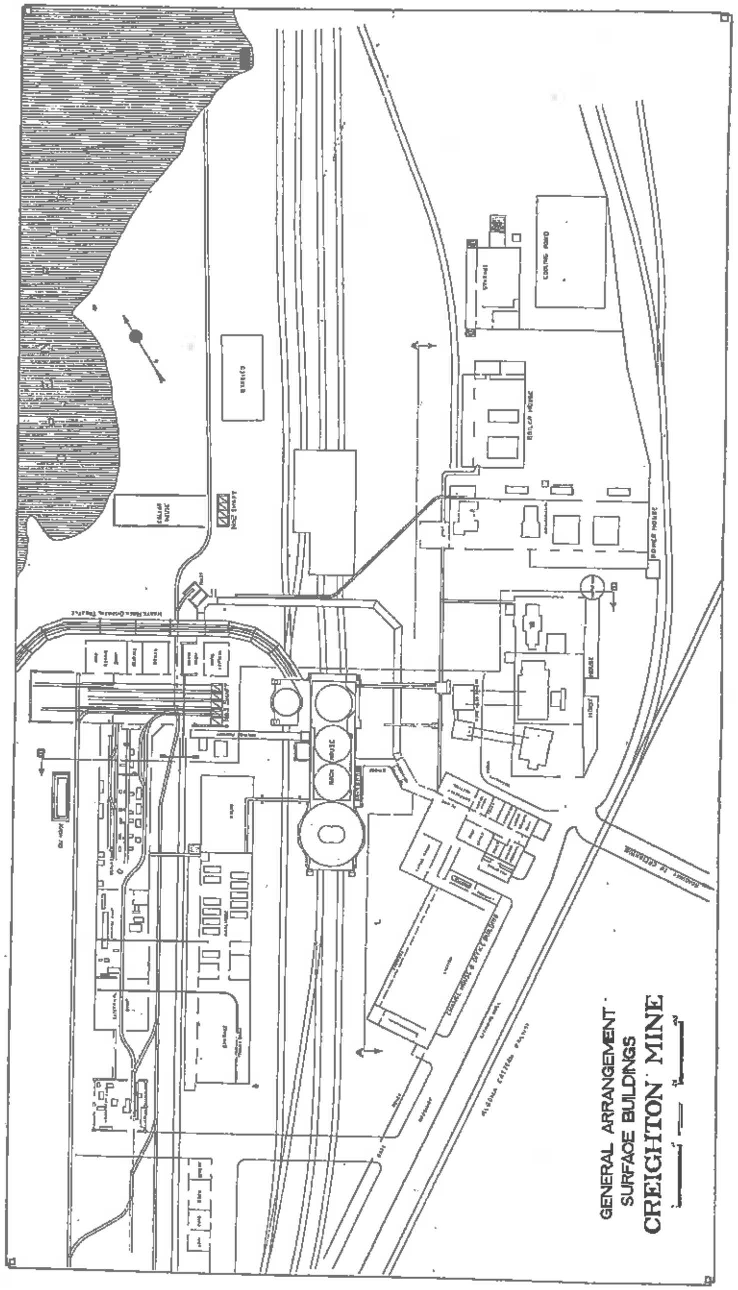


Figure 10.

Warehouse.—The materials and supplies at the mine are stored in a large and well designed building which is served by railway and narrow-gauge tracks at its door. Overhead crawls unload the heavier materials and the overhead track extends into one section of the building and throughout its length.

Powder Magazine.—About 1,000 feet from No. 3 shaft, a tunnel has been driven into a hill and a room of sufficient size excavated to hold 2,500 cases of dynamite. Another room has been made off the tunnel to serve as a thawing-room. This is heated by hot water coils, the water being heated outside by electric heaters. A railway siding leads to the portal of the tunnel, and mine-gauge tracks are laid so that a truck of dynamite can be loaded in either the storage or thawing room and conveyed to the shaft. The tracks in the rooms are of hardwood, the doors are of steel plate, and the electric lights are enclosed in air-tight glass globes. Recording thermometers give the daily temperature record. A powder-man, who keeps the stock records, is always in attendance.

The cap house is of cement and brick construction and is about 500 feet distant from the powder magazine.

Air Compression.—Three air compressors placed in a building by themselves compress the air for use in the mine. These

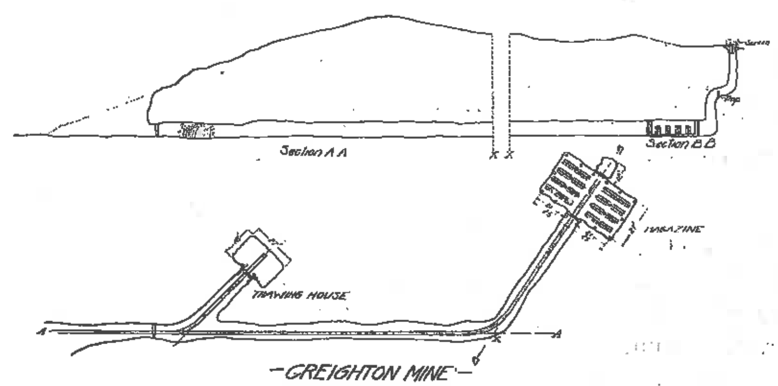


Figure 11

have a capacity of 5,000 cu. ft. each; one is vertical and was manufactured by Bellis & Morcom, and the other two, which are exactly alike, are horizontal, being manufactured by the Ingersoll-Rand Company.

The one made by Bellis & Morcom is a 50-drill, electric-driven, two-crank, two-stage, vertical, enclosed self-lubricating compressor, direct-connected to an auto-synchronous motor, manufactured by the General Electric Company of Sweden,

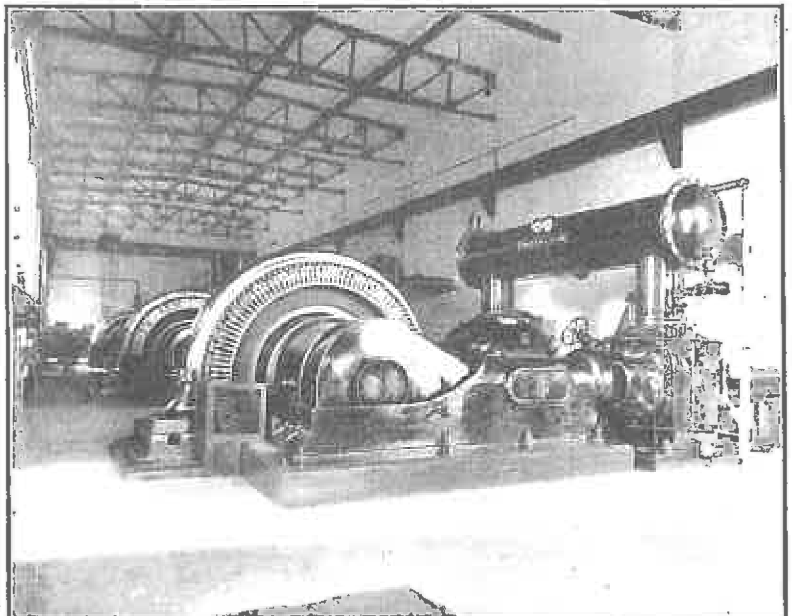


Photo by British & Colonial Press, Toronto.

Plate X.—Interior of compressor building.

and rated at 900 K.W., 2,400 volts, 3-phase, 25-cycle, 187 R.P.M. This compressor is designed to compress approximately 5,000 cu. ft. of free air per minute, at 975 ft. above sea level, to a final pressure of 100 lb. per square inch.

The Ingersoll-Rand 50-drill compressors, are Ingersoll Roegler, class 'R.P.E.2,' duplex, electric-driven, horizontal,

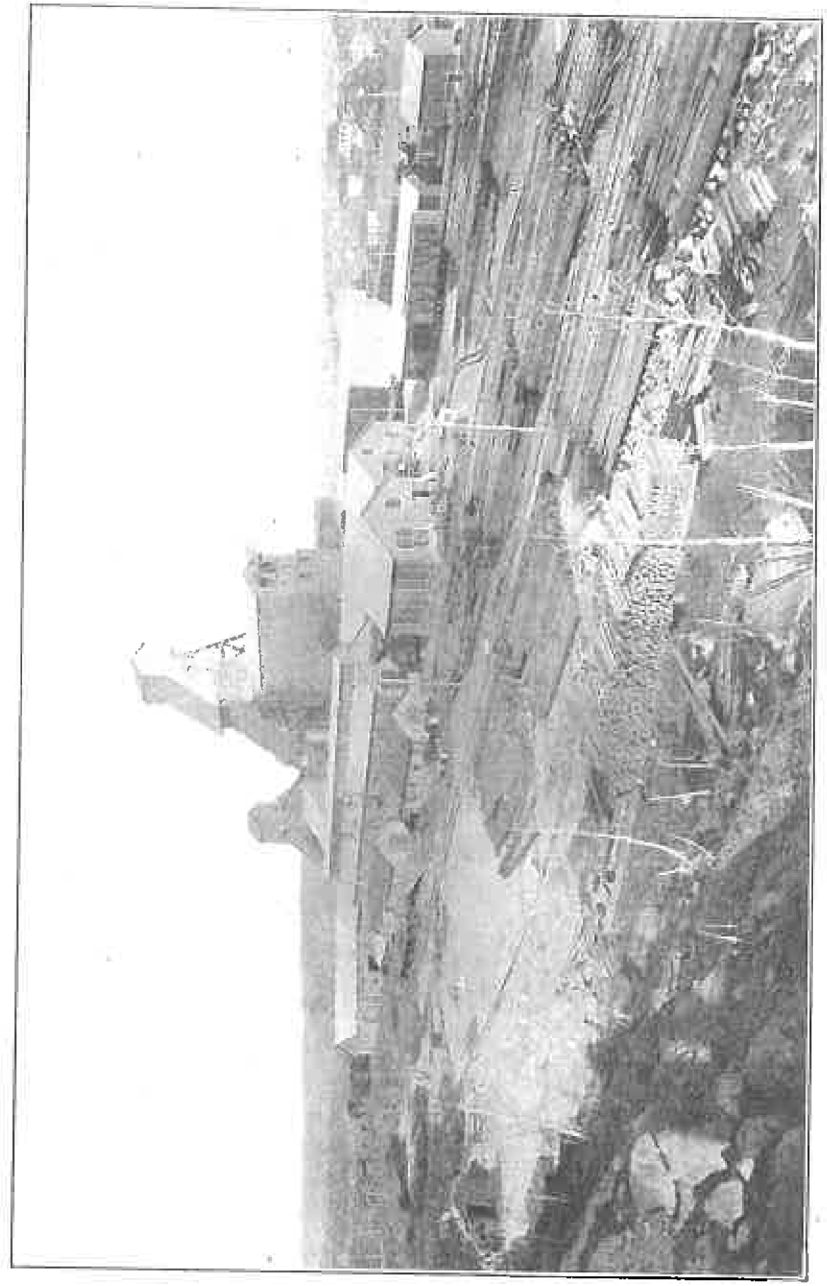


Photo by British & Colonial Press, Toronto.

Plate XI.—Surface plant and buildings, Creighton mine. Timber yard in foreground.

cross-compound, two-stage, direct-connected to a Westinghouse self-starting, sychronous motor rated at 932 H.P., 923 K.V.A. 2,400 volts, 3-phase, 25-cycle, 136.3 R.P.M. Each compressor is designed to compress approximately 5,000 cubic feet of free air per minute to a final pressure of 100 lb. per square inch.

The intakes to each of these are from wire-enclosed boxes standing about 10 feet above the ground. The diameter of the intake pipes is 24 in., and vents are left at joints of pipe-lengths to allow for the backing up of the stream of air when the valves automatically cut off. In the summer, the jacketing water is cooled by being sprayed into the air in a cooling pond, 50 x 80 ft., situated a short distance from the compressor building. The compressed air is collected in receivers from where it is delivered into the 16-in. pipe which carries it underground with a loss of about 3 lb. in transmission.

Buildings, Etc.—The mine buildings and their general layout can be seen in Figure 10 (General Arrangement, Surface Buildings).

These are all of substantial concrete and brick construction and each has been carefully planned to fulfil all requirements as to space, light, warmth and fire protection. All frame work is of steel, all walls of brick, all floors of concrete, and all roofs of asbestos shingles; windows are provided with steel sash and wire-reinforced glass panes.

The two hoists for No. 3 shaft are in a building by themselves. Travelling electric cranes are installed in the hoist and compressor buildings.

The collar house at No. 3 shaft provides room for a drill repair shop, skip repair shop and a car repair shop, and is traversed by four tracks of the shaft gauge on which extra skips and man-cages stand ready for immediate transfer to the shaft when required.

The Change House.—This is so situated that the men on the way to work come into it directly, and, having changed, go through the clockroom and thence through a covered passage

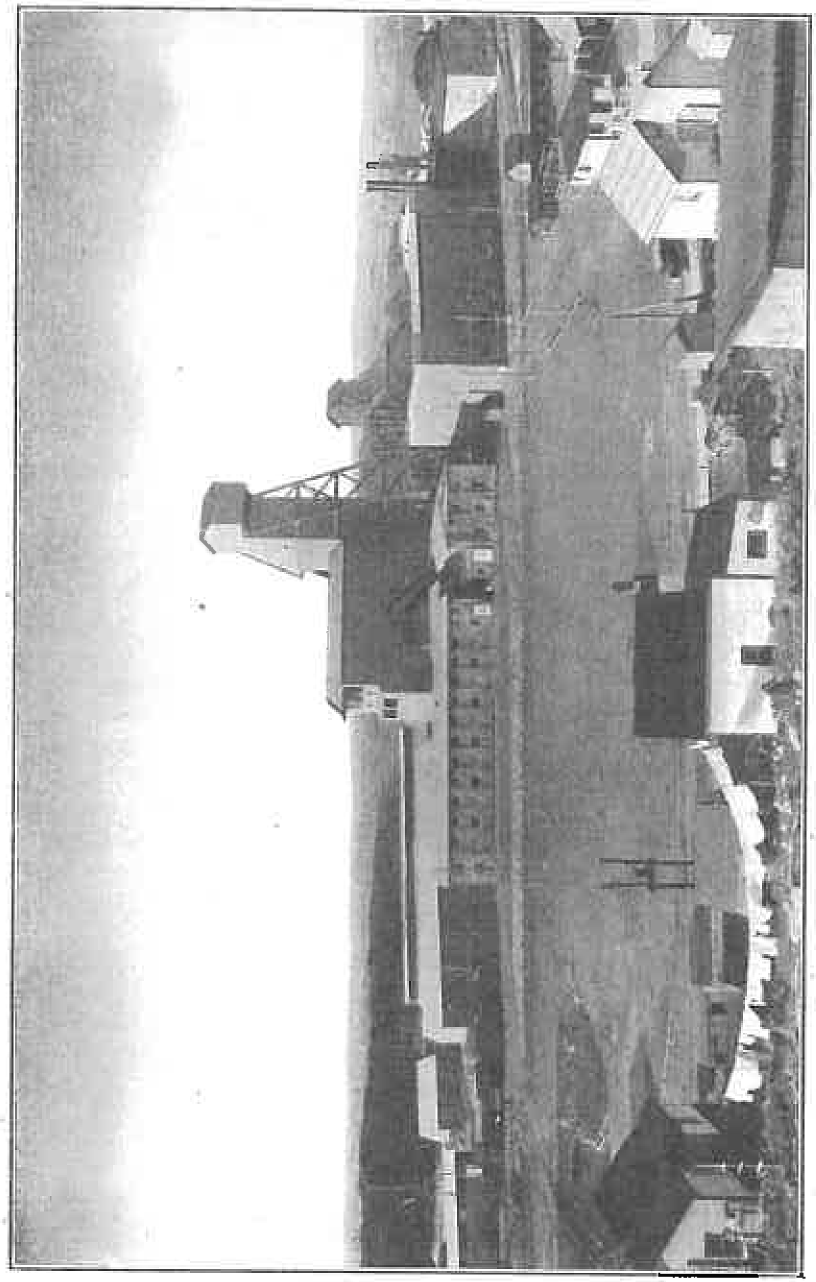


Photo by British & Colonial Press, Toronto.

Plate XIa.—Surface plant and building, Creighton mine.

leading over the railway tracks to a warm room at the collar of No. 3 shaft. The building is of two storeys, there being on each floor 660 individual lockers arranged in 19 banks, together witb 12 wash troughs and shower baths supplied with running hot and cold water. The lockers are of two compartments with screen doors and are connected to an exhaust fan that sucks out the foul air and draws in the warm air supplied by the hot-air pipes which heat the building.

Heating.—A central boiler plant supplies steam to coils in the various buildings, and fans drawing fresh air from out of doors through these coils, drive it through pipes leading to hot-air registers where needed. This method of heating has been very successful, but to cut down the cost of coal, etc., electric grids have been substituted for the steam coils and the cold air is drawn through them, the use of coal being thereby obviated. The water for the change house is heated by electric heaters inserted in long tanks connected witb the water main. This method of heating the water has proved successful during a trial of over a year.

Yard Arrangement.—In laying out the surface buildings, etc., plenty of room was provided for storing timber, machinery, etc., and this yard is well served by railway sidings and narrow-gauge tracks traversing each section. The narrow-gauge tracks are on zero grade; they have been well tamped to prevent movement by frost, etc., and have been laid as carefully as those underground.

The waste rock is conveniently and cheaply disposed of in the open pit.

SMELTING METHODS

The method of treating the ore, as now practiced, includes roasting, smelting in a blast furnace or reverberatory, and converting the matte in basic converters.

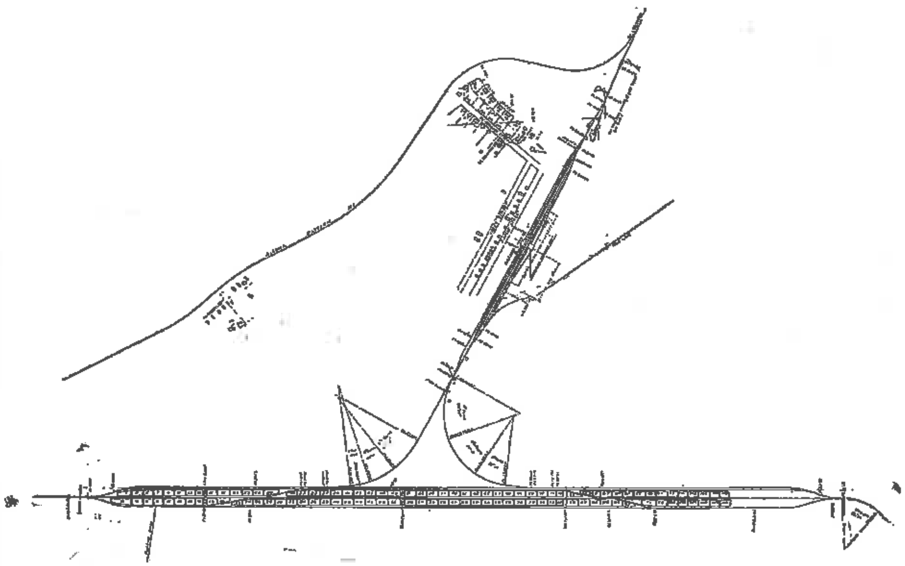
The ore as shipped from the Creighton mine, which is now the only operating mine of the company, consists of two main products, coarse and fines, in the proportion of about 60% of

the former and 40% of the latter. The ore is crushed at the mine to pass a 6-in. ring, and the fines is the portion that passes through circular openings of 1½ in. in the trommels.

A typical analysis of the ore would be:—

	Per cent.
Cu	1.50
Ni	4.00
Fe	41.50
S	24.00
SiO_2	17.00

On account of the high sulphur content of the ore, direct smelting is not practicable. The grade of matte produced would be only about 10% CuNi, which would throw an unduly large amount of work on the converters, and also a large quantity of limestone or other basic flux would be required in the furnaces to make a slag that would flow at a reasonable temperature. From the above analysis it is seen that the iron in the ore offers a basic flux quite as effective as limestone, and at a much lower cost. To make it available, however, it must first be changed from the sulphide, in which form it exists in the ore, to the oxide, in whicb form it can combine with the silicious rock material that has to be fluxed. One method of doing this would be to oxidize it in the furnace by means of the blast, or, in other words, to smelt the ore pyritically. Many and persistent attempts have been made to do this with these ores, but so far without success. On this account roasting in some form or other is the only alternative. Mechanical roasters require a finely ground feed, and a certain portion of the ore is treated in Wedge roasters and charged to a reverberatory furnace. The smelting equipment, however, consists principally of blast furnaces, which are not suitable for smelting fine material. In order, therefore, to provide a product suitable for use in the blast furnaces and at the same time one which will produce the proper grade of matte, it is necessary to roast the ore in the lump form. This is done in large heaps in the open, and is the practice that has been followed from the earliest days of the smelting industry in Copper Cliff. For many years the roasting was



-O'DONNELL ROAST YARD-

Figure 12.

done in the immediate vicinity of the town, but as operations increased in magnitude, it was finally decided (in 1915) to move the beds to a locality where the minimum of damage would be done to vegetation.

The site chosen is in Graham township at mileage 16½ on the Algoma Eastern Railway, and is about nine miles in a direct line, almost due west from Copper Cliff, and about four miles west of the Creighton mine. By the railway, the distances are 13 and 5 miles respectively. Accommodation for the men employed at the roast yard has been provided by the company, and the village of O'Donnell has been built. There are a club house and several boarding houses for the unmarried men, and numerous well-built houses and cottages for those who are married. Special attention has been given to sanitation and the supply of drinking water. The latter is treated by one of

the most up-to-date methods of chlorination (the Wallace-Biernan system), which renders the water perfectly safe without making it unpalatable. Its good quality is further ensured by

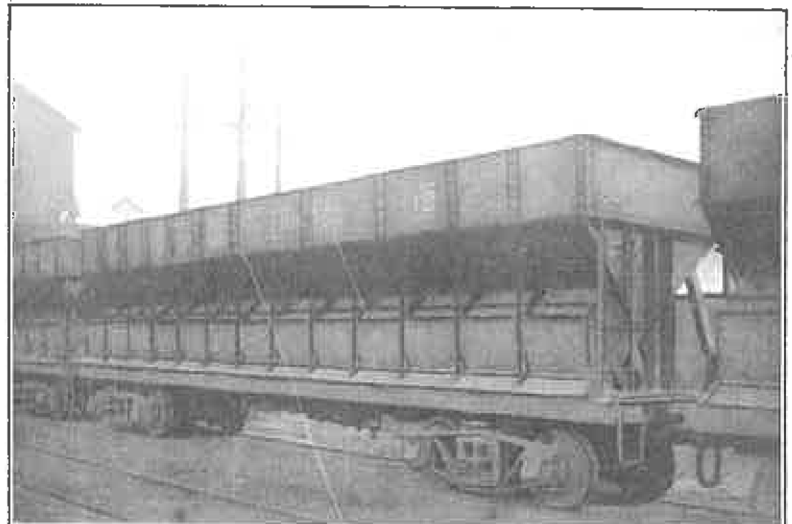


Plate XII.—100-ton transfer car.
Plate XIII.—Transfer plant, roast yard.

taking frequent samples for bacterial examination in the laboratory at Copper Cliff. An additional check is obtained by periodically sending samples to the laboratory of the Provincial Board of Health at Toronto. The same care, in fact, is taken with the drinking water at all points where the company carries on operations.

ROAST YARD

The plan of the roast yard is shown in Figure 12. The numerous running-tracks and storage-tracks assist in the rapid and economical handling of the ore. The ore is brought from Creighton by the Algoma Eastern Railway in 50-ton steel

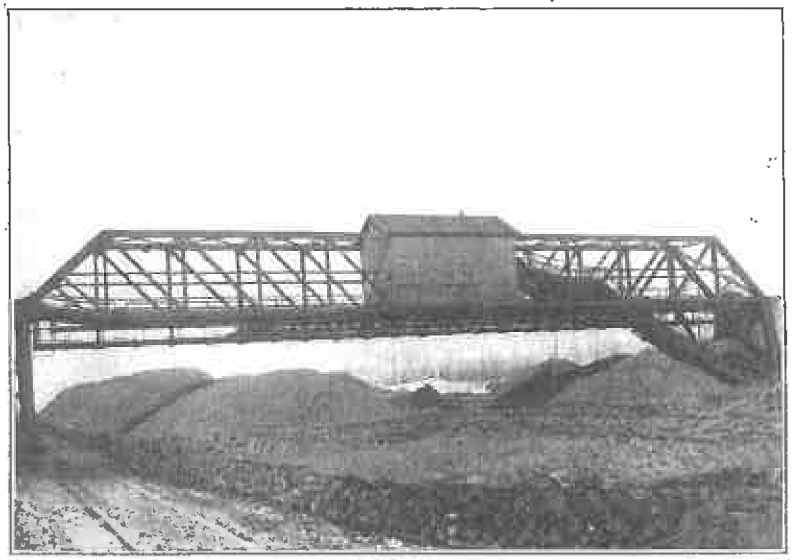


Plate 14.—Unloading bridge, roast yard.

drop-bottom cars and placed on the receiving track at O'Donnell. It is then taken by one of the company's locomotives to be weighed on the 100-ton standard railway scales, after which it goes to a transferring plant near-by, where the ore is dumped into a sunken hopper and thence fed by means of a short pan-conveyor under the hopper, to a 42-in. Stephens-Adamson

elevator which delivers the ore into specially designed 100-ton side-discharging steel cars. It requires about 10 minutes to fill one of these cars. As there are 12 cars in use, 1,200 tons of ore can be held in them at any one time. Some are kept loaded at all times in order to prevent any delay in the next step, which is the distribution of the ore on the beds.

The loaded transfer cars are taken to the roast yard proper and 'spotted' at the unloading bridge. The bridge has a clear span of 170 feet and travels on the inside rail of the standard gauge tracks that run along each side of the roast beds and are used as loading tracks when the roasted ore is picked up. As shown in the plan of the roast yard there are two parallel rows of roast beds witb a track between them, and also a track along the south side and two along the north side. The last are 14 ft. apart from centre to centre, and when the special transfer car is 'spotted' on the outer track on the north side (and opposite the bridge), it is in position to discharge the ore into the receiving hopper of the bridge. In the transfer car there are thirteen discharge doors, each of which is opened and closed independently of the others. The receiving hopper of the bridge can take the discharge from three doors simultaneously. Commencing at the end of the car, the practice is to open the two end doors, and when the flow of ore slows up somewhat, the third is opened. By this time the ore opposite the first outlet is fully discharged, the door is closed and the bridge moved one door's width, when the process is repeated until the car is emptied, which takes from thirteen to seventeen minutes. One man operates the various controls on the bridge, and two others attend to the opening and closing of the doors on the transfer cars and to the guiding of the ore to the receiving hopper.

From the receiving hopper the ore is fed by a 60-in. pan-conveyor to a 36-in. pan-conveyor which elevates it to another hopper at the highest point of the bridge, from which it is discharged in a continuous stream on to a 30-in. shuttle conveyor belt. When building a particular bed the direction of travel of this belt is always the same, but the travel of the carriage on which the belt operates is automatically reversed at limits that

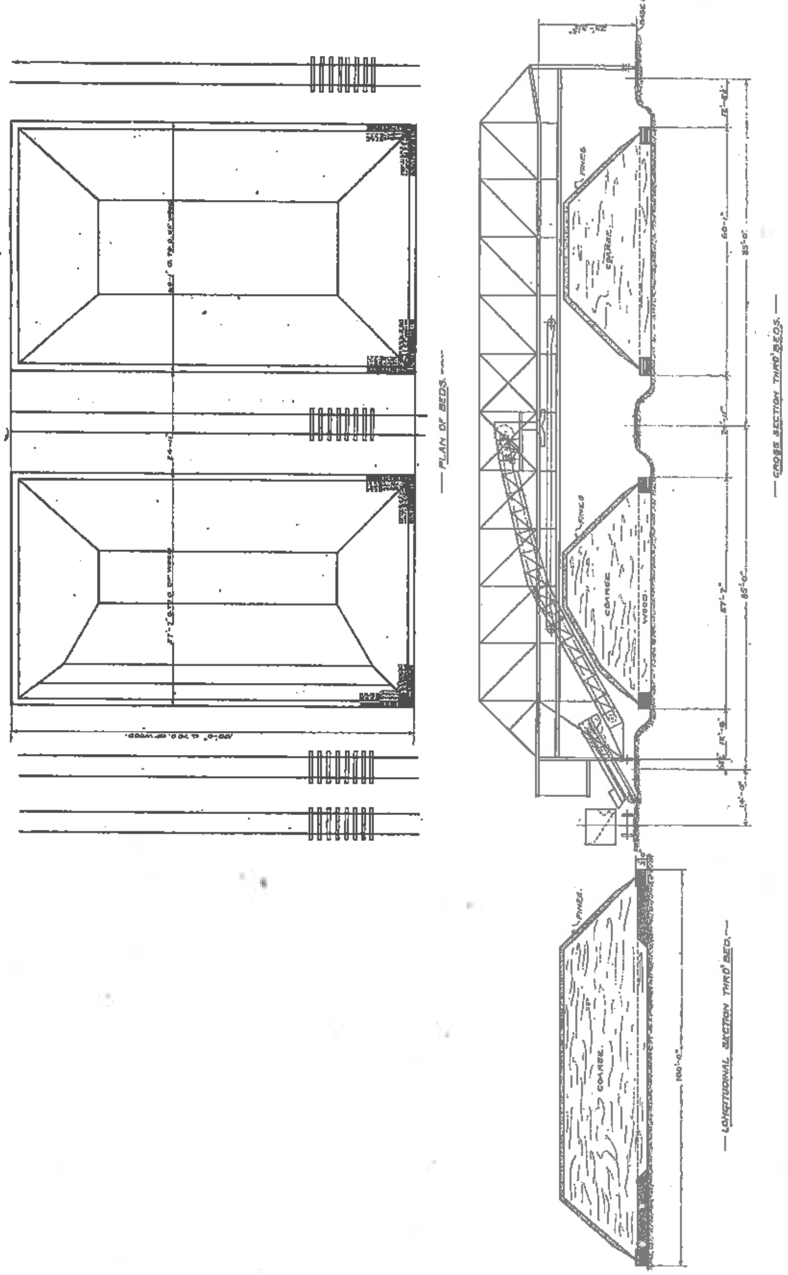


Figure 13.

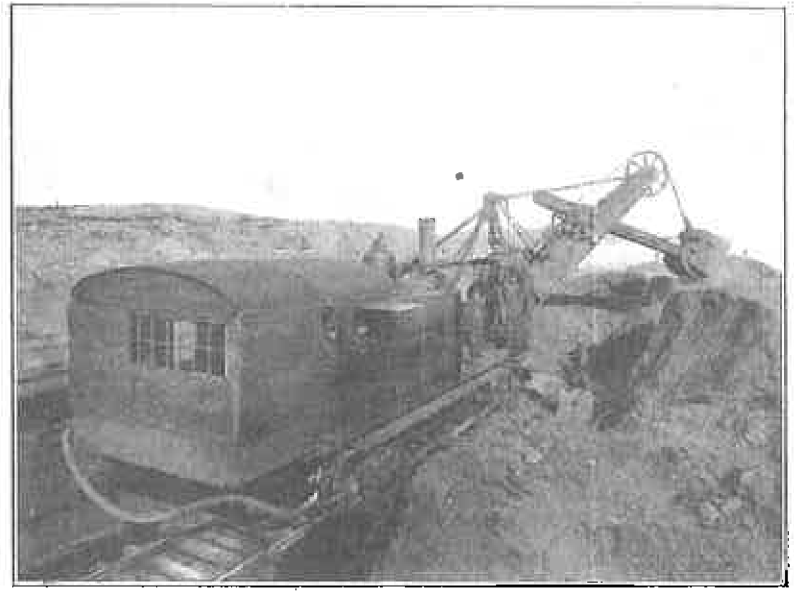


Plate 15.

can be set as desired. The belt travels towards the transfer car when the bed nearest the car is being built, and in the opposite direction when the one farthest away is being built. By the operation of the shuttle and the travel of the bridge itself, the beds are built uniformly over their whole area, but if it is desired to build up a certain part more quickly than another, this can readily be done by stopping the conveyor carriage; the belt will then discharge the ore in the one spot as long as the bridge is stationary.

The ground on which the beds are built is about 4 feet below the level of the tracks. This difference enables the bridge to build larger beds than it could if the tracks and bed bottoms were on the same level. The only preparation necessary before beginning to build a bed is to lay the wood required to start the roasting. The wood for each bed covers a rectangular area about 100 ft. long and 60 ft. wide. The width is determined by the distance between the tracks, but the length is largely a matter of convenience in building and lighting. The wood used is the usual 4-ft. cordwood, and it should be of good quality. The finished bed contains about 5,000 tons of ore.

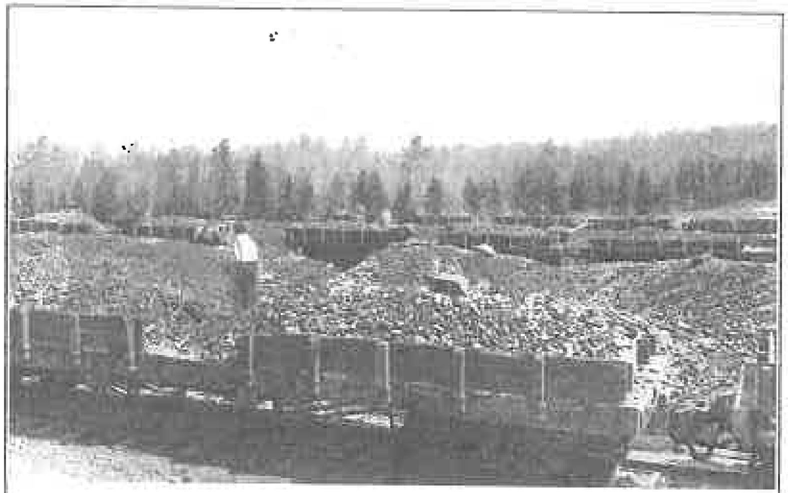


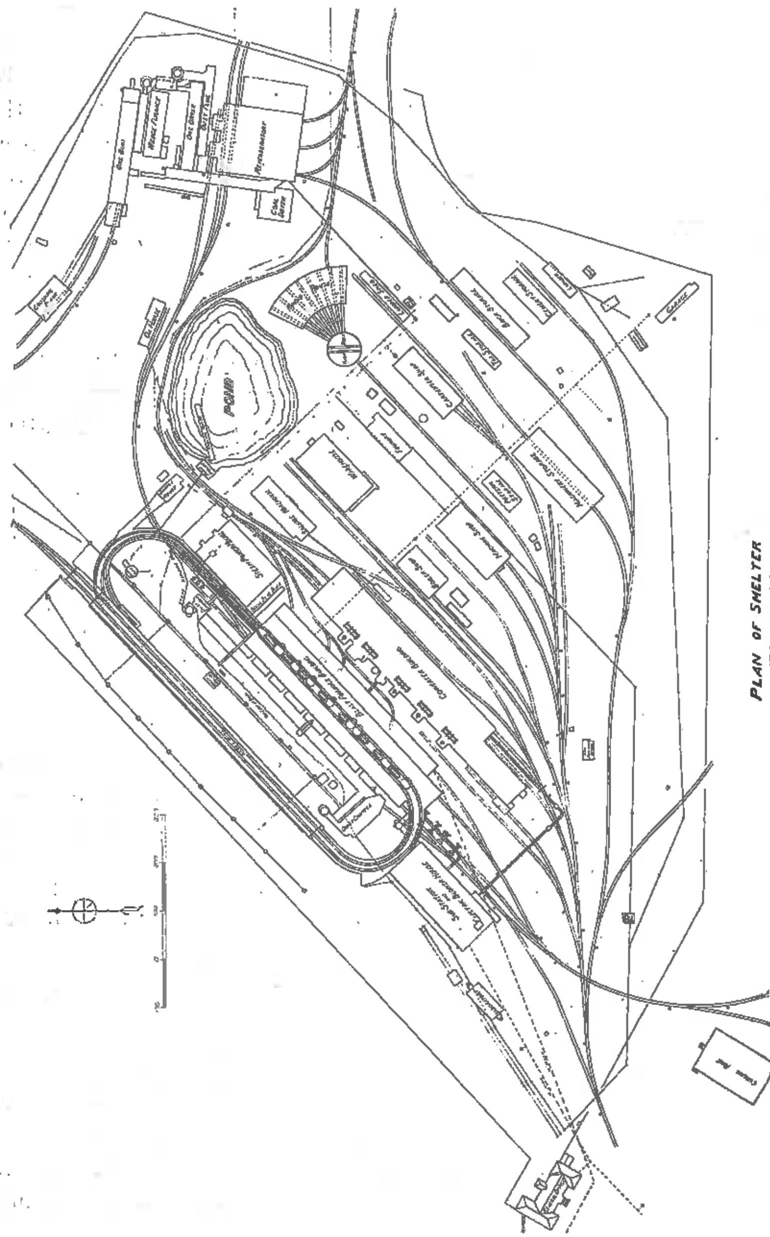
Plate XVI.—Unloading green ore by hand.
Plate XVII.—View of roast beds built by hand.

The beds are lighted as soon as possible after the building is completed. As the wood burns away, the ore settles down and cracks develop in the surface of the bed. This is the critical time of the process, and bed trimmers are constantly on the alert, during the first week or two of the burning, to close these cracks as quickly as possible. After the subsidence is completed no more openings are likely to form, and the bed requires practically no further attention. It will probably burn for six or seven months, by which time the sulphur will be reduced to about 10 per cent.

The roasted ore is re-claimed from the beds by two Atlantic steam shovels having a dipper capacity of 2½ cu. yds. They load the roast ore into the same 50-ton cars that bring the green ore from the mine.

The bridge was put in operation early in 1919. Previously, the ore was unloaded from flat-cars by men with wheelbarrows (Plate XVI), and the beds could not be made so high as at present because the men would not wheel the ore up much higher than the floor of the car on which they were working. An average bed then contained about 2,500 tons, and burned out in three or four months. The larger bed, which it is now possible to build, has several advantages over the smaller one, the chief being that (1) more ore can be stocked in the same space, (2) the ore roasts better, and (3) less wood is required. The same quantity of wood had to be used to start the roasting of the low beds as is required for the higher, and therefore larger, beds. The better roasting is due partly to the fact that the large beds roast more uniformly and for a longer time, but also because they have a smaller percentage by weight of the outer part of the bed that is roasted very inconsiderably in either case.

For a number of years before the bridge was built, various mechanical methods of handling the green ore had been under consideration, but while labour was plentiful and cheap they did not appear attractive, and it was mainly the shortage of labour during the war that finally caused the decision to be made



PLAN OF SMELTER
Figure 14.

to adopt the present method as the best of a number investigated. It has proved even more satisfactory than was anticipated, and after some alterations in the elevators (considered advisable after operating for some time, and which were made in 1919), it readily handles the required tonnages. The operating costs, also, have been quite as low as was anticipated.

BLAST FURNACES

There are eight furnaces in this department. Five of them are 17 ft. in length, one is 21¼ ft. and two are 25½ ft., giving a total furnace length of 157¼ ft. All have the same width at the tuyeres, namely 50 inches, and are similar in construction throughout except for such differences as are due to the different lengths.

The cast-iron hearth-plates are supported by 12-in. I-beams laid transversely to the furnace length. No cooling is provided for these plates beyond that due to the air naturally circulating under them. The crucible of the furnace is built of chrome brick, which, at the sides, come up to within 6 inches of the tuyeres and are stepped down to the centre in the form of a 'V,' the minimum thickness of brick being 18 inches.

The sides of the furnace are formed of sections 4 ft. 3 in. wide, containing six jackets for the full height of the furnace. First there is a pair of cast-iron tuyere-jackets (in which are embedded 1¼-in. water pipes), each 2 ft. 1¼ in. wide and 4 ft. 7 in. high, resting on the hearth plates. Each of the pair has, near the top, two tuyere openings 6 in. in diameter. Above the tuyere jackets is another pair of cast-iron jackets of the same width and of similar construction, but only 3 ft. 11 in. in height. Above these is a steel water-jacket 3 ft. 6 in. high and 4 ft. 3 in. wide, and finally another steel jacket 6 ft. high and 4 ft. 3 in. wide. The use of the small steel jacket was made necessary when the height of the furnace was increased several years ago. The cast-iron jackets are given a slope outward to form the bosh of the furnace, the maximum inside width being 5 ft. 10 in., an increase from 2 ft. 10 in. at the hearth plates. The

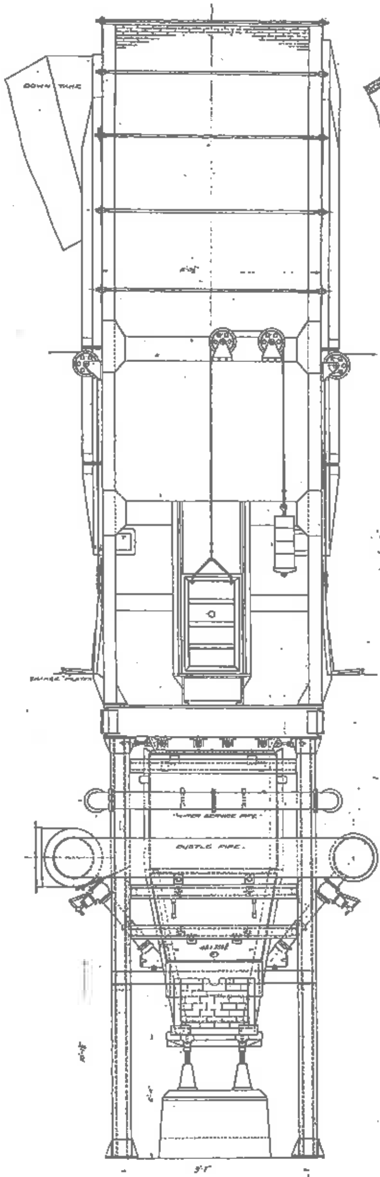


Figure 15.—End elevation furnace No. 6.

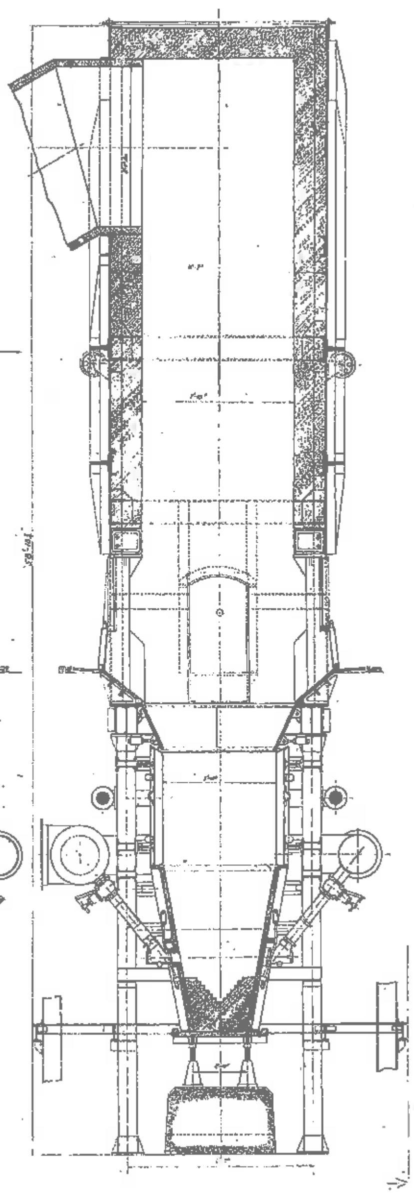


Figure 16.—Cross-section, blast furnace No. 6.

steel jackets are vertical. The 17-ft. furnace requires four of these sections to the side, the 21¼-ft. furnace, five, and the 25½-ft. furnace, six. At the dead end of the furnace the lowest jacket is of cast iron with water pipes imbedded in the same way as with the cast-iron side-jackets. It is 3 ft. 9 in. high and above it are three steel water-jackets, 4 ft. 9 in., 3 ft. 6 in. and 6 ft. high, respectively. At the front end the lowest jacket is of copper 32 in. wide and 5 ft. long, with the lower edge 1 ft. 11 in. above the hearth plates. This forms the trap of the furnace. The top of the copper jacket is the same height above the hearth plate as the top of the cast-iron jacket at the dead end, and hence the remainder of the jackets at the front are the same as those at the dead end. Each furnace is provided with a small side-tap jacket fitted into a notched tuyere-jacket and placed near the middle of the furnace.

The furnace spout is of chrome brick built against the copper jacket. It is carried on a 4-in. cast-iron plate about 4 ft. wide and 5 ft. long, which rests on the bottom plate of the furnace and the side of the settler. Cast-iron water-cooled side-plates retain and protect the brick. A water-cooled cast-iron lip is placed where the mixed matte and slag flow from the spout. Further protection to the brick is given by another cast-iron cooler placed under the lip. The effective depth of the trap formed by the spout and the copper jacket is about 9 inches. This type of spout was developed at the plant to overcome trouble with corrosive low-grade copper-nickel mattes, and has proved very satisfactory. A cut-out of the spout is practically unknown.

The furnace tops are of fire brick stiffened with steel work. They rise about 33 feet higher than the charging floor. The upper 15 ft. is in the form of a catenary arch sprung over the length of the furnace. In the smaller furnaces an 8-ft. downtake connection is just under the centre of the arch. In the larger furnaces the diameter is correspondingly greater. The downtakes are made of steel plate, the first 20 feet being lined with fire brick. They are about 64 feet long and run down at an angle of about 30° with the horizontal to a 20-ft. balloon flue,

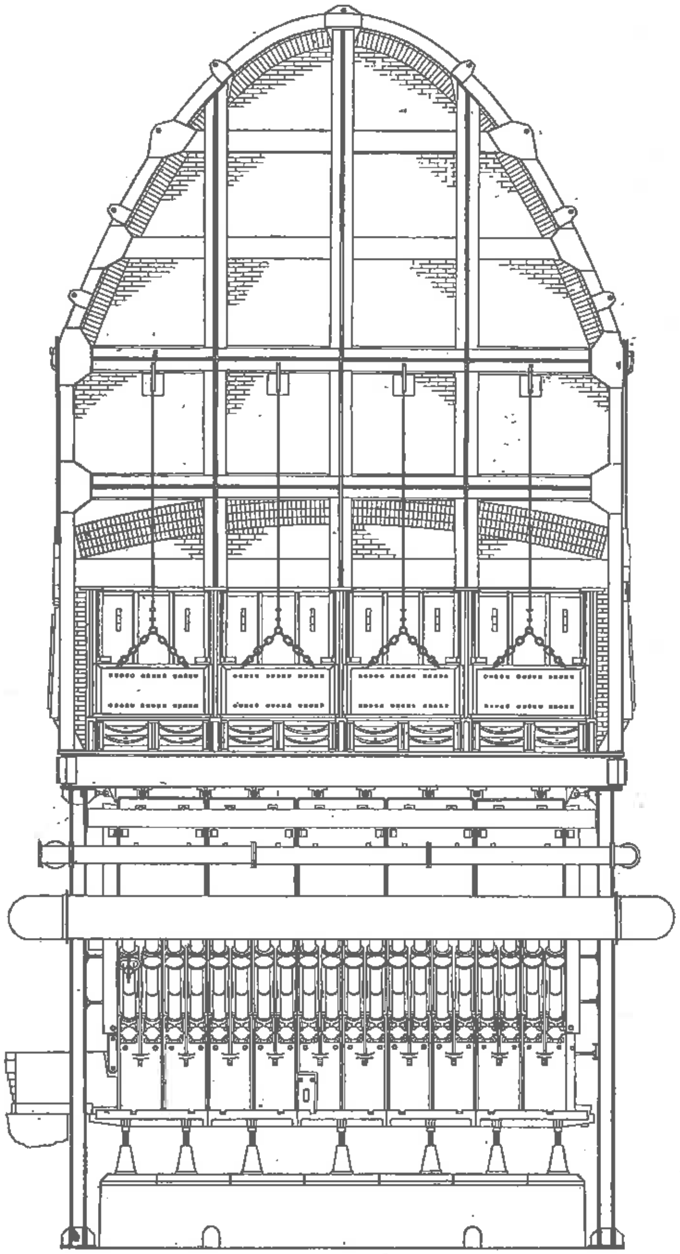
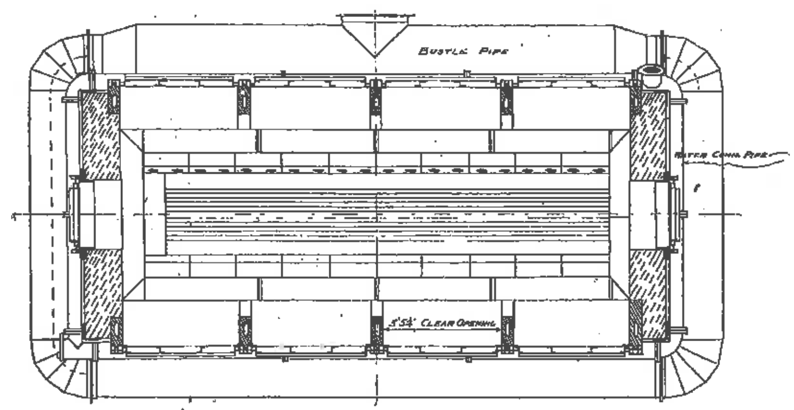


Figure 17.—Side elevation, blast furnace No. 6.

at the rear of the furnaces. This flue is of steel and is about 530 feet long. It is provided with clean-out chutes, which discharge the collected dust into a standard-gauge steel-car. Approximately 2% of the weight of the charge is recovered in this way. Additional dust is also caught in a wire-hung dust collector placed between the balloon flue and one of the two stacks that serve the furnaces. Each of these stacks is 210 feet high above the yard level and has an inside diameter of 15 feet at the top. They are connected to opposite ends of the balloon flue.

The water for use in the jackets is obtained from a small lake near the town (Copper Cliff). As the supply is limited it is used over again after being collected in a cooling pond at the smelter. It is treated with alkali to neutralize any acid present.

The settlers are 5 ft. 6 in. high and the shell is made of ½-in. boiler plate. It is held in position by lugs and bolts sunk in the concrete base. The dimension on the longer axis is about 20 ft. inside the lining. The lining consists of chrome brick about 12 in. thick at the bottom and 18 in. thick in the walls. At the point where the stream from the furnace falls



48"×255" BLAST FURNACE-SECTIONAL PLAN

Figure 18.

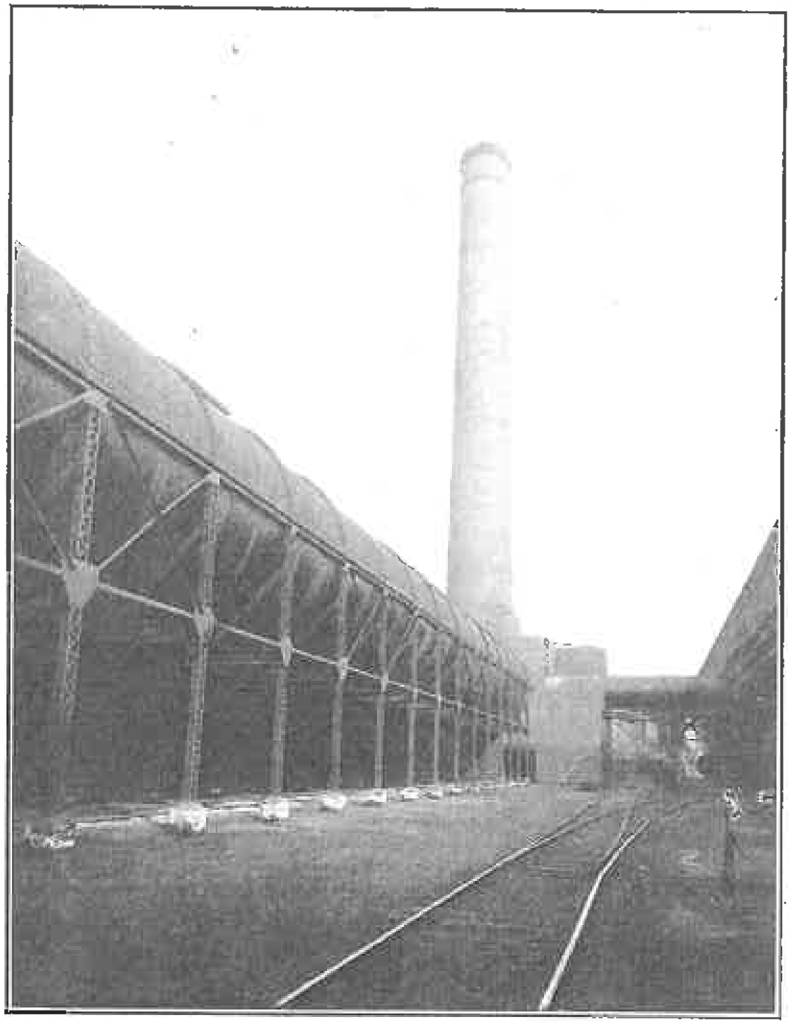


Plate XVIII.—Balloon flue and stack, blast furnaces.

into the settler, and also around the tap-holes, extra protection is given by the use of cast-iron cooling-plates. The tap-hole is formed by two chrome blocks 14 in. square and each 6 in. thick. The diameter of the hole through the blocks is 2 inches.

There are two tap-holes in each settler, and the pouring lip for converter slag is placed between them. The settlers have the shape of a distorted oval, and are placed at the ends of the furnaces with the longer axis at right angles to the longitudinal axis of the furnace.

All material required to make up the charge for the blast furnaces is delivered to a system of bins which are parallel to, and about 200 feet from, the furnace-building proper. The ground on which the bins stand is at the same level as the charging floor of the furnaces, and the top of the bins is 35 feet higher. Along the top of the bins run two parallel standard-gauge tracks and the whole is covered with a train shed. The sides of the bins are also completely housed in except for the lower six feet on the side nearer the furnace. The length of the building is 600 feet and the width about 30 feet. Heavy timber construction is used throughout.

There are 50 separate sections or compartments, each having a capacity of about 6,000 cu. ft. Twenty-four of these are generally reserved for coke, and twelve or thirteen for the roasted ore, giving storage room for about 2,000 tons of the former and 4,000 tons of the latter, sufficient to keep the plant in operation through any ordinary tie-up in transportation. The remaining bins contain converter slag, scrap, green ore, quartz and limestone; a bin is also used to hold coal for the heating boilers and smelter locomotives. As far as possible all material is brought to the storage bins in bottom-dump or side-dump cars, in order to save both time and labour in the unloading. Grizzlies are placed over all bins except the coke bins, to prevent large lumps of ore or 'revert' falling in and possibly blocking the chutes below when the material is withdrawn.

Underneath the bins are two narrow-gauge tracks on which are operated small charging trains drawn by electric locomotives. The tracks continue, in the form of an oval, past each side of the furnaces, and back again under the bins; the trains travel always in the one direction. The trains consist of nine side-dump cars each having a capacity of 25 cubic feet, and are filled from chutes under the bins with the materials desired to make up the charge. The usual charge consists of three cars of coke, one car of 'revert,' one car of green ore, and four cars of roast ore. Sometimes two cars of green ore are used, in which case a car of roast ore is dropped. The cars containing the charge are weighed on a scale at the end of the bins, and the coke adjusted to the exact percentage required at the time. This is usually between 10% and 11% of the weight of the charge, depending on the quality of the coke and the nature of the charge. A typical charge would have about the following composition by weight:

Roast ore	12,000 lb.
Green ore	5,000 lb.
'Revert'	2,500 lb.
Total charge	19,500 lb.
Coke	2,000 lb.

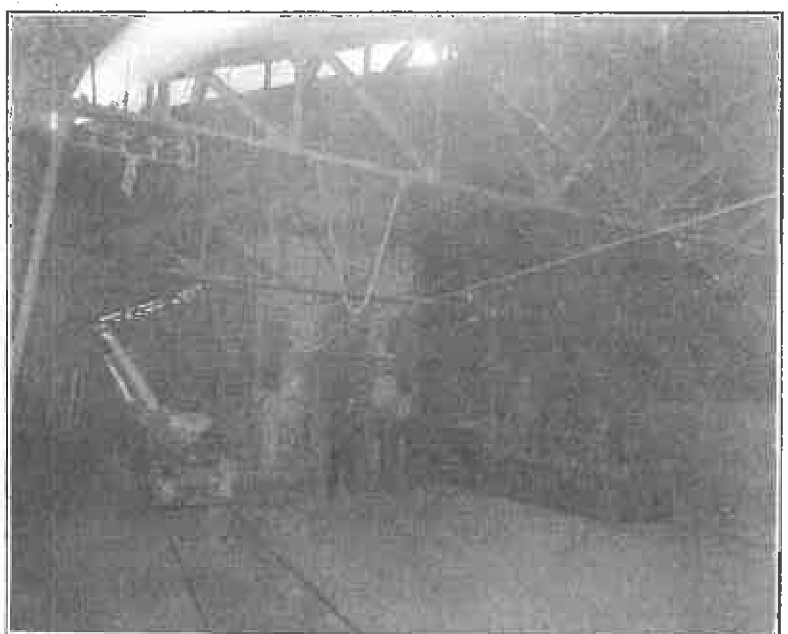


Plate XIX.—Charging floor, blast furnace building.

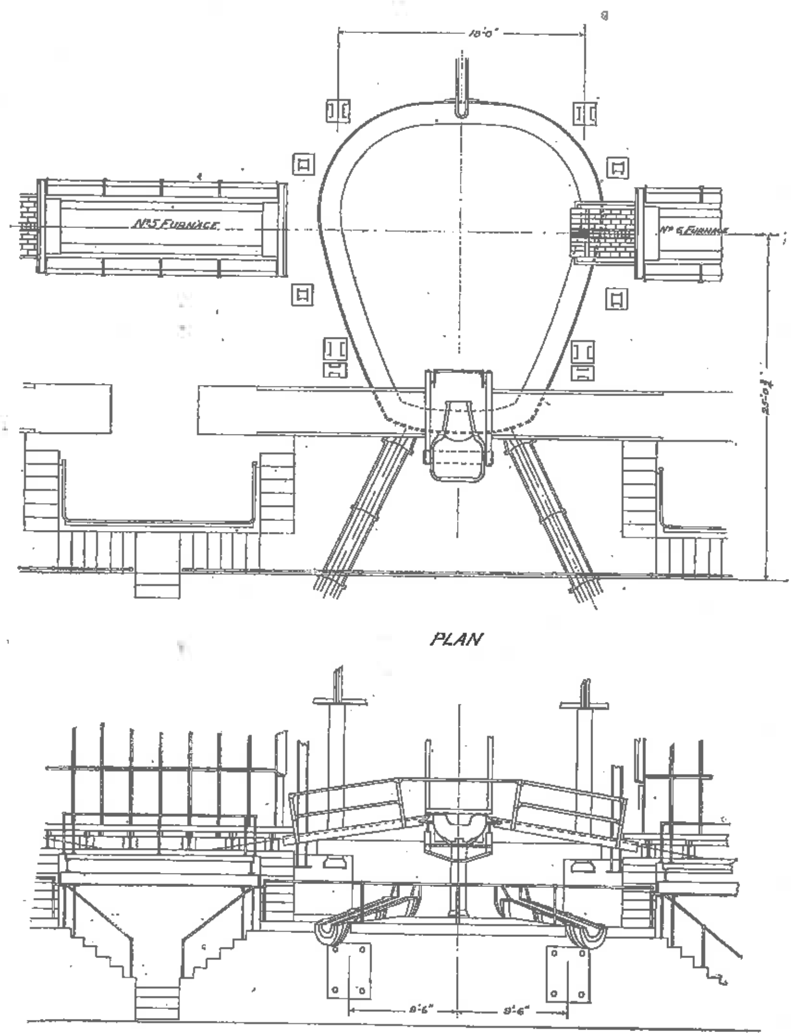
Plate XX.—Furnace blowers.

When using well roasted ore, a charge of this kind will give the grade of matte desired for the converters.

As a rule no flux is required, though both limestone and quartz are available if conditions demand their use. For the whole month of January, 1920, only fifteen tons of quartz and four tons of limestone were used.

When the charge train is brought to the furnace, the doors of the furnace are raised with the assistance of counter weights and the charge dumped in. The coke is put in first, and all material is spread as evenly as possible by moving the train back and forth as the cars are discharging their contents. Charging is done at intervals of from 20 to 30 minutes.

The blast for the combustion of the coke and the oxidation of the sulphur is furnished by four Connersville blowers, three



ELEVATION—MATTE SIDE

BLAST FURNACE SETTLER

Figure 19.

having a capacity of 44,800 cu. ft. per minute, and one of 33,000 cu. ft. capacity. The latter can be operated at three different speeds, furnishing 33,000 cu. ft., 23,800 cu. ft., or 16,800 cu. ft. as required. There is also a Nordberg blower of 24,000 cu. ft. capacity, which is now seldom used, but is available as a spare. It also is a three-speed machine giving 16,000 or 11,000 cu. ft. at the lower speeds. All blowers discharge into a common pipe, the supply to the furnaces being controlled by a gate at each furnace between the common pipe and the bustle pipe. The pressure in the bustle pipe is maintained at about 25 oz. The

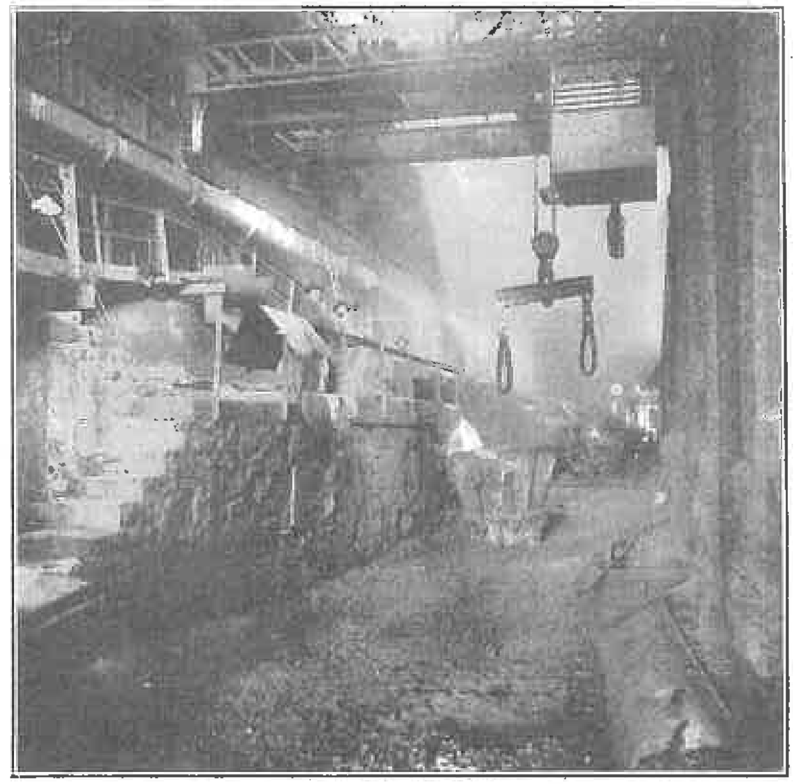


Photo by British & Colonial Press, Toronto.

Plate XXI.—Tapping floor, blast furnace building.

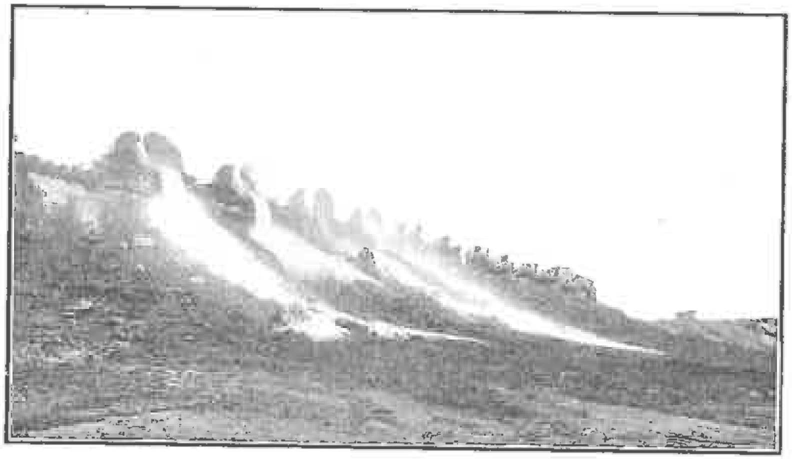


Photo by British & Colonial Press, Toronto.

Plate XXII.—Slag cars at dump.

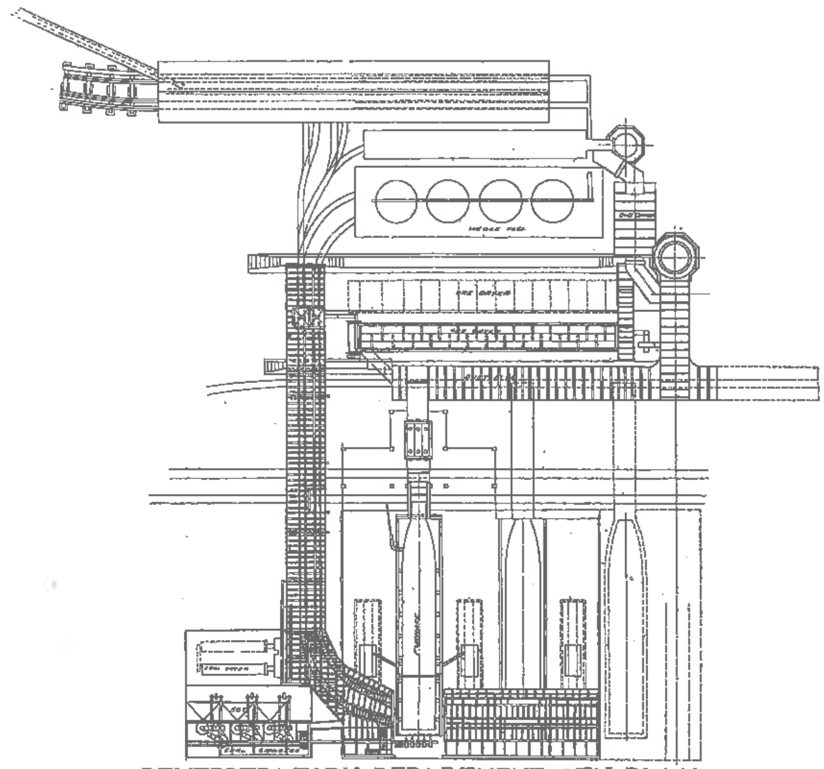
volume delivered to the furnaces, measured by rated blower capacity, is about 1,050 cu. ft. per minute per linear foot of furnace. A larger volume will give a faster running furnace for a time, but blow-holes soon develop and the capacity decreases. Another result is the imposition of additional work in barring down accretions and, in general, the attendant disadvantages of erratic operations.

The molten slag and matte from the furnace flows into the settler close to the side and nearer to the back of the settler where the slag overflows than to the front where the matte is tapped. This is far from being an ideal arrangement, but, without incurring excessive expenditure, was the best that could be done in the existing building when the settlers were enlarged to provide for pouring converter slag into them. A greater distance of travel for the furnace slag before overflowing would be a decided improvement. However, for the converter slag, the maximum distance of travel is provided.

The slag from the settlers overflows into pots of 225-cu. ft. capacity, standing on standard-gauge tracks. When full, they are drawn away to the slag dump where the liquid slag is poured.

The turning down of the pots is done by an electric motor. Before being placed under the slag stream, the pots are given a lime wash to aid in the removal of the 'skull' at the dump.

Before adopting the practice of pouring the converter slag into the settlers, a long series of experiments was undertaken. Two settlers, connected to furnaces smelting similar charges, but one having converter slag poured into it, were sampled independently over a period of nearly two years. Not only were the usual ladle samples taken at the settlers, but channel samples of the dump were also taken. This duplicate sampling gave added confidence in the results obtained. Over a period



REVERBERATORY DEPARTMENT GEN. PLAN

Figure 20

of 21 months the average of the dump samples showed only a difference of 0.008% Cu.Ni from the ladle samples.

In cases where the converter slag is required as a basic flux in the furnaces, the saving may not be so pronounced as in this plant where the tendency is to the use of a silicious flux rather than a basic one. The amount of slag poured into a settler is approximately equal in volume to the matte tapped from it.

An attempt is made to pour in the slag just after tapping. It is also poured as slowly as possible. A too rapid overflow into the furnace slag pot is prevented by damming the slag chute for a time.

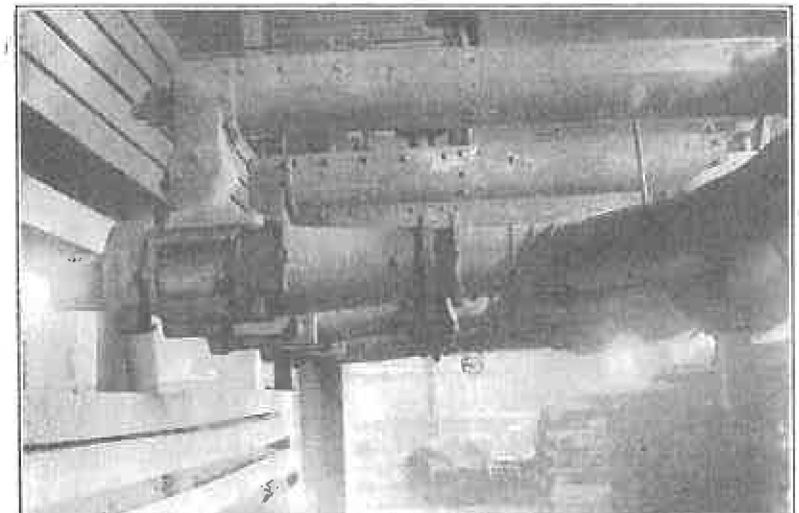
Furnace matte is tapped into sectional cast-iron ladles which hold about 8 tons. The ladles are given a protective lining of coverter slag before being used for matte. The matte is transferred to the converter building over a short narrow-gauge track provided with a 35-ton scale.

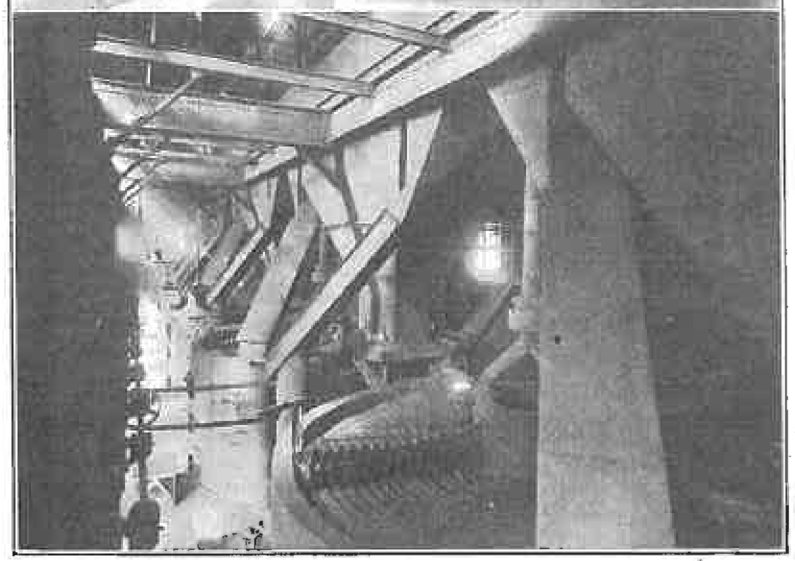
From the standpoint of both converters and blast furnaces, the most economical grade of matte to make is found to be from 25% to 27% copper-nickel, but as there is now considerable excess converter capacity, advantage is being taken of this fact to use up some green ore that is on hand, and at present the matte is somewhat lower.

REVERBERATORY FURNACES

The reverberatory plant was first put in operation about the end of 1911. The complete plant as now operated includes a system of steel storage-bins, a ball-mill department, a Wedge furnace roaster plant, a pulverized-coal department, and the reverberatory furnace itself, all housed in separate buildings.

The principal part of the charge to the reverberatory is fine ore from the mine. This ore has passed through a screen having 1½-in. circular openings, and about 50% of it is less than ¼-in. size. As the ore has no tendency to decrepitate on heating, it is necessary to crush it still smaller before it will





Place XXIII.—Coal burners at reverberatory plant.
Plate XXIV.—Top floor, Wedge roasting furnaces.

roast satisfactorily, and for this purpose four No. 8 Krupp ball mills are used. Standard-gauge tracks, at the same level as those that serve the blast furnace bins, bring the ore to the ball-mill plant, and coal and other material to the reverberatory storage bins. About 1,000 tons of coal and 3,500 tons of ore, besides smaller amounts of flue dust, fettling and flux can be stocked at the bins.

The ball mills are driven by 60-H.P. motors and rotate at a speed of 21 R.P.M. They are fitted with manganese-steel grinding plates and carry a load of 4,000 lb. of balls five inches in diameter. The consumption of balls is about 0.5 lb. per ton of material crushed. Ore is fed into the mills by a 24-in. link-belt pan-conveyor, intermittently operated by an adjustable rachet and pawl device. Before escaping from the mill, all ore must pass through 5-mesh steel screening, which gives a product of which 65% will pass a 20-mesh screen, and 25% a 100-mesh screen. When operating in this way, each mill will grind about 200 tons per 24 hours. Three mills, operated on two eight-hour shifts, are at present producing the necessary tonnage. Finer material is sometimes required and is secured by replacing the 5-mesh steel screens with 10-mesh bronze screens. In this size, steel screens would clog badly and would also have a much shorter life than the bronze. The discharge from the mills is taken by a system of 16-in. conveyor belts to a section of the storage bins holding about 3,500 tons.

The roaster building, which is parallel to, and about 25 feet from, the storage bins, contains four Wedge furnaces, 22 ft. 6 in. in diameter, equipped with seven hearths besides a top or drying hearth. The central shaft carries four arms on the seventh floor and on the drying hearth and two on each of the others. The arms are cooled by air supplied from two fans, one of which is usually found to be sufficient. The air pressure on the discharge side of the fan in equivalent to about three and a half inches of water. The air used for cooling the arms also supplies the oxygen necessary for the combustion of the sulphur and the oxidation of the iron. The furnaces are driven from a line shaft operated by a 35-H.P. motor and rotate about once in three minutes.

The fine ore in the storage bins is taken, by means of conveyors and an elevator, to bins above the Wedge furnaces from which it is fed automatically to the drying hearths, where practically all the moisture is driven off. When entering the furnace the ore contains about 24% sulphur, which is reduced to 10% or 11% when discharged. Lower sulphur can be obtained if desired, but at the expense of output. Each furnace has a capacity of about 115 tons of charge per day under the conditions at which they are now being operated. The heat of the furnaces is easily maintained by the roasting reactions, the temperature on the fourth, or hottest floor, reaching about 750° C. under normal operating conditions. Should a furnace become cooled, as when changing arms or through some other delay, it may be necessary to throw in a few shovelfuls of coal to assist in raising the heat again, and after a shut-down the heating up is done with oil.

Before escaping to the stack the gases pass first through a steel balloon-flue, below which is a narrow-gauge track for removing any collected dust; and then through a wire-hung dust chamber similar to the one at the blast furnaces. In this way about 250 tons of dust is recovered each month. The gases from the roaster plant usually pass up the same stack that serves the reverberatory furnace, but may be diverted to a steel stack serving the roasters only.

The hot calcines from the Wedge furnaces are removed in 3½-ton bottom-discharge cars and dumped into bins at the back of the reverberatory furnace, passing first over a scale to be weighed. Other ingredients of the charge are drawn from the storage bins in a similar manner. The haulage is done by 6-ton electric locomotives similar to those used at the blast furnaces. From the bins at the end of the reverberatory the calcines, etc., are drawn into small bottom-dump cars which operate on tracks above the side walls of the furnace. Just below each track is a trough, extending the length of the furnace, and kept filled by means of the small cars. Six-inch pipes, spaced two feet apart, extend from the bottom of the trough to holes in the roof of the reverberatory near the side

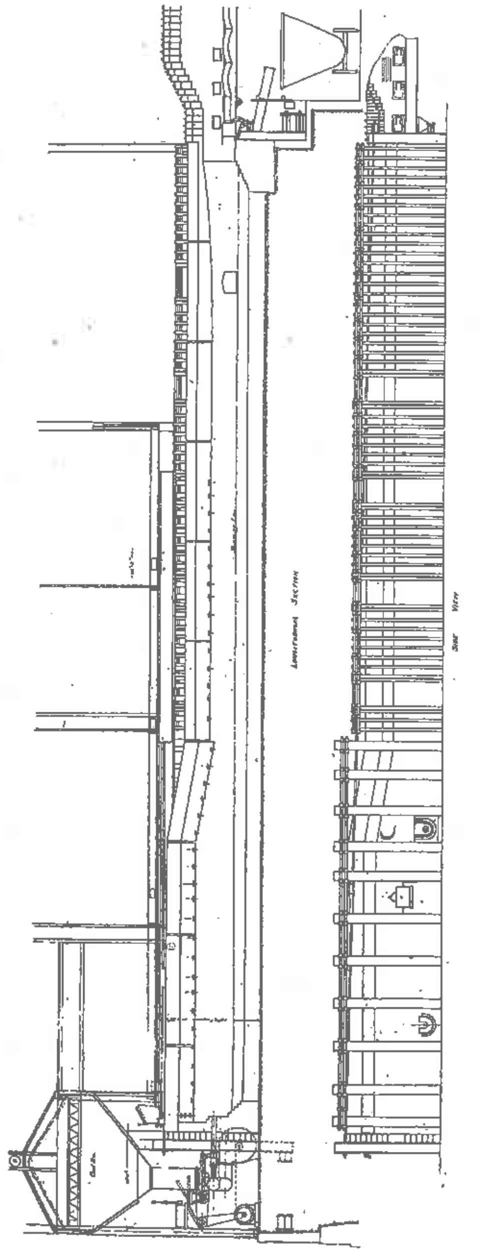


Figure 21.

walls, by which the charge enters the furnace. As the trough is always kept filled, the charge piles up against the inside walls of the furnace and reaches up to the roof. It sinks down only as smelting takes place and is immediately replaced by fresh material from above. In this way the charging is made absolutely continuous, the side walls are always protected, and the same area is constantly exposed to the action of the flame.

The reverberatory furnace itself is 112 ft. long by 19 ft. wide and has a cross-sectional area at the throat of 48 sq. ft. For the first thirty feet from the bridge wall the roof is two and a half feet higher than at the flue end, and a 12-ft. inclined section connects the two levels. The roof is built of silica brick, the thickness of the first forty-two feet being 20 inches and thereafter 15 inches. The side walls are also of silica brick except at the tap-hole and skimming bay, where some chrome and magnesite bricks are used, and at the bridge wall where fire brick is used. There were originally a number of doors along each side of the furnace, but these were bricked up when the side charging was adopted. The skimming bay is in the side wall of the furnace,

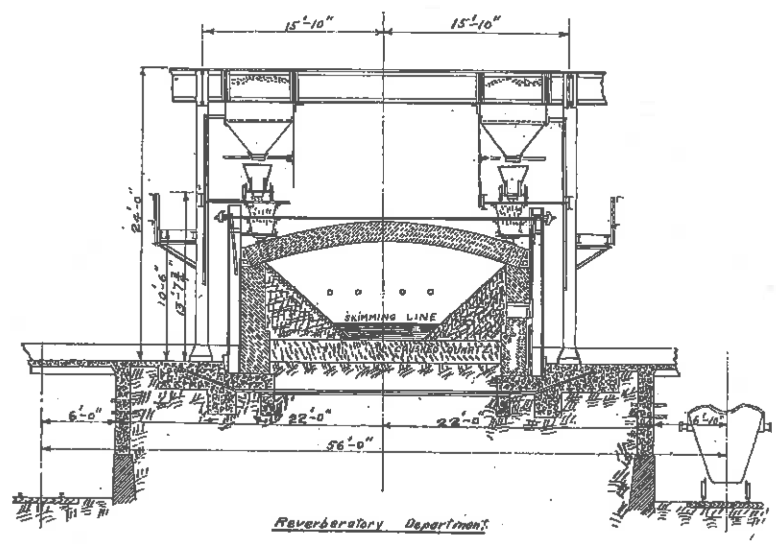


Figure 22.—Cross section of reverberatory furnace.

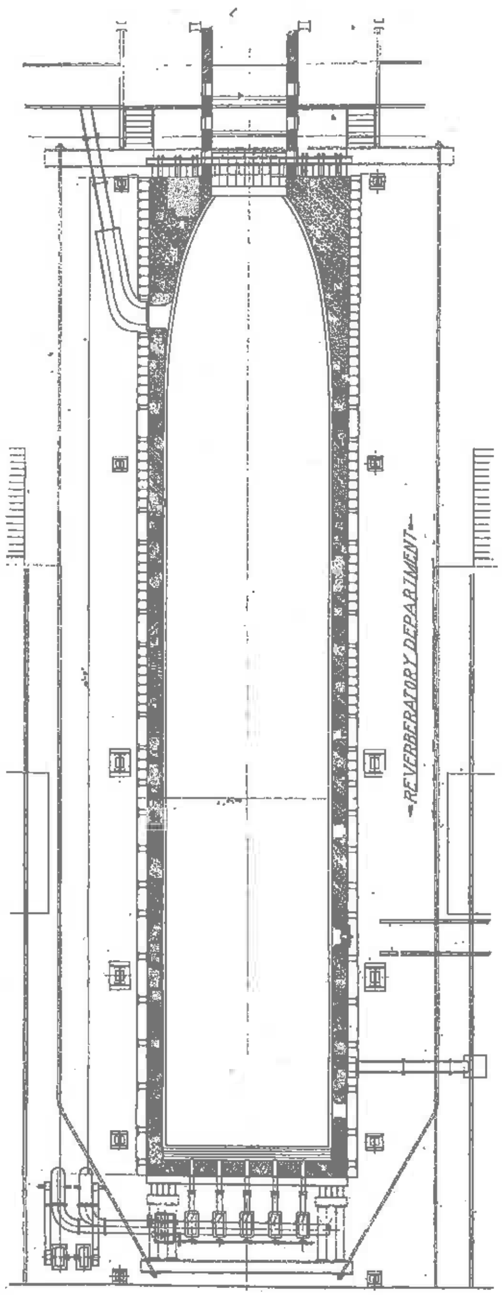


Figure 23.—Sectional plan of No. 1 reverberatory furnace.

fifteen feet from the throat, and the tap-hole is close to it. Skimming is done intermittently at intervals of about two and a half hours and the matte is tapped as required, but is never allowed to accumulate to a higher level than four inches below the skimming plate.

The fuel used is pulverized slack coal, having the following composition:—

	Per cent.
Volatile matter	37.00
Free carbon	50.00
Ash	13.00
S	3.30

The coal, after being drawn from the storage bins, passes through a Jeffrey coal cracker, which breaks up any lumps to three-quarters of an inch or less. A 16-in. conveyor belt now carries the coal about 300 feet to a storage bin of 70 tons capacity. A Merrick weightometer weighs the coal on the belt. The coal as received contains from 8% to 10% moisture, and before going to the pulverizers it is passed through one of two Ruggles-Coles double-cylinder rotary driers which reduces the moisture to about 2.5%. Further moisture escapes in the subsequent handling so that the coal, as finally delivered to the burners, contains about 1.0% to 1.5% moisture. From the drier, the coal is elevated to three 50-ton bins, from which it is fed to three 4-roller Raymond pulverizers. When fine enough, the coal is drawn up by the suction of a No. 11 Sturtevant special exhaust fan and delivered to a dust collector on the roof of the building. It is fed from the collector into the trough of a 16-in. helicoid screw conveyor, which delivers it to a 60-ton bin above and behind the reverberatory furnace. The fineness of the coal is very important, and a screen test should show at least 85% passing through a 200-mesh screen, and none larger than 100-mesh.

Five short screw-conveyors take the pulverized coal from the storage bin and drop it in front of a blast, which carries it into the furnace through five 5-in. pipes, projecting through the bridge wall. Two No. 8 Sturtevant fans are installed to supply the blast, but ordinarily only one is used. The blast furnishes only part of the air required for combustion, the balance being drawn in by natural draft through openings in the bridge wall. (Plate XXIII.) Control of the draft is given by a damper in the flue about 30 feet from the throat of the furnace. The speed of the screws which supply the burners is adjustable, and the amount of coal fed to the furnace can be varied from 60 tons to 120 tons per 24 hours when all burners are operated. At present about 80 tons of coal is being burned, and the furnace is smelting about 500 tons of charge per day, made up as follows:—

	Tons.
Hot calcines	400
Roaster flue dust	10
Blast-furnace flue dust	30
Green ore	30
Sample-house discards	30
	500

This mixture requires no flux and gives a slag containing about 33% silica and a matte with about 16% copper-nickel. If a higher-grade matte is made, a silicious flux must be added to keep the slag loss from increasing. The reverberatory slag contains approximately the same percentage of copper-nickel as the blast furnace slag.

In the above charge it will be noted that 80% comprises hot material, and that about six tons of charge is smelted per ton of coal burned. This is the average practice, but on occasions the amount of calcines is as high as 90% of the charge, and the coal ratio then becomes about seven to one.

The temperature at the hottest part of the furnace is about 1 580° C. (2,900° F.) and the gases escape at 1,100° C. (2,000° F.) A 400-H.P.-boiler is placed in the flue to utilize this heat. After leaving the boiler the gases pass through a brick flue before entering the stack. Provision is made for recovering

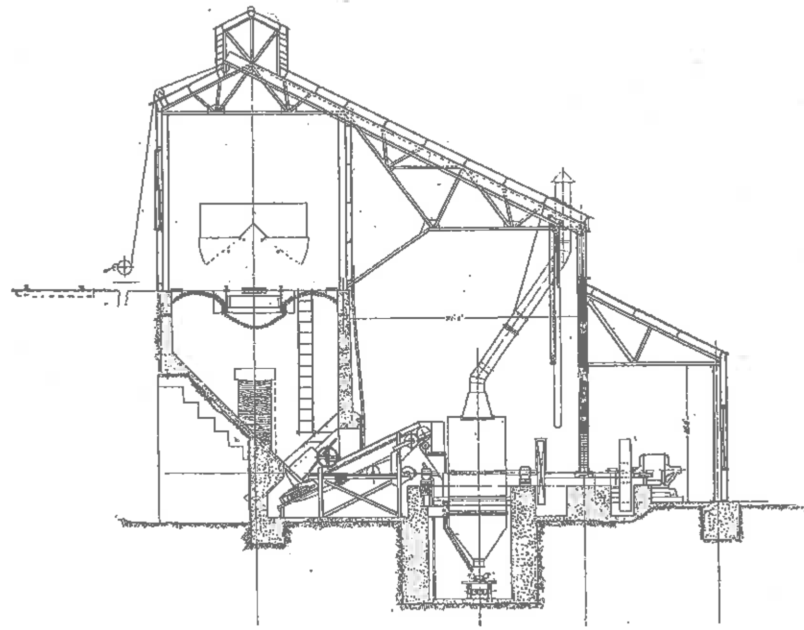


Figure 24.—Cross-section of crushing house, reverberatory plant.

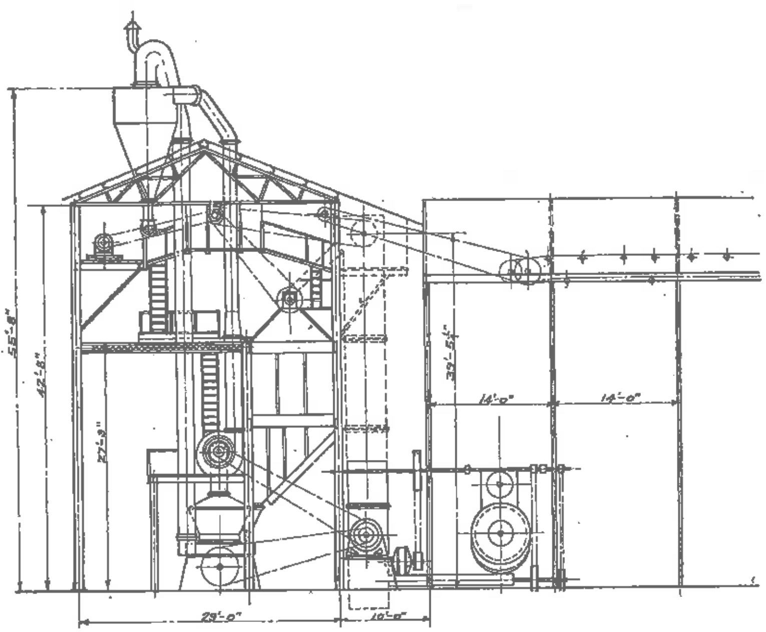
any dust that settles in the flue, but very little is ever found here. The gases are finally discharged through a brick stack similar to those at the blast furnaces and of the same height.

CONVERTERS

The converter department contains six basic-lined Peirce-Smith converters. The shells are 37 ft. 2 in. long by 10 ft. in diameter and are constructed of ⅝-in. steel plate. The throat for the escape of the gases is placed at the centre of the shell, and is about 5 ft. 6 in. in diameter. On either side are two small openings through which the flux is charged. At the front of the shell there are two spouts for skimming slag or casting the finished matte, but only the one near the control levers is used.

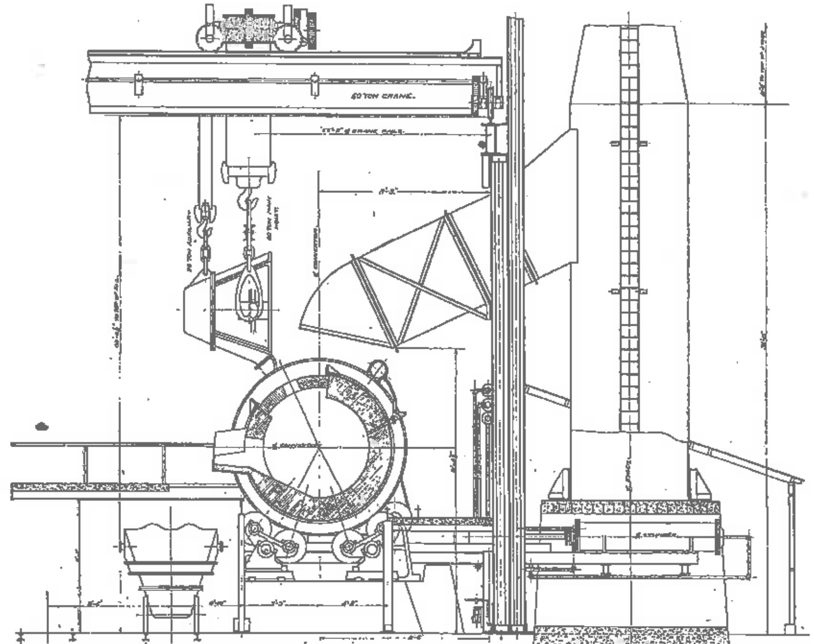
The converter rotates on four 12-ft. tread rings carried on equalizing trucks, which are mounted on concrete piers. The shell is turned by means of two wire cables anchored to the front and back just below the throat and operated by a hydraulic piston working in a horizontal cylinder. On account of the low temperatures during the winter, oil instead of water is used to transmit the pressure. In addition to a hand-controlled operating system each converter is provided with an automatic safety device which brings the converter off the tuyeres should the electric power be unexpectedly cut off.

There are forty-four tuyeres 1½ in. in diameter on each converter; only twenty-eight of these, however, are regularly punched. The blast is supplied to the tuyeres from a 16-in. bustle pipe traversing the length of the converter about 3 feet above the tuyere line. The pressure in this pipe is maintained at about 10 lb., under which condition there is no difficulty in keeping the tuyeres open. If more than twenty-eight tuyeres



COAL DRYER BLDG. GENL. ARRANGEMENT.

Figure 25



CONVERTER UNITS.
TYPICAL GENL. ARR'GT. OF COMPLETE EQUIPMENT — CROSS SECTION.

Figure 26.

are kept open the blast pressure tends to fall and the punching becomes very laborious. The tuyeres just below the stack opening are seldom punched on account of their tendency to throw molten material out of the shell.

Probably the most interesting feature of the smelter-power plant is the Rateau Battu Smoot turbo-blower installed in 1917, which is the chief source of blast supply for the converters. This blower has a capacity of 42,000 cu. ft. of free air per minute to a discharge pressure of 13 pounds gauge. A constant pressure is maintained by an automatic unloading valve in the blower inlet. This automatic unloading and stabilizing equipment is such as to permit a constant pressure of 13 pounds for all volumes up to, and including, 42,000 cu. ft. per minute within a variation not exceeding 4%, except such momentary variations as may occur at the instant one or more converters are thrown on or off the tuyeres. The regulating equipment is specially adapted to meet these conditions, and the variations in pressure are confined to a few second's interval, after which the regulator assumes control and normal pressure is regained. The synchronous motor direct-connected to the blower engine operates at 2,400 volts, 25-cycle, 3-phase, alternating current, with a normal speed of 1,500 R.P.M. This, with a similar installation at the Creighton mine, helps to maintain a power factor of around 95.

The magnesite lining of the shells is 24 in. thick at the bottom, 18 in. at the back or tuyere section, and 15 in. at the front. The top arch is of 9-in. magnesite. The life of the

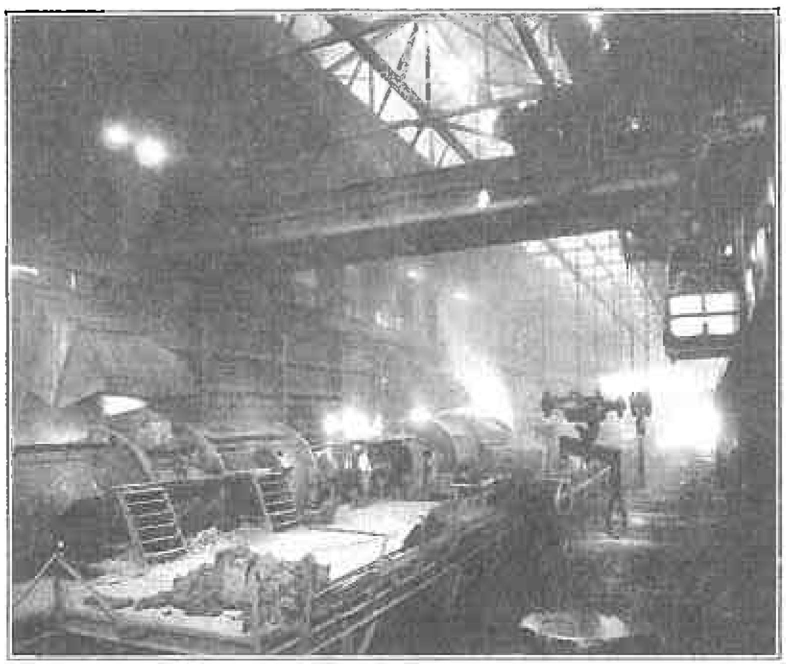


Photo by British & Colonial Press, Toronto.

Plate XXV.—View of converter building.

linings varies considerably, owing mainly to the difference in the grade of matte treated, but a converter in constant use should last for from six to nine months without repairs of any kind, and usually all that is required then is small repairs above the tuyere line and the removal of the accumulations that have solidified at the bottom. From records kept over a long period, the average consumption of magnesite is about six pounds per ton of iron oxidized or about 12 pounds per ton of converter matte produced.

The building is served by two 50-ton Morgan cranes having a travel speed of 250 feet per minute. They are used for charging matte scrap and flux into the converters casting the Bessemer matte 'sculling' ladles and loading excess scrap into steel cars. The converter slag is handled by a 'Dinkey' locomotive on a narrow-gauge track and the matte is brought from the reverberatory plant in the same way. The matte from the furnace is brought in over a short narrow-gauge track connecting the two buildings, the ladle being placed on a car operated by an endless cable. As already mentioned, the furnace-matte pots are of sectional cast iron and have a capacity of about eight tons. Similar pots are used for the reverberatory matte. The slag pots are also of sectional design, but the side sections are of cast steel instead of cast iron. They have a capacity of about nine tons, and are occasionally used for reverberatory matte as well as for converter slag. The slag pots are held in a yoke on the slag cars and can be turned down to spill their contents on the ground, or can be lifted by the crane for pouring converter slag into the settlers in the furnace-building or for pouring reverberatory matte into the converters.

The finished converter matte is poured into cast-iron moulds of which there are fourteen situated in a matte shed along one side of the main building. Each mould is about 36 ft. long by 6 ft. wide, and when full, holds matte to the depth of about 4½ in. There are eighteen sections in each mould, and every third section has a rib on the upper side extending across the mould except for an 18-in. gap in the middle. This rib creates a line of weakness across the matte after it has solidified and facilitates breaking. Before the matte is poured into the moulds a cast-iron lifting-block with an aperture for a hook is placed in the centre of the area marked out by a pair of ribs, and the moulds are given a lime wash to prevent the matte sticking to the iron. Soon after the matte has solidified, and while it is still quite hot, an air lift which travels on an overhead I-beam is hooked to the lifting block and a slab of matte about six feet square and four inches thick, weighing about 1½ tons, is picked up and transferred to a set of grizzlies at the floor level. Below the grizzlies is a small pocket in which stands a side-dump steel car that has a capacity of about 2½ tons of matte. There is one of these loading pockets at each end of the matte shed. The matte is broken by sledges to pass through the grizzlies and when the car has been filled it is drawn up a short incline by an electrically operated hoist into a standard box-car for shipment to the refinery at Bayonne, N.J., or to Port Colborne, Ontario.

At one end of the main building there is equipment for handling and storing the clay and flux required in the operations. There are two Ruggles-Coles rotary driers through which all flux passes before being elevated to the storage bins. These bins hold about 700 tons and have compartments for quartz and mine rock, the fluxes used. By means of a system of small hoppers and conveyor belts below the bins, straight quartz or mine rock, or any desired mixture of the two, can be sent to the converters. The flux is discharged into a 10-ton steel box which has lifting trunnions by which it can be handled by the crane. Two Carlin pug-mills prepare all clay required at the different departments of the smelter. Most of the clay used is obtained locally, but a small amount of fire clay is used for 'budding.'

Converters are started with a charge of 70 or 80 tons of furnace or reverberatory matte, and 5,000 or 6,000 lb. of flux. The first blow is continued until a good slag has formed, which may require an hour or more. Subsequent blows are usually of 35 or 40 minutes' duration. If everything is running as it should, this is sufficient time to raise a pot of slag. The skim-

ming of the slag and the charging of fresh matte and flux takes from 10 to 15 minutes. The operations of blowing, skimming and charging are treeated until the converter has received from 300 to 400 tons of blast furnace and reverberatory furnace matte. The amount of matte that a converter will take depends on the grade of the matte, and the condition of the interior of the converter. The accretions on the sides and walls, and the accumulations at the bottom of the shell reduce its capacity after it has been in commission for a few months, and it is found advisable to shut the converter down periodically to have these cleaned out, even though no repair to the brick is required.

When all the iron in the matte charged has been oxidized, no further slag rises, and any additional blowing simply reduces the sulphur content. This, however, cannot be carried on to the extent of removing all the sulphur, but matte containing only 13% sulphur is readily obtainable if desired. It is found that practically all the iron is removed when the grade of the remaining matte is about 75% CuNi., but at this plant it is customary to raise the grade to 79% or 80% before casting. Matte of this grade will contain from 0.30% to 0.50% iron, the balance being almost entirely sulphur. Further blowing has very little effect on the iron. The amount of iron oxidized per minute of blowing is about 190 lb. and is used as a measure of the efficiency of the converter work.

As already explained, under the blast furnace practice, part of the converter slag made is poured into the furnace settlers. This amounts to about 70% of the total, the remaining 30% consisting of the 'sculls' from the ladles, and a portion that is poured on the ground and smelted as part of the blast furnace charge.

The slag usually made contains about 27.5% silica, which is higher than is found at the majority of copper smelters. It is quite feasible to make the silica 20% or even lower, and it is advantageous to do so from the standpoint of the converters alone, but when the practice of pouring the slag into the settlers was adopted it was found necessary to keep the silica around

27% in order to get the best results. A low silica converter slag soon caused the settlers to fill in with a sticky mass not conducive to good settling.

As mentioned above, quartz and mine rock are the fluxes used. The former contains about 91% silica, but no metal values, the latter about 53% silica and 1% copper-nickel. As the rock is an otherwise waste product from the picking belts at the mine, but contains some metal value, it is advantageous to use a certain amount of it, but its low silica content, none of which is present as free silica, makes it an inefficient flux. It is found possible, however, to use it in about equal quantities with the quartz, but for the last three or four blows of each charge it is preferable to use quartz only. The silica content of the average mixture used is about 72%, which, after fluxing, its accompanying bases to a 27.5% silica slag, leaves about 62% to do useful work in the converter.

POWER

Hydro-electric power, developed at High Falls on the Spanish River, is used at both mines and smelter. High Falls is a small settlement reached by a company-owned spur from Turbine, a flag-stop on the Soo branch of the C.P.R. and on the Algoma Eastern about 28 miles from Sudbury.

The river-flow past High Falls is not sufficient at all times of the year to furnish enough energy. To offset this disadvantage, a natural storage has been developed on the upper reaches of the Spanish river in the vicinity of Biscotasing. In this neighborhood are situated several large lakes, upon which the lumbermen years ago had built water storage and regulating dams to facilitate the delivery of logs to their sawmill in Biscotasing. The right to use these dams for conservation purposes has been acquired and the structures themselves have been greatly improved. Their use in regulating the supply of water, while very materially improving the stream flow, is, however, necessarily wasteful because of their distance and the three and a half days' time required for the water to reach High Falls.

A new concrete dam is now under construction, about three-quarters of a mile above High Falls power-house. This structure will create a large lake 25 miles long, which, although not duplicating the Biscotasing storage, will, on account of its position, provide for efficient regulation of the water, with practically no wastage. The actual flow, save in flood periods, will come through three large automatically operated Johnson valves. The control for these valves will be obtained by floats placed below the new dam on the forebay above the water-wheels of the High Falls power-house. This control will be transmitted electrically to the valves themselves. The new storage will also conserve the summer rains over at least three-fourths of the Spanish River drainage area above High Falls. Thus, except in the three months' period of extreme high water, the best average flow of the river can be maintained with practically no loss.

There are two power houses at High Falls. Both are steel frame buildings with brick walls. The older building has a wood roof 4 inches deep with tin roofing, but on the new structure the roof is of tile with a built-up Barrett specification covering. The floors are plain and of reinforced concrete where necessary. In the new building the window frames are of steel, with wire glass. Only in exceptionally cold weather has it been found necessary to heat these buildings, and for this purpose in the old building there is placed a hot water heating boiler.

In the new power-house there is a vertical type I. P. Morris water turbine rated at 7,500 H.P., driving a 5,500 K.V.A. Westinghouse generator. This machine has an over-all efficiency of 89% at full load. In the old building there are four horizontal type I. P. Morris water-wheel turbines with Crocker-Wheeler generators having a nominal rating of 2,000 K.V.A. each.

In the old building there are also two exciter units of 320 H.P. each and two triplex oil pumps, one a spare, furnishing hydraulic power for the governors to control the supply to each of the water wheels. Back of the wheels, on a high platform, is the switching equipment. All of the generators' output from both buildings is here distributed and handled by remote electric control through the various breakers. These breakers are all situated over the transformers to the rear of the switching and alongside the substation, together with the switching performances of the machines. The electric energy developed is a 3-phase, 25-cycle current and is generated for the transformers at a nominal voltage of 2,600. There it is stepped-up to 33,000 volts for transmission to the mines and the smelter.

The transmission line consists of two sets of three No. 1 B. & S. gauge copper wires arranged with equilatoral triangle 4-ft. spacing. The wires are supported by stands made of A-poles with a common cross-arm. The stands are spaced at about 150 feet and number about 1,150 in all. The poles are untreated and, after 14 years of service, are seemingly good for as many more years. Practically no replacements have been made. The cross-arms have been painted the usual mineral red; during the current year it was found necessary to replace about 10% of the cross-arms on account of dry rot.

The current delivered to the mine at Creighton and to the smelter at Copper Cliff is stepped-down at these points to 2,200, 550, and 110 volts for various uses. The lighting circuits everywhere are 110 volts and most of the motors about the plants are 550 volts. Motor generator sets furnish direct current to charge the battery and trolley locomotives underground, on the furnace charge floor, and to dump slag pots at the smelter. The 2,200-volt circuits drive the larger machines in the power-houses at both mines and smelter.

At the mines, the maximum portion of the power is consumed by the hoists, the air compressor, and the rock-house motors. In the hoist-house, the 1,500 K.V. capacity flywheel motor-generator set furnishes D.C. energy for one 1,800 H.P. Wellman-Seaver-Morgan hoist. In the compressor building, two Ingersoll-Rand compressors use 923 rated K.V.A. each at full load, and a Belliss & Morcom compressor uses 900 K.V.A.

at full load. Each of these furnishes approximately 5,000 cu. ft. of air at 100 lb. per square inch. In the rock-house, about 500 rated horse-power is used when in full operation.

In the event of the current being interrupted from any cause, there is in constant readiness at the smelter an auxiliary steam plant capable of furnishing 1,200 boiler horse-power at 160 pounds pressure.

SAFETY AND WELFARE WORK

This paper would not be complete without at least a brief description of the safety and social work carried on by the company for the protection and betterment of its employees.

Accident prevention was organized as a separate department in July, 1913, and placed in charge of an experienced safety engineer. The movement was decidedly successful and the accident rate was considerably lowered. In 1915 the Workmen's Compensation Act became effective in Ontario, and the work of reporting accidents to the Board was added to the duties of the Safety Department. In this way a record is kept of each accident, from the date of its occurrence until final compensation is paid by the Board and the case settled. Assistance is given the employees in their correspondence with the Board and every effort made to follow up the cases, to see that workmen, prevented by disability from following previous occupations, are given employment at suitable work; and also care is taken to prevent malingering and lessen in every way the great economic waste caused by lost-time accidents.

Hospital and Medical Attention.—In order that employees and their families may have proper medical attention in case of accident or sickness, the company has provided a commodious hospital equipped with the latest approved devices for the treatment of surgical and medical cases.

Eight experienced doctors are employed and one is on duty continuously, ensuring prompt attention in case of accident.

Education and Entertainment.—In the town of Copper Cliff, primary education is provided for by a large public school of modern design and equipment, where a school population of over six hundred is accommodated. All grades of the public school course are taught, and on the completion of this course the High School at Sudbury is available for Copper Cliff pupils.

At Creighton and Crean Hill mines, modern public schools have been erected. It may be noted that the population of the nickel district is a most cosmopolitan one; hence the number of nationalities represented by the pupils at these schools is remarkably diverse.

At Copper Cliff the company has erected what is conceded to be one of the finest club buildings in the province. All employees are eligible for membership. The club is provided with bowling alleys, pool and billiard tables, swimming pool, reading room and a large assembly hall where weekly dances and card parties are held. Membership in the club is restricted to boys over eighteen and girls over sixteen; for those under these ages the Junior Copper Cliff Club is available. In this club a specially qualified Lady Superintendent teaches those who desire it domestic science, dancing, deportment and physical training. In connection witb the club there is also a Cadet Company of over eighty members, a well-equipped gymnasium with classes for boys of all ages, also a class in boxing, wrestling and physical training for workmen.

Insurance.—By the change in the Dominion regulations respecting group insurance, whereby the writing of insurance of this nature in Canada became permissible, the Company was enabled to protect the families of employees by issuing policies, effective November 1st, 1919, insuring every employee who had completed one year's continuous service, for the sum of $500, with an increase to $800 at the end of two years' service; increased again to $1,200 at the end of three years' service, and of $1,600 at the end of four years' service; while five years' continuous service is recognized by the issue of a policy to the maximum value of $2,000.

Pensions.—In order to provide for aged employees who have given the company long service, the plan of pensioning employees who have become incapacitated after twenty years' service is in force. Pensions are paid from the general fund in the form of monthly payments, the amount paid being based on the average earnings (calculated on a full time basis) of the pensioner during his last year of active work. Upon retirement, after twenty years of continuous service, a pension representing one-half the average earnings of the recipient is paid. If there has been thirty years of continuous service, the pension is equivalent to 62½% of average earnings, and for longer service than thirty years the percentage is proportionately increased. Either old age, sickness, injury or incapacity from any cause may be considered sufficient reason for granting a pension to an employee who has completed twenty years of continuous service with the company.

DISCUSSION

MR. H. M. PAYNE: May I ask what steps do you take to control the hanging wall in the lower levels?

MR. J. C. NICHOLLS: Above the 10th level we hold the wall by means of waste. The wall as encountered has a small proportion of slabbing. Some of this gets into the ore and requires to be sorted out.

MR. O. HALL: What is your tonnage production per man, per day, underground, at the mine?

MR. J. C. NICHOLLS: About six or seven tons.

THE CHAIRMAN (Dr. W. G. Miller): I am sure we are all much indebted to Mr. Nicholls for his able presentation of this highly interesting subject. This paper is the most complete description of mining and smelting operations as carried out at the Copper Cliff mines that has yet appeared.

MR. JACOB W. YOUNG (*Communication to the Secretary*):— The section of the "Geology of the Creighton Mine," in the above paper recalls to mind my oft-recurring intention to record some of my thoughts and conclusions on the origin of the copper-nickel sulphides, based upon a reading and study of the somewhat voluminous literature on the subject. Not being so situated at present as to have access to the many publications dealing with those deposits, as well as other reference works, I will here only outline my views, hoping that at no far distant date to be so situated that I can largely amplify them, and present more or less in detail the evidence upon which I base my conclusions.

The description of the orebody by the authors of the paper is not such as would lend weight to the idea of magmatic segregation *in situ*. The shape of the orebody as developed is entirely against that supposition, for had the copper-nickel deposits of the Sudbury region been formed by an action similar to that of a matte-fall from the liquid norite magma, one would look for a more or less even distribution of the copper-nickel sulphides over the whole area of contact between the norite and the underlying formations, and thus a more or less continuous outcrop of ore along the contact would be predicated, and a continuous sheet of it downward, comparable somewhat to the Rand banket for regional extent, would have been formed. Instead of this, we find at the greatest mine in the district, where mineralization has been equal to, if not greater than, any found elsewhere along the contact, that the deposit is roughly an elongated pipe-like orebody, "oval to narrow lenticular" in plan, and "over 2,500 ft." on the pitch.

At the Creighton orebody, the authors state: "The cross-section varies in form from oval to narrow lenticular, with wavy outlines, which, in some places, are strongly marked and irregular." This citation and a reference to the plan of the 14th level (Fig. 3), disposes of the idea of segregation *in situ*, at least for this orebody.

To further strengthen this idea it is stated, regarding the orebody that, "It is usually sharply defined against the country rocks." By magmatic segregation *in situ* there would of necessity be a certain amount of gradation from solid sulphides to unmineralized norite.

The statement, "There is no gangue in the ordinary sense of the word, but inclusions of rock are abundant in many places," shows that the sulphides were introduced into a shattered zone or fissure which had been formed after the norite had solidified. The intruded sulphides filled around and enclosed the rock fragments of the shattered zone. The fissure or zone in which the copper-nickel sulphides occur, does not confine itself to the norite-granite contact, but occurs at places out in the footwall formations, which would hardly be expected from segregation *in situ*. The evidence this article presents on this phase of the matter is: "Geological work at the mine had disclosed facts that indicate the origin of the ore by its intrusion in molten condition along a plane of shearing in the footwall rocks adjacent to the norite, after the latter had solidified." Thus the ore deposit is in the footwall rocks and mostly, presumably, entirely out of contact with the norite, and the field evidence shows it was introduced after the solidification of the norite. That the norite had long solidified before the intrusion of the ore is shown by the following quotation: "The most interesting evidence is offered by a dike of comparatively fresh younger norite that intrudes the main body of norite and its footwall rocks. It is itself intruded by the ore and also altered by it."

A further evidence of the intrusive character of the ore is shown by the following: "The alteration is an unusual variety of contact metamorphism. It appears as a dark margin, varying in width from a quarter of an inch to six inches, against the ore. It is found in every rock with which the ore comes in contact, except the diabase dikes, which are younger."

The off-shoot deposits are surely not explainable by the magmatic segregation hypothesis, as applied by Barlow and others. Then, in some of the copper-nickel deposits of the Sudbury region, there are, at places, evidences of pneumatolitic and metasomatic action (metamorphic action), on a larger scale than the contact metamorphism referred to in the Creighton mine. This would seem to be against the idea of the sulphides having been introduced in a molten condition as the staff of the International Nickel Company infers.

All of the metamorphic action noted by the various writers on the Sudbury copper-nickel deposits could be well accounted for by the action of such mineralizers as would accompany an intruded colloidal gel of the iron-copper-nickel sulphide such as I consider, and believe, was the condition of the sulphide material when it was intruded into the shattered zones and fissures along the contact, and near the margin of the norite in the underlying rocks.

That these deposits are a magmatically segregated product, the evidence presented by many writers seems well to substantiate, but that it was segregated *in situ*, or that it was intruded in a molten condition, the evidence does not at all bear out. To superimpose pneumatolitic action upon the moltenly intruded sulphides would introduce complications of no inconsiderable sort, unless a considerable time interval was allowed before the pneumatolitic process was introduced.

My conception of the process is that beneath the Sudbury region the magma was unusually rich in copper and nickel, especially nickel, and also contained much iron and sulphur. By some process, for which I, at present, can offer no reasonable explanation, the iron-copper-nickel sulphides were segregated to themselves in large reservoirs.

When, from causes internal or external, fractures were formed in the earth's crust extending down to these reservoirs with incidental compressional stresses, the more or less pasty and jelly-like colloidal sulphides would have been forced up and along the fissures and shattered zones filling up all the available space whether it was between cleanly separated walls or in the interstices between the rock fragments of shattered zones.

Instead of one big reservoir from which the material for all the Sudbury copper-nickel deposits drew their material, a local reservoir for each, or for several not too distantly separated

deposits would seem to be the more logical conception of the reservoir source. This arrangement would permit of a greater variation in the amount of mineralizers present in the various reservoirs. With various amounts of mineralizers accompanying the collodial sulphides from the different reservoirs, different amounts of metamorphic action would be produced from place to place, and even in a particular deposit, for, once the fissures were formed, and the intrusion of the colloidal material up into the fractures began, it is possible that there would be a tendency for a separation of a portion more highly charged with mineralizers from a part with a smaller amount of mineralizers. That part with a minimum of mineralizers may be so near the neutral point as to produce little or no alteration in the adjoining rocks.

This hypothesis of magmatic reservoirs of colloidal sulphide charged with mineralizers, and the sub-hypothesis of a certain amount of differentiation of the material in any particular reservoir after the intrusion into the upward extending fissures had once begun, will account in a reasonable and logical way for all of the various and many differences noticeable in the various deposits as well as in individual orebodies.

RECENT ORE CONCENTRATION DEVELOPMENTS BY THE CONSOLIDATED MINING AND SMELTING COMPANY OF CANADA, LIMITED

By R. W. DIAMOND

Western General Meeting, Vancouver, November, 1919.

The great advance in recent years in the science of ore concentration by the oil flotation process has rendered possible the profitable treatment of ores which had formerly proved too refractory for successful treatment by gravity concentration methods or by the earlier flotation processes. In consequence, early in 1917 experiments were begun by the Consolidated Mining & Smelting Company of Canada, Limited, in the treatment (especially by flotation) of the low-grade gold ores of Rossland, and the complex lead-zinc-iron ore of the Sullivan mine at Kimberley, as well as other company ores which might warrant a trial.

Sullivan Ore:—At first efforts were directed mainly towards solving the problem of concentration for Sullivan ore. An experimental mill, having a daily capacity of 250 tons, was built and equipped. The equipment comprised: gravity concentration tables, magnetic machines, one large Minerals Separation standard machine, and the requisite auxiliary mills, screens, thickeners, and filters.

The Sullivan ore is a finely crystalline, complex mixture of lead, iron and zinc sulphides, carrying only from 3% to 6% insoluble matter. An idea of the intimate association of minerals may be obtained when it is known that grinding the ore to 65-mesh liberates only 35% to 45% of the galena, and inspection under the microscope of material ground to 200-mesh shows grains containing both galena and blende.

From a preliminary series of laboratory tests it was known that a good separation of zinc could be made from the lead and iron by flotation of the ground ore after it had been roasted at a temperature which almost completely oxidized the lead and iron sulphides, but which oxidized only a small percentage of

the zinc sulphide. This roasting operation is commercially practicable owing to the wide range between the roasting temperature of blende and that of iron or lead sulphide.

It was therefore decided to duplicate this treatment in the test mill. The roasting was effected either in the 25-ft. Godfry furnaces or in Wedge furnaces, and, after cooling, was subjected to treatment by flotation. It was demonstrated that a separation was possible, but for practical reasons this process was abandoned. A long series of large scale concentration tests along various lines was then made during the course of which it was satisfactorily demonstrated that wet magnetic separation of pyrrhotite from blende was commercially possible. While most of the iron in the Sullivan ore is present as pyrrhotite, it is so feebly magnetic that the advantage of wet magnetic separation was questionable, and it was not until a novel preparatory heat treatment was suggested by Mr. C. Thom, of the company's staff, that its real possibilities were apparent. A complete series of laboratory tests was made, and these were so encouraging that an entirely new 150-ton test mill was built and equipped with special machines developed to suit the needs. This plant has since been increased to a capacity of 250 tons per day and is now about to begin operation at a further increased capacity of 600 tons per day.

Laboratory tests on preferential flotation have been carried on continually for effecting a separation of Sullivan ore into three products, namely: lead sulphide, zinc sulphide, and iron sulphide. The results are very gratifying and to some extent success commercially is assured.

As a result of the long series of tests outlined above, a satisfactory treatment by concentration is possible for the Sullivan ore; and this will consist of a combination of the following treatments, all on finely ground ore: (1) table concentration, (2) wet magnetic separation, and (3) preferential flotation.

Simultaneously with the above development, large scale flotation tests were inaugurated on the tailing from the electrolytic zinc plant. The leached tailing contains some zinc sulphide, and flotation tests showed that this could be recovered and returned to the process advantageously. Accordingly, two short mineral separation machines were installed and are now in satisfactory operation.

Rossland Ore:—Much study and attention has been directed to the concentration of low-grade Rossland ores. During the last twenty years at the Le Roi No. 2 a large tonnage of ore has been milled; the original treatment was by the 'Elmore Oil Process.' This plant, however, was later converted into a simple gravity-table mill.

The Centre Star and War Eagle companies did much experimental work on concentration and cyaniding, following which the Rossland Power Company was formed. This company built a large concentrator near Trail, but the plant proved to be a failure largely because the treatment practiced in the tests was not followed in the mill.

The Le Roi Company also built a mill for work on low-grade ore. This mill was later converted by the Consolidated Mining and Smelting Company into a test mill, where flotation tests were conducted for several months under Mr. Graham Cruickshank, at a time when flotation was still in its infancy in America. The tailing from these tests contained enough gold to make cyanidation necessary. Good extractions were assured, but the cost of milling and cyanidation were so nearly the same as the cost of smelting that, for a time, this work was abandoned.

In 1917, flotation treatment of the Rossland ores was again investigated at Trail by Mr. John Bray, with results similar to those previously obtained at Rossland, indicating that both flotation and cyanidation were necessary.

When the concentration problem of the Sullivan ores was practically solved, the concentration of Rossland ores was investigated by the writer, who found from laboratory tests, that good recoveries were obtainable by a combination of flotation machines and tables. The ratio of concentration, however, was quite low. It was also found that fair recovery could be made, with a much higher ratio of concentration, by flotation treatment only; the recovery, however, was such as to make the advisability of adopting this treatment questionable. It is doubtful whether this work would have been continued but for the fact that the original Sullivan experimental flotation mill was available and could, without much difficulty, be adapted to the concentration of these ores. This mill was accordingly equipped with tables and flotation machines so arranged that both could be used, or flotation machines only. Before the changes in the table equipment were completed in this plant operations were commenced with the flotation machines alone. From the outset as high a recovery was obtained in this manner as was obtained by both tables and flotation in the laboratory tests, and at the same time a much higher ratio of concentration was secured. On completing the installation of the tables, tests (each of a few days' duration) were made, using both treatments, but with no additional recovery and with a much lower ratio of concentration. Careful study of the flotation conditions since that time has resulted in securing a still higher ratio of concentration by flotation alone, with as good or better recovery. Fortunately a mill equipped for this treatment will be cheaper and more simple in operation; and plans are now being completed for a 1,500-ton concentrator.

DISCUSSION

THE CHAIRMAN (Mr. E. E. CAMPBELL): The value of ore concentration is obtaining more recognition than ever before. I know of very few metallurgical works that would not be benefited by the addition of concentration. Preferential flotation is a most important refinement of the main application of the flotation of sulphides, and it is being used more widely every day. At Anyox, where the ore averages about 30% sulphur, a hundred-ton experimental mill has been in operation for more than a year, and it is probable that the results obtained by the selective work done there, will radically change the present methods in vogue at that place. The ore consists of pyrite with a small percentage of chalcopyrite, and the main problem is to separate these two metallic sulphides.

MR. F. H. HOLLER: It would interest everyone to hear what the ratio of concentration is. If Mr. Diamond mentioned that, I missed it.

THE CHAIRMAN: I regret that Mr. Diamond is not present and there is no one here who can reply on his behalf. I, myself, am curious to know what this ratio would be. The success of all flotation is the ratio of concentration, and if a high ratio can be obtained with a good recovery, the method is bound to be successful. Another matter that, to my mind, calls for discussion is that of the three products, lead, zinc and iron sulphide, for, unless there is an important reason for saving the iron, it should be relegated to the tailings. According to the paper, the pyrite or pyrrhotite is a product of the mill, and I do not see why that should go into the tailings.

MR. E. W. GEPP: The necessity of removing the iron is in order to get a high-grade concentrate. The iron is there as pyrrhotite and if allowed to remain, there would be a much higher percentage of insoluble material. It carries practically no value. The idea is to reduce the amount of iron and zinc.

THE CHAIRMAN: I understood from the reading of the paper that there were three products, and I could not see why pyrrhotite should be claimed as a mill product.

CANADIAN CYANIDE, ITS MANUFACTURE AND UTILIZATION

BY W. S. LANDIS

Annual General Meeting, Toronto, March, 1920.

The alkali cyanides play a very important part in the extraction of the precious metals from their ores. In addition to this use in the recovery of precious metals, cyanides also find a very considerable application in the chemical industry for the production of ferrocyanide, in the dye industry, in electroplating, in case hardening of steel, in fluxing powders, and in fumigation. Thus the metallurgist must compete in the open market with many other users for his supply of this reagent.

OLDER COMMERCIAL PROCESSES

Up to 1917, there were only two well-defined and developed processes for the production of alkali cyanides. The most important of these is the Castner Process, in which the raw materials consist of metallic sodium, ammonia, and wood charcoal. The reaction takes place in three stages, in the first of which dry ammonia gas is conducted over metallic sodium to form soda amide. The soda amide is next heated with charcoal to form sodium cyanamide. Sodium cyanamide is again heated with additional charcoal to form sodium cyanide. The resulting product is obtained in a melted form and of commercial purity, averaging 96% to 98% for the higher grade products. Metallic sodium is made by the electrolysis of caustic soda or of sodium chloride. It is a comparatively expensive reagent and at present there is very little likelihood of any material cheapening of the sodium. Ammonia is obtained in the form of a fairly pure solution in water, either from the coke ovens, gas plants, or cyanamid works. Its price, to-day, is very high, owing to certain existing economic conditions; but in the future it will possibly be cheaper. Nevertheless, even at the lowest price at which ammonia has been sold, it must be regarded as an expensive reagent.

The Castner process has never been utilized in Canada, but much of the cyanide consumed in the Dominion has been derived from plants employing the process in Great Britain, the United States, or Germany.

The second commercial process uses as its raw material the vinasse produced as a residue in beet sugar manufacture. By distilling this material and treating the vapours obtained, hydrocyanic acid is produced which can be absorbed in caustic soda solution and the cyanide recovered therefrom. The process is complex and requires very exacting chemical control. The recovery of cyanide from solutions is a very difficult manufacturing problem, and as a result its operation has never been developed outside of Germany, and the cyanide thus produced only competes in a small measure with that obtained by the Castner Process.

Before 1917, Canada did not produce cyanide, but was entirely dependent upon importation. The processes hitherto employed have either used expensive raw materials or have been complicated in character, and hence the price of cyanide to the Canadian consumer has of necessity been high.

The recent great war seriously affected the supplies of cyanide for metallurgical use in this country. Production of the large German plants was excluded from commerce. The great demands occasioned by the manufacture of munitions for electric power, caustic soda, and ammonia, interfered greatly with the production of plants in countries other than Germany. As a result, in the fall of 1916, the price of cyanide rose to almost unbelievable figures, and the supply practically vanished.

DEVELOPMENT OF AERO BRAND CYANIDE

The discovery, in the laboratory, that calcium cyanamid could be transformed into cyanide, antedated by fifteen years its commercial application. This was done, however, in the fall of 1916, by the American Cyanamid Company of Niagara Falls, Ont., and the first units of a plant for this purpose were installed and in operation about January 1st of the following year.

Cyanamid is a nitrogenous material containing calcium cyanamid, calcium oxide, graphitic carbon, and various minor impurities obtained from the raw materials used in its production. Its principal use has been as a fertilizer, it being one of the cheapest known sources of combined nitrogen.

The process of its conversion into cyanide involves the mixing of cyanamid and common salt and the melting of the mixture in a peculiar type of single-phase electric furnace, whose operation is continuous and largely mechanical, the power requirements for this melting process being very moderate. Temperatures are not high, and wear and tear on the apparatus is not excessive. The furnace is tapped continuously, and the molten product runs to a cooling device, converting it into thin and extremely brittle flakes of a dark gray to glossy black colour. These flakes are packed into caustic soda drums and shipped to the consumer.

In the early experimental work, it was hoped to obtain a product containing about 12% of equivalent sodium cyanide content. The first experimental processes, which operated on the batch principle, amply met expectations both as to quality and output. But the quality was further improved as experience was gained and the process improved, and by the close of 1917 the equivalent sodium cyanide content had been increased to 22%, about four tons of sodium cyanide being contained in the material produced each day. In 1918, the batch furnace was abandoned for one of the continuous type, and by the end of that year a material containing approximately 30% equivalent sodium cyanide was produced.

In July, 1919, after a temporary suspension of the operation, still further efforts were made to improve the quality of the product and cheapen its cost of manufacture, with the result that the average grade of the product was increased to between 36% and 37% equivalent sodium cyanide. Toward the close of the year 1919, the quality of the product was again improved and several hundred tons of material containing about 46% equivalent sodium cyanide were produced.

The 36% grade is styled Aero Brand Cyanide Grade X, and the product above 45% Aero Brand Cyanide Grade XX. The current year's demand for these products is the equivalent of nearly thirty million pounds.

PROPERTIES OF AERO BRAND CYANIDE

The product consists of a dark gray to black shiny material in the form of thin scales, varying from $\frac{1}{16}$ to $\frac{1}{32}$ of an inch in thickness and from ¼ to ½ square inch in area. The old system of designating this material by expressing its content of available cyanogen in terms of either potassium cyanide or sodium cyanide has been preserved.

In addition to the available cyanogen, the product also contains free lime, sodium chloride and, in small amounts, impurities such as cyanamid, miscellaneous calcium compounds, and such impurities as are picked up in the ash of the coke used in the cyanamid process. These impurities do not in any degree affect the efficiency of the cyanide.

Aero Brand Cyanide, like all other cyanides, undergoes decomposition on contact with moist air, carbon dioxide, and water. It is, therefore, packed in tight metal containers. The standard package contains about 375 pounds and is marked to indicate the nature and grade of its contents. If the air is excluded the contents will keep indefinitely, but if moisture is permitted to enter or the material is exposed in a pile to the air, decomposition will occur and a loss of cyanide will take place. The large exposed surfaces of the flakes of this product are affected by exposure to air and moisture to a greater extent than the ordinary flat slabs of the usual and higher grade cyanides.

The solubles of Aero Brand Cyanide dissolve readily in water. Care must be taken to agitate thoroughly while dissolving the cyanide as otherwise the flakes will sink to the bottom of a dissolving tank. The lime present will hydrate with a considerable local evolution of heat; the mass of damp flakes

will then decompose and the wet mass will give off ammonia with consequent loss of cyanogen.

Cyanamid contains a small amount of calcium carbide. Occasionally, very small quantities of additional carbide is charged in the cyanide furnace. As a result, Aero Brand Cyanide contains upwards of 1% of calcium carbide and when thrown into solution evolves acetylene gas. The quantity of gas evolved is quite small, and no difficulty has been experienced from it, but its odour is quite noticeable. Acetylene is a non-poisonous gas, but nevertheless the ventilation of the mill building should be such as to prevent an accumulation of the gas.

As mentioned above, the soluble constituents dissolve very readily in water. There is, however, a certain proportion of insoluble material consisting of lime, graphitic carbon residue, and various minor impurities. It was anticipated that this carbonaceous residue might give trouble in the mill circuit, causing precipitation of the precious metals. During the first few months of the use of Aero Brand Cyanide this insoluble residue was filtered out and only clear solutions fed to the treatment tanks. It was soon realized, however, that the residue did not have any precipitating effect on the precious metals in the solution, and the practice of filtration was therefore abandoned. Aero Brand Cyanide, however, must not be dumped into the pregnant mill solution as the calcium carbide it contains evolves acetylene on contact with water, and acetylene will precipitate gold and silver. Since this gas is instantly set free on touching water, and where a thorough agitation exists no trace remains in solution, no difficulties whatever are experienced from this source, provided the addition has not been made to a solution carrying appreciable quantities of gold and silver.

MILL EXPERIENCES

The most extensive test, during which careful records were kept of the performance of Aero Brand Cyanide in the mills, was in Mexico. It was introduced into two mills, each treating dissimilar ores, and after 2½ years experience a very comprehensive report covering its behavior was submitted by the operating company. At first, the cyanide was ground in a small tube-mill in order to get it into solution, the solution afterwards being filtered. This filtering was later abandoned and, instead, lead oxide was added. Finally the regular Aero Brand Cyanide was fed, together with the pulp, into the large tube-mills regularly employed for grinding the ores, and no additions were made. The report states that in both mills recoveries were equal to those obtained from 98% sodium cyanide. There was no change is respect of zinc consumption nor in the grade of bullion produced. The lime in the Aero Brand Cyanide was valuable to the extent that it was available, but a change in the character of the ores recovered at the mills did not enable definite conclusions as to whether there was any ultimate actual saving in lime. In every case Aero Brand Cyanide was just as effective as the 98% cyanide to the extent of its actual cyanide content.

The only actual difficulty experienced in this connection was due to an unusual condition arising, in that at one time the mill solutions were increased in concentration to about 1% KCN. Since the only solution run to waste in this mill was a small amount not washed out of the slime cake, there was a decided increase in the concentration of the chlorides found in Aero Brand Cyanide in the mill circuit. This prevented lime going into solution and reduced the protective alkalinity. But since this report was prepared the quality of Aero Brand Cyanide has been very materially improved and the quantity of chlorides correspondingly reduced.

Aero Brand Cyanide had been extensively used in the silver districts of Ontario. The same difficulty of concentration of the chlorides has been experienced in Ontario was as noted in Mexico, but the difficulty was very quickly overcome by the addition of a precipitating agent which precipitated the lime and permitted it to be separated on the slime filters. This is a peculiar condition arising from the very high cyanide consumptions met with in the treatment of these silver ores.

Before introduction into the Cobalt district an extensive series of tests was undertaken in which Aero Brand Cyanide was compared with the 98% product heretofore used in that district. Assays indicated that the cyanide consumptions were somewhat lower for Aero Brand than for the 98% product and extractions were equally as good. As a result of these tests, including one conducted by the Haileybury School of Mines, its adoption by operators in the Cobalt district followed and they have now been using Aero Brand Cyanide many months with very successful results.

Following the introductory sales, repeat orders for carload quantities have been received from mills in California, Colorado, and Nevada, in addition to the districts mentioned above, and extensive shipments have also been made to mills in Arizona, Montana, Oregon, and South Dakota.

CONCLUSIONS

Canada, to-day, possesses a cyanide manufacturing industry which within three years has so grown and expanded that it is supplying the bulk of the cyanide consumed in the mining and fumigating industries of the United States, and will shortly lead in the Canadian and Mexican fields.

The product is made of raw materials that are either under control of the manufacturing company or are easily obtainable in the open market. They are very cheap in comparison with those used in the Castner Process. For example, common salt is available in the Niagara district on both sides of the International boundary, and the cheapest form of rock salt may be used. Cyanamid is a fertilizer material and is probably the cheapest source of fixed nitrogen to-day. It has always been sold in competition with nitrogenous fertilizers of like character. These conditions insure a most reliable source of supply of this Canadian product.

DISCUSSION

MR. W. R. ROGERS: Would Mr. Landis kindly tell us the highest percentage of sodium cyanide and its equivalent in potassium cyanide?

MR. W. S. LANDIS: There are two brands of Aero Brand Cyanide on the market, one is known as grade X., which contains, and is guaranteed for all sales purposes to contain, 36 per cent of equivalent sodium cyanide, NaCN. I multiply 36 by 132 and I thus obtain the potassium cyanide content of that material. The corresponding grade, X*m.* is guaranteed for freight purposes at the present time to contain 45% equivalent NaCN: multiplied that by 132 gives a result of about 57 per cent KCN. The grade XX is a material that we hope to improve. We expect shortly to make a product containing about 50 per cent NaCN.

DR. J. B. PORTER: Would not this process lend itself to the production of a higher grade cyanide?

MR. W. S. LANDIS: The material does not lend itself readily to the production of a pure 100 or 96 or 98 per cent. cyanide, for the simple reason there is some salt in it, and I believe, there is to-day no successful commercial process of separating salt from cyanide.

SODIUM CYANIDE FROM CYANAMID; AND SOME NOTES ON THE CYANIDING OF GOLD AND SILVER ORES

By HORACE FREEMAN

Western General Meeting, Vancouver, November, 1919.

The cyanide extraction process, as is well known, is based on the solubility of both gold and silver in a weak solution of either potassium or sodium cyanide. During the experimental stages in the development of this process potassium cyanide was employed as the reagent, but in actual practice it was soon realised that the less costly sodium, cyanide was equally as effective, and its use, therefore, became general. Nevertheless, even sodium cyanide was a relatively expensive chemical; and since the introduction of the cyanide process the aim of the chemist has been to reduce the cost of its manufacture in order that it might be supplied for metallurgical requirements at a lower price.

The most important constituent in cyanide is nitrogen, which occurs in limitless quantities in the atmosphere, but which is very difficult to bring into the combined state. In sodium cyanide the element, sodium, is somewhat costly to prepare. Prior to the outbreak of the war, the major portion of the world's cyanide was obtained from Germany and it was there prepared by using metallic sodium and nitrogen in the expensive form of anhydrous ammonia. Numerous, but commercially unsuccessful, attempts have been made in Germany to produce cyanide direct from atmospheric nitrogen; but success had been attained there in manufacturing a commercial fertilizer from atmospheric nitrogen. This product, known as calcium cyanamid, at the outbreak of war, was manufactured also on a large scale in Canada. In Germany, it should be noted, attempts to produce cyanide from this material had been unsuccessful.

For many years the writer had interested himself in the development of the cyanamid process, and had become convinced that cyanamid could be made to yield its nitrogen in the form of cyanide of sodium. Further research to this direction was strongly incited by the cyanide famine caused by the cutting off of German supplies. The scarcity of cyanide, for which the war was responsible, was an incentive to research in this direction, and, in collaboration with the American Cyanamid Company of New York, the writer undertook, in Vancouver, the research that led finally to the manufacture of cyanide from calcium cyanamid.

The method of manufacture, briefly described, is as follows: Calcium carbide is prepared by fusing coke and lime in the electric furnace; pure nitrogen is produced by the distillation of liquid air. The nitrogen is brought into chemical combination with finely crushed calcium carbide, producing a substance known as calcium cyanamid. The calcium cyanamid is then powdered and mixed with ordinary salt, which, in the cheapest possible form, supplies the necessary sodium. This mixture is heated rapidly to a white heat in a special form of electric furnace, and is then quickly cooled to below a red heat and packed in sheet-metal drums.

The Canadian plant of the American Cyanamid Company at Niagara is the first to produce cyanide commercially from atmospheric nitrogen by this process.

The product contains substances other than sodium cyanide that are helpful in the cyanide process and are frequently added when pure cyanide is used in the mill. This new form of cyanide is sold under the trade name of "Aero Brand," designating its derivation from the atmosphere. It contains upwards of 36% of sodium cyanide (equivalent to 48% of potassium cyanide), 43% of chlorides of calcium and sodium, and 15% of caustic lime. Thus, it provides cyanide for dissolving the precious metals; chlorides, to coagulate the colloids in the solution; and lime as protective alkalinity. It is sold on the basis of its cyanide contents and is delivered at the mines at a considerably lower price, pound for pound of contained cyanide, than the material heretofore imported from Europe.

The cyanide extraction process involves two distinct steps; first, the bringing of the precious metals into solution, and second, the recovery of the metals from this solution. This second step has been the subject of much research, but it may be said that much scope for investigation remains.

Any method of precipitating gold and silver from cyanide solutions should fulfil at least two conditions; i.e., (1) it should produce a high-grade bullion, and (2) should make possible a recovery or regeneration of the cyanide solvent that was employed in dissolving the precious metal. In addition to these points, it must, of course, be simple in operation, practicable, and economical.

Briefly, the chemistry involved in the process of precipitation by metallic zinc is as follows: When gold or silver dissolves in cyanide solution, it does so by reason of the formation of a double cyanide of sodium and the metal. The reactions for gold and silver in cyanide solutions are identical. In the case of gold, gold-sodium-cyanide, which remains dissolved in the solution, is formed. Before precipitation, this solution is filtered and clarified and zinc dust is added thereto. A chemical interchange takes place whereby the zinc is dissolved and gold is thrown down in the form of a black powder. The reaction involved is as follows:—

$$2AuCN.\ NaCN + Zn = Zn\ (CN)\ 2.\ 2NaCN + 2Au.$$

This means that the zinc forms a compound with the sodium cyanide in place of the gold. This zinc-sodium-cyanide remains in the solution and accumulates as the solution passes repeatedly through the cycle of operations in the mill. It will be seen that the cyanogen, which is introduced originally for the purpose of dissolving gold, finally becomes combined in zinc-sodium-cyanide, which is of no use for the purpose of dissolving gold. Furthermore, the metal (gold or silver) is thrown down as a black precipitate, forming a sludge and carries with it a considerable amount of unconsumed zinc and insoluble cyanide compounds. This sludge must be treated with acid to remove the excess of zinc—a dangerous operation, inasmuch as large quantities of prussic acid are frequently evolved,—and then must be dried, fluxed, and melted into bars of base bullion, which invariably give unreliable assays, creating some uncertainty as to exact value.

In studying the chemistry of the gold-cyanide solutions, it would seem that in order to recover the cyanide in the form in which it would again be available for dissolving gold, the obvious thing would be to replace the gold with metallic sodium instead of metallic zinc. The objection to metallic sodium is that it decomposes water with explosive force and therefore is useless as a precipitant. In investigating this subject, the writer was attracted by the properties of the alloys of sodium, more particularly those of lead, for reasons to be presently stated.

Alloys of sodium and lead are very remarkable. Sodium is a much softer metal than lead, yet the addition of 1% or 2% of sodium to lead results in an alloy hard enough for bearing metal. An alloy of lead and sodium containing 10% of sodium is so brittle that it may readily be ground by hand to fine powder with an ordinary mortar and pestle.

When this alloy is introduced into water, the water is still decomposed with a brisk evolution of hydrogen gas but there is no combustion or explosion. When the proportion of sodium is cut down to 5% or under, the sodium still reacts with the water but so slowly that the evolution of gas is barely perceptible.

These alloys, therefore, present a means of bringing sodium into the place of gold in the cyanide solution, and the writer has found that a total recovery of all the bullion in the solution may be obtained with perfect ease and with regulation of the consumption of sodium. At the same time, the cyanogen is brought back to its original state as sodium cyanide and therefore may be used again; in other words, a complete regeneration takes place as this reaction shows:—

$$NaCN.AuCN. + Na = 2NaCN + Au$$

When lead-sodium alloys are used for precipitating, the lead is not affected in any way; only the sodium dissolves and the precious metal is precipitated, yielding a mixture of gold and lead, containing upwards of 20% of gold. The recovery of this gold in a most desirable state of refinement is facilitated by the presence of the lead, for it is only necessary to dry, without any acid treatment, and cupel the lead-gold precipitate in the well-known cupellation furnace, sometimes used for refining the gold-zinc precipitates. The product is a bullion entirely free from base metals and ready for the parting process,

Several methods, employing common salt, have been developed, for the production of these sodium-lead alloys, For this purpose it is electrolized in the fused state over a cathode of molten lead.

DISCUSSION

MR. R. R. HEDLEY: Is this new solvent more efficient than potassium cyanide?

MR. FREEMAN: Mr Hedley is under the misapprehension that potassium cyanide is used commercially for dissolving gold and silver. Potassium cyanide has not been used on a very large scale in actual practice. There is, however a certain 'trade' commodity sold as potassium cyanide. Pure sodium cyanide, expressed in terms of potassium cyanide, is the equivalent of 130% of potassium cyanide. But there is a demand for 98% to 99% KCN for metallurgical use. It is extremely difficult to make potassium cyanide. It is more difficult to prepare than the sodium cyanide, but the German and British manufacturers take the sodium cyanide and add to it a certain amount of potassium chloride. This causes no reaction, the addition merely reducing the KCN content from 130% to 99%, but the miner is satisfied because it contains potash.

MR. G. S. ELDRIDGE: I do not think Mr. Freeman emphasised sufficiently the importance of the new alloy to which his paper refers. If, in the case of ores consuming much cyanide, the cyanide can be recovered, it makes a very important difference to the industry.

MR. FREEMAN: A high consumption of cyanide is frequently due to the presence in the ore of copper, which forms a cyanide somewhat similar to that formed with gold and silver. If copper cyanide were precipitated with lead-sodium alloy, sodium cyanide would be regenerated and the copper would be thrown down with the gold precipitate, but the copper would be eliminated in the simple process of cupelling that gold precipitate.

MR. HEDLEY: You did not state the source of the calcium oxide in the 'Aero' cyanide.

MR. FREEMAN: Calcium carbide is prepared by fusing calcium oxide with carbon. The charge in the carbide furnace consists of 50% lime and 50% carbon, but the carbide made in that way is not chemically pure. It contains 80% of calcium carbide and some calcium oxide which eventually finds its way into the cyanide.

PROF. WALLACE: Is this new alloy being prepared on a commercial scale today?

MR. FREEMAN: It has been prepared on a commercial scale in a rather different strength for the production of caustic soda. When steam is blown into the molten alloy of sodium the lead and sodium are dissolved by the steam forming caustic soda. That process was carried on for some years at Niagara Falls and the equipment is still there, but no longer in operation. It produced very high grade caustic soda.

MR. A. C. GARDE: I understand there is 5% sodium in the lead.

MR. FREEMAN: Yes.

MR. GARDE: If you have a silver-lead ore carrying from 2% to 3% lead, how would that be handled?

MR. FREEMAN: In treating lead-silver-ores it is not the usual practice to cyanide. In lead silver ores, silver is probably present as a silver sulphide in combination with lead sulphide, which it is usually impossible to cyanide. If it were possible to do so the silver would go into solution and the lead would not.

The process of recovery would not be affected in any way by the presence of the lead in the ore.

MR. GARDE: This ore would contain a small percentage of lead.

MR. FREEMAN: I still think that the lead would not affect the reaction in any way. If it were possible to dissolve the silver out of the ore without cyanide the same reaction would hold. The silver would be dissolved and the lead would be unaffected by the cyanide.

FUTURE HYDROMETALLURGY OF COPPER SULPHIDES

By William E. Greenawalt

Western General Meeting, Vancouver, November, 1919.

In 1912, when my book on the "Hydrometallurgy of Copper" was published, there was not a single electrolytic copper extraction plant in successful operation. The book concludes with the following remarks:

"The prevailing idea, especially in metallurgical literature, seems to be that the hydrometallurgical processes for the extraction of copper, are applicable only to low-grade ores. But why limit them to low-grade ores? There is no reason why wet methods should have any limitations, either as to the grade of the ore, or its mineralogical composition, provided the process is chemically adapted. This adaptation will, in its ultimate analysis, resolve itself down to the consumption of chemicals, the same as in chlorination, cyanidation, or smelting.

"Copper is one of the most readily soluble of all the metals, and one of the most readily precipitated either chemically or electrolytically. Theoretically, the solvent processes, especially the electrolytic processes, offer all that could be desired, on ores chemically adapted; close extraction, cheap deposition, copper in its metallic form, saving of the precious metals, the installation of plants at the mines, which may be operated in any unit and without admixture of other ores or fluxes. With these theoretical advantages, it is reasonable to suppose that the chemical methods will ultimately be in as general use for the extraction of copper as the cyanide and chlorination processes now are for the extraccion of gold and silver."

Since the book was published, many large copper leaching plants have come into successful operation. Two of the largest copper mines in the world are now using electrolytic processes exclusively. These plants are operating on low-grade oxidized ores. There does not appear to be a single plant in operation treating concentrate or high-grade sulphide ores. There is one large plant treating sulphide tailing which requires roasting, but in this plant the copper is chemically precipitated.

Manifestly, the purely chemical processes do not offer sufficient advantages in the treatment of copper concentrate or high-grade ores to warrant extensive investigation, for, the end product, in purely chemical processes, is not the metal which can be sold direct to the consumer, but simply a higher grade concentrate which has to be shipped, smelted, and refined,

much the same as the original concentrate or high-grade ore from which the precipitate was obtained.

Up to the present time, only two methods of precipitation have looked promising, in connection with purely chemical processes. These are:

1. Iron precipitation.
2. Hydrogen sulphide precipitation.

In iron precipitation, both the acid and the iron are irrecoverably lost, and the expense is, ordinarily, prohibitive. The end product is cement copper, which has to be shipped, and smelted with other material into matte, and then refined in the usual way. With hydrogen sulphide precipitation, the acid combined with the copper is regenerated. For every pound of copper precipitated, 1.56 lb. of acid is regenerated; this is an advantage. The end product is copper sulphide, in a voluminous form, and has to be smelted and refined to get the pure electrolytic metal. The precipitated copper sulphide would offer some problem in preparing it for proper smelting, especially in a blast furnace, but this difficulty is by no means insurmountable. It is quite conceivable how either cement copper or precipitated copper sulphide could be converted into the electrolytic metal by electrolysis, without a preliminary fusion or smelting.

The purely chemical processes do not offer a hopeful outlook in the general hydrometallurgical treatment of copper ores: First, because they are usually too expensive to operate; second, because the end product is in a very unprofitable and undesirable form, unless a smelter and a refinery are operated in connection with the hydrometallurgical plant.

It is evident that the general application of the hydrometallurgical processes to the treatment of copper ores, if it comes at all, must come along the lines of leaching, and electrolytic precipitation with the simultaneous regeneration of the combined acid. Under ordinarily favorable conditions, it is quite commercial to deposit 1 lb. of copper per kilowatt hour with the simultaneous regeneration of from 2.5 to 3.0 lb. of acid, with the pure electrolytic metal as the end product. It is quite conceivable also, how, under favorable conditions, an electrolytic copper extraction plant can be operated by the acid formed in the process, and without the purchase of acid from extraneous sources, and without the installation of an acid plant to produce acid in the ordinary way.

The application of electrolysis to leach copper solutions appears to be very simple; it is, in fact, quite difficult. The entire problem of copper electrolysis from leach solutions resolves itself into the problem of the ferric and ferrous salts in the electrolyte. Copper leach solutions always contain iron, either in the ferric or ferrous condition. The ferrous iron is not particularly harmful. The ferric iron is highly deleterious. In the deposition of the copper, the iron in the electrolyte is converted into the ferric state by the oxygen released at the anode, and this oxygen converts the iron into the ferric condition, and the ferric iron so formed re-dissolves the deposited copper at an alarming rate unless quickly checked. Under aggravated conditions the copper may be re-dissolved as rapidly as deposited, and the practical results of the electrolysis would be nil. On the other hand, if the action of the ferric salts can be subdued, the process may become highly efficient.

TREATMENT OF SULPHIDE ORES

In the electrolytic precipitation of copper from leach solutions it is immaterial whether the ore is a sulphide or an oxide. The fundamental difficulty—that of ferric salts in the electrolyte—is the same. It is also immaterial whether ore or concentrate are being treated. In the treatment of sulphide ores and concentrate, the resulting solutions are more quickly charged with iron than in the treatment of highly oxidized silicious ores, and, on this account, the effects of the iron in the electrolyte is considerably aggravated. It is doubtless, principally, for this reason that the application of electrolysis to the treatment of ores and concentrates high in iron has not been as rapid as might have been expected. It must be evident, however, that any scheme which will adequately provide

against the deleterious effects of the iron in low-iron electrolytes will also be adequate in the electrolysis of solutions high in iron.

In the application of electrolysis to sulphide copper ores, the treatment may conveniently be considered in its three most important steps. These are:

1. Roasting of the crushed ore.
2. Leaching.
3. Electrolytic deposition and solvent regeneration.

Roasting.—Sulphide ores offer certain advantages over oxides in the hydrometallurgical treatment. Roasting is far from being a total loss. Under certain conditions it may be a distinct gain.

One essential consideration in the wet treatment of copper ores is the acid consumption. In oxidized ores the acid has to be supplied from external sources, and this is usually the crucial consideration in the successful leaching of oxidized material. In sulphide ores, however, it is quite within the range of possibility to make the treatment self-sustaining in acid, by a judicious manipulation of the roasting and electrolysis.

In the treatment of sulphide ores, preliminary concentration is advisable if the concentration losses are not too high. If the ore is a massive sulphide, preliminary concentration is precluded from consideration, and the ore, if amenable to a solvent process, must be leached as a whole. Many sulphide ores contain much lime; the direct leaching treatment of such ores is quite impractical on account of the high acid consumption. If the ore is amenable to table concentration or flotation, the lime difficulty is easily eliminated. It is always better to get rid of the larger bulk of the gangue, if possible, by a cheap preliminary process such as table concentration or flotation, than to attempt a wet treatment on large volumes of low-grade material, the gangue of which usually contains the greater portion of the injurious elements for any leaching process. Hydrometallurgical processes, especially the electrolytic processes, are expensive to install, and it is better to confine the operation to a concentrated material, where possible, than to put volumes of barren gangue through an expensive treatment. Between the very high recovery usually obtainable by a roasting and leaching process, and a moderate tailing loss by preliminary concentration and flotation, in comparatively low-grade ores, it will usually be best to take the preliminary concentration loss.

As a rule, all sulphide ores have to be roasted as a preliminary to successful leaching. With care, the ore can be roasted so as to make from 25% to 50% of the copper soluble in water. The ideal roast is one in which the roasted material contains the least possible amount of soluble iron and the greatest possible amount of water—soluble copper. Roasting at a high temperature is likely to make a large percentage of the copper insoluble in either water or dilute acid. A low temperature roast will always give a better extraction than a high temperature roast, assuming, of course, that all the sulphides are decomposed. A low temperature roast is likely to result in more acid soluble iron than a high temperature roast. The difference between the decomposition temperatures of ferric sulphate and cupric sulphate is not very large. Ferric sulphate is decomposed at a temperature of 530 deg. C. (986 deg. F.), while cupric sulphate is decomposed at 653 deg. C. (1207 deg. F.). Manifestly, the best temperature condition for roasting, at least theoretically, is one which is above the decomposition temperature of ferric sulphate and below the decomposition temperature of cupric sulphate, to get the largest amount of water-soluble copper and convert the iron into the more or less insoluble oxides. In the hydrometallurgical treatment of sulphide copper ores, roasting is one of the most important and delicate steps in the operation.

In the roasting of suitable sulphide ores, the sulphur dioxide from the roasting furnace may be used in the subsequent steps, either as a solvent for the copper, as a reducing agent for the ferric salts, or as a depolarizer. In any event, under the right conditions, an equivalent of acid can be regenerated from the sulphur dioxide in the electrolytic deposition of the copper.

It is easy to see, therefore, why sulphide ores present a certain advantage over purely oxidized ores, for, in the roasting

process a large percentage of the copper can be made soluble in water as the sulphate, and the sulphur dioxide liberated in the roasting can be beneficially used in the deposition of the copper. Suppose, for example, that a concentrate containing 10% copper can be roasted so as to make 50% of the copper soluble in water as the sulphate. When this water-soluble copper sulphate is electrolyzed 1.56 lb. of acid is regenerated, per pound of copper deposited. By the use of the sulphur dioxide, from 2.5 to 3.0 lb. of acid is regenerated under suitable conditions. It is evident, therefore, that, by the roasting of such an ore and the electrolytic deposition of the copper, the treatment can be made self-sustaining in acid without the use of an ordinary acid plant.

Roasting installations are expensive. This is one, among other good reasons, why the material to be treated should be as high-grade as possible. The capacity of a given sized furnace will be much the same for a high-grade as for a low-grade copper ore, assuming that the sulphur content is much the same for both. It makes a big difference, in the cost of installation for a given output of copper, whether the roasting operation is conducted on a 10% or on a 2% copper material. The fuel consumption for both will also be much the same.

It may be assumed, in the treatment of a heavy sulphide ore, that the roasting installation will cost approximately $1,000 per ton of material roasted per day of 24 hours. A roasting plant of (say) 25 tons daily capacity will cost, installed, about $25,000. This would be a moderate investment in treating a fairly high-grade ore; it would be a rather large investment in the treatment of (say) a 2% heavy sulphide copper ore. On the other hand, as already indicated, the furnace can, to a very large extent, take the place of an acid plant, in the leaching of purely oxidized ores The fuel consumption on most ores may be assumed as approximately 10% of the weight of the ore.

Roasted ore, as a rule, leaches better than the same material unroasted. The leach solutions are also likely to contain less slime, but this is not always the case. With modern high-class roasting installations, roasting offers no particular difficulty, especially at the low temperature necessary to permit of a good extraction of the copper.

Careful roasting is quite certain to reduce the acid consumption, especially if the ore contains a considerable quantity of lime Much of the lime is, by roasting, converted into the comparatively harmless calcium sulphate, which is not acted upon by dilute sulphuric acid. If the ore is a massive sulphide and contains too much lime for a normal acid consumption, direct leaching of the ore should not be attempted.

In the treatment of sulphide ores, the percentage of extraction of the copper is mostly dependent upon the roast. On a suitable material and with care in roasting, it should be possible to get extractions quite closely approaching those obtainable by smelting. In experimenting with Mt. Lyell flotation concentrate, Middleton got extractions of from 97% to 98% of the copper from a material assaying from 10% to 20% copper, and 29% sulphur.[1] With a chalcopyrite ore containing 1.07% copper, 49.25% iron, and 32.60% sulphur Wedge succeeded in making 27.8% of the copper soluble in water and 63.4% soluble in dilute acid or a total extraction of 91.2%. With another chalcopyrite ore assaying 4.38% copper, 9.61% iron, 16.37% sulphur, and 61.21% insoluble, 73.7% of the copper was soluble in water and 15.7% in weak acid, or a total extraction of 89.4%. With a bornite ore[2] containing some silicate and carbonate of copper, assaying 3.40% copper, 11.05% iron, 13.02% sulphur and 63.90% insoluble, 80.80% of the copper was soluble in water and 11.8% soluble in acid, or a total extraction of 92.6%. These results confirm my own experimental work, and the general conclusion that with careful roasting a fairly high percentage of the copper can be made soluble in water, and that usually a very high percentage of extraction may be obtained with dilute acid. Addicks got a total extraction of 84.70% only of the copper in the experimental roasting and leaching of Tyrone concentrate assaying 15.48% copper, 28.0% iron, and 30.0% sulphur.

[1]Min. and Scientific Press, June 7, 1919.
[2]Congress of Applied Chemistry, Vol. 3.

The tests were of short duration, and it is probable that much better results could have been obtained with more experience in the roasting of this concentrate and a close control of the temperature.

Leaching.—In the leaching of roasted ore and concentrate, agitation will usually give better results than percolation. Under certain conditions it will be found desirable to separate the fines from the sands, and agitate the fines, and percolate the sands. Roasted ore, especially if fairly high-grade in copper, is likely to form an impenetrable mass if percolation is attempted. In almost all cases it is desirable to treat the fairly hot ore with the leach solution. The hot solution causes a better extraction, lowers the voltage in the electro deposition, and causes a quicker reduction of the ferric salts. On the other hand, it also causes a greater activity of the ferric salts in re-dissolving the deposited copper, and this tends materially to reduce the ampere efficiency of the deposition. Leaching of the roasted material, whether of ore or concentrates, offers no particular difficulty, and the methods and apparatus are no different from other leaching operations, except, of course, that all tanks and conduits have to be made acid proof. This implies an increased cost of installation, and, to some extent, a slightly increased cost of operation, over neutral or alkaline solutions.

Precipitation of Copper.—In the extraction of copper from suitable ores by a wet process, the vital step usually lies in the method of precipitation. Chemical precipitation, as already indicated, does not present a hopeful outlook, and it appears to be quite impossible if a high-grade ore or concentrate is to be treated; for, in addition to the expense of chemicals for precipitation, the resulting precipitate represents simply another stage of concentration, and not a product which will command the highest market price and which is saleable direct to the consumer.

Electrolytic precipitation is the key to the situation. Electrolytic precipitation presents some difficulties but they are not insurmountable. The problem of the electrolysis of leach copper solutions can be focussed on the action of the iron in the electrolyte. The difficulty is much the same to-day as it was in 1883 when Body, the first patentee for an electrolytic process of extracting copper from its ores, applied for a patent. Iron, as every chemist and metallurgist knows, is a variable valent element. Ferrous sulphate in the electrolyte is not particularly harmful. Ferric sulphate is highly deleterious, for the reason that it re-dissolves the copper, more or less in proportion to its presence in the electrolyte. It is, however, beneficial in the leach solutions, and, to a large extent, is the equivalent of free acid, in dissolving copper from its ores. In some respects, it is superior as a solvent to the acid for the reason that it will dissolve copper from combinations which are unattacked by sulphuric acid. Copper, in the form of chalcocite, is quite insoluble in sulphuric acid: it is readily attacked by ferric sulphate.

Manifestly, in the treatment of ores high in iron, as for example, copper concentrate or massive sulphide ores, the iron in the solution accumulates quite rapidly, and it is mostly because of this rapid accumulation of the iron in the leach solutions that the electrolytic treatment of copper ores high in iron has not made the progress that was hoped for. The large and successful electrolytic plants at Chuquicamata and at Ajo do not represent the average conditions of copper mining. At Chuquicamata, where 20,000 tons of silicious oxidized ore are treated per day by sulphuric acid solution and electrolytic precipitation, the ferric iron appears to be the greatest difficulty. Large volumes of solution have to be wasted daily to keep the ferric iron in the electrolyte within workable limits, but this, at Chuquicamata, is not a serious matter, since the ore contains much of the copper as the sulphate, from which acid is regenerated in the electro deposition of the copper, so that the process is practiclaly self-sustaining in acid. At Ajo, about 25% of the copper is precipitated with iron to maintain the electrolyte sufficiently low in ferric salts to keep its deleterious action within bounds.

Both at Chuquicamata and at Ajo, the deleterious effects of the iron in the electrolyte are kept within workable limits by a process of close elimination above a certain low percentage of iron. Evidently, in the wet treatment of a massive sulphide ore or copper concentrate, much more iron will go into solution than in the treatment of a highly silicious ore, and close elimination of the iron in the leach solutions would mean the discarding of enormous volumes of solution and the chemical precipitation of a very large percentage of the copper, unless more effective methods are used to overcome the difficulty.

Ferrous sulphate, as stated, is not particularly harmful in the electrolyte, It may be highly beneficial. Ferrous sulphate is an effective depolarizer, and this depolarization may considerably reduce the E.M.F. required in the deposition of the copper. It is quite possible, with effective ferrous sulphate and sulphur dioxide depolarization, to get as high as 2.5 lb. of copper per kilowatt hour, under what would appear to be practical working conditions. This has not yet been accomplished in practice, and must, for the present, be considered only as a possibility. The demonstrated results under commercial conditions, of 1 lb. of copper per K.W. hour with the simultaneous regeneration of from 2 to 3 lb. of acid, would appear to be safe basis for prospective installations and operations.

Manifestly, if a high-iron electrolyte, using ferrous sulphate as a depolarizer, can be made to yield such excellent results, the idea would naturally be to make the iron work with the process, instead of against it, and the problem would then resolve itself down to the reduction of the ferric sulphate produced by the electrolysis, to the ferrous sulphate, and this is practically the crux of the entire situation. Sulphur dioxide, as ordinarily applied; will not do it effectively, as is evidenced at the works of the New Cornelia, at Ajo, where only one-sixth of the copper is deposited from the solutions in passing though the cells, when it is again sent on its cycle through the leaching vats, principally to neutralize the ferric salts.

For the most effective work, in practice, the ferric sulphate in the electrolyte should not exceed a maximum of 0.25%. It may, with advantage, be considerably less. The amount of ferrous salts in the electrolyte is not of much consequence, so long as it does not become a nuisance through crystalization and impede the flow of solution through the entire leaching system. A solution, heavily charged with ferrous sulphate, seems to retard the solution of the copper from the ore, but this is not a serious defect. At the same time, it will also act to retard the solution of iron from the ore, which would be an advantage. The electrolysis of high-iron electrolytes in the deposition of copper from leach solutions seems reasonably within realization, and, if it is fully realized, it is likely to change the entire aspect of the present hydrometallurgy of copper.

Next to lime in the ore and iron in the electrolyte, aluminium and zinc are the most important factors ordinarily entering into the situation. Aluminium acts as an acid consumer, but, within reasonable limits, also acts to restrain the solvent action of the ferric salts in re-dissolving the deposited copper, as was discovered by Addicks.[1] Zinc acts much in the same way.[2] The amounts of injurious elements that an ore may contain before a wet process is impractical, can only be determined by direct experiment. The acid consumption, per pound of copper extracted, will usually be the determining factor. In the last analysis, the application of an electrolytic process to the treatment of copper ore will depend on the cost of the acid, per pound of copper extracted, and the cost of the power for the copper deposition and acid regeneration.

Treatment of Sulphide Ore, High in Lime

In the foregoing discussion, it was assumed that the ore was commercially amenable to an acid solvent process. It frequently happens that the ore contains so much lime as to make the acid consumption excessive, and, hence, make the process impractical. If the ore is low in mineral content, as is usually the case, most of the injurious gangue can be eliminated by table concentration and flotation, and then treating the concentrate by a wet process, as already indicated. Since it is presumed that the concentrate are treated at the mine by an electrolytic process to obtain the pure electrolytic metal, the grade of the concentrate need not

[1]Met. and Chem., Eng., Oct. 15, 1915.
[2]Middleton; Mining and Scientific Press, Aug. 2, 1919.

be very high, and hence the loss in the preliminary concentration need not be very large, for much of the loss in mechanical concentration is due to the necessity of obtaining a fairly high-grade concentrate to stand shipping and smelter charges.

It sometimes happens that the ore contains too much lime for direct leaching, and that the loss by mechanical concentration is so large as to preclude this as a profitable step. Such ores are usually amenable to profitable smelting. Under favorable smelting conditions, it is quite feasible to smelt the ore, either into a high or a low-grade matte, as a preliminary treatment to eliminate the injurious elements, and then treat the matte by leaching and electrolysis, much the same as in the treatment of suitable sulphide ores or concentrate. The outlook, along this line, would appear to be very hopeful and the practice should be highly profitable when employed at smelting plants which are not equipped to turn out the pure electrolytic metal. In smelting, practically all of the highly injurious elements are eliminated, except the iron, and the iron may not be an altogether injurious element. It is not intended to electrolyze the matte. The direct electrolysis of copper matte has been frequently proposed, but, as in the Marchese process, such attempts have usually ended in failure, largely on account of the complications due to the casting of the matte into suitable anodes, to the liberated sulphur, as a result of the electrolysis, and to the insecurity of the connections between the matte and the electrical conductors. The present suggestion is not to electrolyze the matte, but to first crush it to a suitable fineness, then roast it with a view of making as much as possible of the copper soluble in water and dilute acid, leaching the roasted matte, and electrolyzing the resulting copper solutions to obtain the pure electrolytic metal and regenerate the acid, as already described for sulphide ores and concentrates. Some time ago, this line of procedure was being extensively investigated on a fairly large scale by the writer, in Mexico, until the revolutionists blew up the company's power plant and temporarily put an end to our activities. The investigation will be resumed as soon as conditions will permit. The outlook was most hopeful.

A large proportion of the copper in matte can be made soluble in water with proper roasting, and no unusual difficulties presented themselves as far as the investigations went.

TREATMENT OF COPPER ORES CONTAINING PRECIOUS METALS

Practically all copper ores contain some, if only in considerable amounts of precious metals. Occasionally, the precious metals occur in amounts which make their recovery worth considering. This is especially true of concentrates and copper matte. A chloride solution is capable of extracting the copper, gold, and silver, in one operation, but it is not ordinarily advisable to work with chloride solutions in the extraction of the copper. If electrolysis is involved in the treatment, sulphate solutions are to be preferred. This is also usually the best for the precious metals, because, by using sulphate solutions, the precious metals are left behind, in the residue, and can then be extracted separately, preferably with a chloride solution. An acid treatment for the extraction of the copper, is an excellent preliminary for the extraction of gold and silver by a chloride process, since chlorine is in the nature of an acid element. The electrolytic generation of chlorine from common salt is now so commonly used in the production of chlorine and caustic soda, as to make its adoption free from any unusual uncertainty, and the chlorination of gold and silver ores is a time-honored process. Electrolytic chlorine was used on an enormous scale in the treatment of Cripple Creek ores before chlorination was supplanted in the treatment of those ores by the cyanide process. Its use, therefore, in the treatment of copper-leach tailing for the extraction of the gold and silver would offer no great difficulty.

In the recovery of gold and silver from copper-leach tailing, the remaining copper will also be extracted almost to completion. The chlorine will also act on small amounts of iron. Copper chloride and iron chloride are thus inevitably brought into the precious metal solution, and both cupric chloride and ferric chloride are effective chloridizers of the silver contained in the ore tailing. Brine may be used as the leach solution, if the copper

tailing contains silver worth recovering. If the tailing carries only gold, an ordinary chlorine solution will suffice.

In a recent experimental investigation of an Arizona copper-gold ore, containing 19.3% copper, roasting and acid leaching showed an extraction of 96.3% of the copper. The copper-leach residues assayed 0.9% copper, and 1.11 oz. in gold. The tailing, after chlorine treatment, assayed 0.10% copper, and 0.02 oz. in gold, showing a total laboratory extraction of 99.5% of the copper and 98.2% of the gold. These extractions could certainly not be realized in practice, but the particular point of interest in this connection, is, that 0.80% of the copper was extracted with chlorine. Manifestly, this represents a high chlorine consumption, for, the chlorine will act more or less on the copper to the exclusion of the gold. It also indicates the close relation between the gold and the copper, and that when the copper is closely extracted a close extraction of the gold is also quite certain. This brings up the very pertinent question, as to whether the cost of the chlorine which acts on the copper before the gold is attacked, makes the entire process unprofitable. It would seem that this is not the case, for, assuming that the copper is extracted as the cupric chloride, it will take approximately a pound of chlorine to extract a pound of copper. The chlorine, generated from common salt by electrolysis, should not cost more than 5c. a pound, but the copper extracted at this cost is worth from 15c. to 25c. a pound, so that the copper so extracted would show a reasonable profit of itself. The extra consumption of chlorine to extract the gold is practically nil.

The suggestion to cyanide copper leach tailing in order to recover the precious metals has frequently been made. Some claim to have got most encouraging results. My own results have been of a highly negative character. The copper and acid can hardly ever be so thoroughly eliminated from copper-leach tailing as to prevent reaction with cyanide, and, as four pounds of cyanide will react with one pound of copper, it can readily be seen that it will not take a great deal of copper to show a heavy loss. If cyaniding is contemplated after acid copper leaching, it should be approached with the utmost suspicion.

If chlorine is generated by electrolysis, for the extraction of the precious metals, and the plant is located at or near a manufacturing centre, the accompanying caustic may have a market value sufficient to reduce greatly the cost of producing the chlorine. In the electrolysis of salt, caustic appears to be in very much greater demand in the industries than chlorine.

ECONOMIC CONSIDERATIONS

It would not be advisable to attempt any elaborate figures either of the installation or of the operation of an electrolytic process, for, evidently, such figures are largely dependent on the location of the mine, the nature of the ore, the cost of fuel for roasting, and the cost of power for electro deposition of the copper and the regeneration of the solvent. It may be assumed, however, that the installations are expensive, but it must also be remembered that the end product is the electrolytic metal, so that the electrolytic extraction process takes the place of smelting the ore to matte, blowing the matte into blister copper, and the usual electrolytic refining to obtain the marketable metal.

In the solvent treatment of concentrate, roasting can hardly be regarded as much of an extra expense over that of a smelting process, either as to installation or operation. If sulphide concentrate is smelted in a reverberatory, preliminary roasting is necessary; if smelted in a blast furnace, either sintering or nodulizing is a necessary preliminary. The roasting, however, for a solvent process, must be more carefully done for leaching than is necessary in roasting, sintering, or nodulizing, as a preliminary for smelting, still, the difference in the requirements is not very marked.

As a rough approximation, the estimated cost of a modern efficient roasting plant, for the roasting of sulphide ores or concentrate as a preliminary for leaching, will be about a thousand dollars per ton of ore roasted per day of 24 hours. The cost of roasting under ordinarily favorable conditions should not exceed one dollar per ton. Under favorable conditions, these costs may

be considerably less; under unfavorable conditions, they may be considerably more.

In connection with the roasting of a sulphide ore, as a preliminary to leaching, it should be remembered that the sulphur gas is a valuable by-product, and may be used as a source of sulphuric acid for the subsequent leaching and as a reducing agent in the electro deposition of the copper.

The power required for the electrolytic deposition of the copper and regeneration of the solvent, in electrolytic plants, is one of the most formidable items of expense, both of installation and operation. The assumed and demonstrated cost of 1 kw-hour, per lb. of copper deposited with a simultaneous regeneration of from 2.5 to 3.0 lb. of acid, gives a fair basis of power estimation. In addition to the 1 kw-hour, per pound of copper deposited, about 0.5 kw-hour, per pound of copper, should be provided for other uses, such as crushing, pumping, agitating, conveying and concentration, etc. Ordinarily, the necessary power can be cheaply produced in copper mining districts within transmitable distances from the reduction plant, from hydro-electric installations. Electrolytic processes are peculiarly favorable for water-power development, for the reason that in the deposition of the copper the load is constant and in use 24 hours per day. In steam or oil-power plants, the fuel consumed is more or less in proportion to the power used. Under ordinarily favorable conditions, especially under the constant and continuous load of an electrolytic process, power can be produced for approximately 1c per kw-hour. Under favorable water conditions, it should cost less. Ample provision would, of course, also have to be made for the power requirement of the mine, and this would tend to reduce the installation and operation costs directly chargeable to the reduction plant.

The power installation, will, therefore, be much the largest item of expense, and in the power installation it is intended to include transmission, transformers, and motors, including the motor-generator sets required for the electrolytic deposition of the copper.

The leaching plant, whether agitation or percolation is used, or a combination of both, will cost considerably more than a similar plant using non-corrosive solutions. The machinery and other apparatus will be much the same, except that it has to be made acid proof.

In the electrolytic department the principal item of expense will be the metal in the conductors, the anodes, and the cathodes. Lead will ordinarily be used as the anode. Antimonial lead resists oxidation better than chemical lead.

If the acid consumption is rather large, it might, under certain conditions, be advisable to install a small acid plant in connection with the reduction plant. Under the assumed conditions of roasting a sulphide ore to extract the copper, a portion of the roaster gas can be used in the acid plant.

In remote or inaccessible districts, especially where the winters are severe, electrolytic extraction plants offer a decided advantage over shipping the ore to smelters, or of smelting at the mine, especially if water-power is available during the winter months. The production of copper can go on economically and uninterruptedly, and as the metal represents only from 5% to 10% of the weight of the average ore shipped, a saving of from 90% to 95% can be made in the expense of hauling and freight. The freight and hauling are usually serious items of expense.

Under the assumed conditions of a suitable sulphide ore or concentrate, and possibly water-power within transmittible distance, practically all the elements for continuous operation occur in the vicinity of the reduction plant. Acid is one of the essentials of a leaching process, and the acid may be obtained from the ore by roasting and converting the resulting sulphur dioxide into sulphuric acid, either through electrolysis, or a small acid plant, or both. Either iron, or iron sulphide and acid, may be used for the chemical precipitation of the copper from extremely lean and extremely foul solutions; these precipitants also may be obtained from the ore. Wood frequently occurs in the vicinity of copper mines, and makes an excellent roasting fuel. Oil is preferred, if cheaply obtainable.

Smelting on a scale of 50 to 100 tons a day presents great difficulties. An electrolytic process can be operated in any unit. The deposition of the copper and regeneration of the acid is independent of any mass action, and the power consumed in the electro deposition per pound of copper is the same in the operation of a plant of a capacity of one ton of ore a day as in one of a thousand tons. Of course, large scale operations are more economical than small scale operations, but this is independent of any inherent technical difficulties. A plant to smelt 25 tons of 10% copper ore per day to produce a high-grade matte, would be technically impractical and quite surely unprofitable. The same ore treated by an electrolytic process to produce the pure metal at the mine, is not technically practical, but under ordinary conditions should be highly profitable. Even a 10-ton a day plant should be made to yield profitable results, under favorable conditions.

In the consideration of the adoption of an electrolytic process, all the essential elements can be determined on small scale operations. Laboratory tests to determine the roasting, extracting, and acid consumption, will usually decide the amenability of the ore to the process. If the laboratory tests are encouraging, a small pilot plant, if only to treat a ton, or even a few hundred pounds, of ore a day should be installed at the mine or some convenient place, and operated continuously for several months to determine all the important factors entering into the consideration of a final large scale installation. It is essential that the small pilot plant be operated continuously, for the injurious effects of the ferric salts are cumulative and the pilot plant should be operated long enough to determine a proper balance of the impurities in the solution, the acid consumption, under the conditions, and the efficiency of the electro deposition and regeneration. If the results in the pilot plant are satisfactory, the large scale final installation can proceed with confidence. Even the final large scale installation can be made in sections, with proper provision. The capacity of the roasting furnace would naturally determine the size of the first unit section, and the leaching and electrolytic units could be made to conform with the roasting unit, having always in mind the largest possible unit that can be used. The electrolytic unit would have a large capacity cell, which would also be the most convenient unit for the final completed installation.

FUTURE HYDROMETALLURGY OF COPPER SULPHIDES

The hope of the independent copper miner lies in the hydrometallurgical processes, and, above all, in the electrolytic method of copper deposition and acid regeneration. This applies with particular emphasis to mining districts where fuel is abundant and where power can be cheaply developed, and especially to districts remote from lines of transportation It seems almost unbelievable that a field so promising as electrolytic extraction processes, should have been so long neglected.

Prophesying is attended with great uncertainties, but there seems to be no escape from the conviction that in a few years electrolytic extraction methods will be so well established as successfully to compete with smelting, and, while each may have its own particular field, electrolytic extraction will be much the more generally applicable of the two. Successful smelting is more or less dependent upon suitable mixtures and fluxes, and these are not usually found in the same mining district, whereas, in the hydrometallurgical treatment, fluxes are not required, and mixing of widely different ores is undesirable.

It is not likely that any hydrometallurgical process will displace table concentration and flotation, as a preliminary treatment of sulphide ores, assuming, of course, that the ore is amenable to successful concentration and flotation. Where fuel is cheap and the ore does not yield a satisfactory recovery by table concentration and flotation, there is a wide field for sulphate roasting and leaching of low-grade copper ores, even with chemical precipitation of the copper from the leach solutions.

The wider field for the hydrometallurgical processes, is not in treating low-grade material, but in direct competition with smelting of the higher grade ore and concentrate, and the pro-

duction at the mines of the electrolytic metal which can be sold direct to the consumer at the prevailing market price of electrolytic copper. Since only the electrolytic processes produce an end product which has a direct open market value, and, as other hydrometallurgical processes at best simply represent another stage in concentration, it is difficult to see how the wet processes involving chemical precipitation can successfully compete with any process which produces the electrolytic metal as the end product. The conclusion is inevitable, that the future of the hydrometallurgy of copper sulphides will be along the lines of electrolytic processes, and that their best application will be to the higher grade ores and concentrate, with table concentration and flotation, in low-grade ores, as a preliminary step to the more expensive and more efficient roasting, and electrolytic extraction of the copper from the enriched and concentrated material.

The small and independent miner, who can rely on a steady supply of ore, may, no doubt, in a few years be shipping electrolytic copper instead of ore and concentrate. This conclusion seems to be inevitable.

RECAPITULATION:

1. Hydrometallurgy offers an effective solution to the high shipping and refining costs of copper ores and concentrate.

2. The hope of the hydrometallurgical treatment of copper sulphides is along the lines of roasting, leaching, and electrolysis of the leach solutions to obtain the electrolytic metal and regenerate the solvent.

3. Electrolytic extraction plants may be installed in any unit and successfully operated without the admixture of other ores or fluxes.

4. Electrolytic processes are applicable to sulphides as well as to oxides; to the concentrate as well as to the ore direct

5. Electrolytic extraction plants are expensive to install, but they are highly efficient.

6. An electrolytic process, on suitable ores, takes the place of smelting the ore to copper matte; of converting the matte into blister copper; and of the ordinary refining process to convert the blister copper into the electrolytic metal.

7. The process is applicable to copper matte, if smelting is advisable as a preliminary treatment for ores which are not otherwise amenable to direct treatment with an acid solvent.

8. Cheap power is one of the essentials of an electrolytic process, and cheap water-power can frequently be developed in the vicinity of the copper mines, especially in the Rocky Mountain regions.

9. On the basis of 1c. per kw-hour for power, the power cost for the deposition of 1 lb. of copper and the regeneration of from 2.5 to 3.0 lb. of acid, will be 1c. The total power cost would be approximately 1.5 c.

10. It is quite within the range of possiblity that an electrolytic process, treating suitable sulphide ores, can be made self-sustaining in acid. If it cannot, the small deficiency can readily be made up with a small acid plant in connection with the roasting furnace sulphur gas.

11. The vital issue, in the electrolysis of copper leach solutions, will be found in the ferric and ferrous salts. The success of an electrolytic process will depend mostly on the efficiency of the reduction of the ferric salts in the electrolyte, and largely on the extent to which ferrous salts can be used as a depolarizer.

12. Metallurgically, the efficiency of an electrolytic extraction plant is independent of its magnitude. The character of the roast, the extraction of the copper, and the electrolytic deposition of the copper and regeneration of the solvent, are, metallurgically, about the same for a small plant as for a large one.

13. Electrolytic extraction processes will find their widest application in the treatment of the higher grade ore and concentrate.

14. The end product of an electrolytic process is the electrolytic metal saleable in the open market direct to the consumer.

DISCUSSION

PROF. H. N. THOMSON: Mr. Greenawalt refers to the difference between the cost of roasting for leaching and the cost of roasting for smelting as being negligible. Taking his figure of $1 per ton for the former this difference will run well over 50 cents. This is mainly because roasting for leaching is an extremely delicate operation compared with roasting for smelting. The latter, in some cases, is not much more than a drying process and is then accomplished with consumption of about five gallons of crude oil to the ton of concentrate. In experimental roasting to convert copper sulphides into compounds soluble in sulphuric acid, one usually tries, at the same time, to convert iron sulphides into compounds insoluble in this substance. When this is accomplished, it is sometimes found that an appreciable amount of the copper has become insoluble. I found on roasting the Anaconda pond slime at a high temperature that all the copper could be rendered insoluble in dilute sulphuric acid.

With reference to the effect of ferric sulphate, $Fe_2\{SO_4\}_3$, in dissolving the precipitated copper. This fact was brought forcibly to the attention of the Arizona metallurgists when the copper roof on a roaster dust-chamber failed from the solvent action of ferric sulphate that was formed in the roasting operation. This was carried in the gases, and deposited on the underside of this copper roof, on which it at once acted according to the equation: $Cu + Fe_2\{SO_4\}_3 = CuSO_4 + 2FeSO_4$. Anyone who has had occasion to make an electrolytic copper determination on a sample high in iron and low in copper will realize what a nuisance this ferric sulphate is.

Mr. Greenawalt's "inevitable" conclusion regarding leaching sulphides is much to be desired, but in the light of our present knowledge, I consider the word "inevitable" to be unwisely chosen and not at all appropriate.

MR. MILNOR ROBERTS: I may be permitted to mention some of the processes now used in the treatment of copper ores and then point out how important it would be in this part of the world (Western America) to have available such a process as Mr. Greenawalt describes for the treatment of either ores or concentrate.

The simplest of all methods of treating copper ores is that applied to native copper rock. The mass copper of the Lake Superior region, an almost pure form of the metal, is merely melted in reverberatory furnaces after the adhering rock has been removed. The low-grade deposits of native copper in the same region are treated by milling processes and the resulting copper, freed from its enclosing gangue, is melted and cast in marketable forms. A very common and also very ancient method for treating high and medium grade copper ores is direct smelting. A good example of this practice has been offered at Bisbee. Arizona, where throughout the life of the camp the ores have been smelted directly, although recently concentration has been tried.

Another common mode of treatment consists of concentration followed by smelting. The process of concentration formerly was performed on jigs and various tables, but within the past decade the oil flotation process has been added either to supplement the previous apparatus or to replace it wholly or in part. Last spring, when at the Vancouver meeting of the International Mining Convention an excursion was made to the Britannia mine on Howe Sound, we found the mill using jigs, tables, and flotation machines. Now, Mr. C. P. Browning, general superintendent of the property, informs me that the tables are no longer used, but the saving is made by jigging and flotation. At Copper Mountain, the Canada Copper Corporation has just completed a large mill which will use the flotation process alone. At Ajo, Arizona, the New Cornelia practises direct dissolving and precipitation of the copper in its oxidized surface ores. At Kennecott, the carbonate ores are leached by the ammonia process, while the sulphides are recovered by jigging and tabling.

This enumeration brings before us the wide range of processes adopted under the various conditions of practice. If, in addition, the process described by Mr. Greenawalt proves satisfactory for leaching copper ores directly or for leaching concentrate, relief will be afforded to many mines. In British Columbia especially the conditions are highly favorable for such a process and it would be well for owners and engineers to consider the possibilities of this process before determining upon any other.

Many mines in British Columbia have cheap power. Waterpower can be developed at many of the properties, wood fuel is abundant at a great majority of them; fuel oil is fairly cheap on the Coast, and coal can be had at reasonable prices at tidewater and along the railways. Electric power is available now near the cities and in future it probably can be obtained from trunk lines extending up and down the Coast and along the main arteries of industry. We have clear water in abundance. Although we are inclined to joke about its prevalence overhead and underfoot in the Coast regions, anyone who has operated in the desert appreciates the blessing of a plentiful supply of clear water for milling purposes. Another special asset of the Pacific Coast is wood suitable for the construction of tanks. To sum up, it appears that conditions in British Columbia are especially favorable to the operation of the process described in the paper and the success of the process would mean great relief for mines situated in remote regions.

MR. N. THOMPSON: Some twelve years ago, I saw a sheet of copper in an office in London, and in reply to my enquiry was informed that it was made by the electrolytic process in Belgium from low-grade sulphide ores. I had some samples of the Britannia ore with me and showed them to my informant, who remarked that this ore was precisely of the character needed for this process. Subsequently, I sent samples of the Britannia ores to England, and also some others, including ores from the Red Cliff. I got a report from that company which pronounced the ores quite amenable to treatment by this process.

The process that has been described in the paper is the same process used in Belgium at that time. Later, a plant was built in Cumberland, England, where we treated copper ores as low as seven-eighths of one per cent. They produced pure copper from $1\frac{1}{2}\%$ ore at a cost of 7c. a pound.

PROF. DANIELS: A question of prime interest in connection with processes that may be developed in our northwest country is that of a local supply of sulphuric acid. As you probably know, on the American side we have no deposits of pyrite sufficiently large or cheaply operated to enable anyone to build a sulphuric acid plant on the Pacific Coast. This was particularly felt during the war when the demand for sulphuric acid was very great. I do not know where your local plant at Vancouver gets its present supply of raw material for the manufacture of sulphuric acid.

A MEMBER: Japan principally.

PROF. DANIELS: I am aware that The DuPont Powder Company, operating near Tacoma, imported sulphur from Japan for manufacturing purposes. I am also informed that on the Eckstal river in British Columbia, there is promise of the development of large and valuable pyrite deposits. I may also mention that the Tacoma smelter is saving the sulphur dioxide gas from its roasting plant. But it seems to me that a cardinal point in the discussion is this question of the source of sulphur for sulphuric acid manufacture.

MR. E. E. CAMPBELL: So far as natural resources are concerned, there are practically unlimited quantities of sulphide ores in British Columbia, from which sulphuric acid could be manufactured. The difficulty, however, is the limited market. At Anyox, the Granby Company has at least five million tons of such ores with a sulphur content of from 40% to 50%. At the Ecstall mine on which the Granby Company holds an option, there are several million tons of sulphide ore with a sulphur content of almost 50%, and I know of two or three other deposits of nearly equal magnitude.

Speaking generally on the matter of leaching copper sulphides, I personally would not feel disposed to venture such an operation. Not long ago, I visited a mine in Arizona, where they are conducting the largest leaching operation in the world. This, however, is on oxide ore. The name of the property is the New Cornelia, and is controlled by the Calumet & Arizona Mining Company. This property, at present, is treating about 5,000 tons of ore per day. The leaching is done with a weak solution of sulphuric acid, and the copper is precipitated electrically. This oxide ore is merely a capping on a much larger body of sulphide ore. The Company would, later on, be in a fine position to leach their sulphides, having a modern mill at hand, if they could do so commercially. The heads of this concern are amongst the leaders of modern mining and metallurgical practice in Arizona and have already carried on experiments with roasting and ultimate leaching of their sulphides. In discussing the matter with their manager, it is evident that their experiments were not sufficiently satisfactory to consider this method. Another gentleman spoke of the cost of acid. At this Arizona plant, as I remember it, the acid was made at the Calumet & Arizona smelter at Douglas. The amount produced was 150 tons per day at a cost of $10 per ton. I was greatly taken with the first speaker's statement regarding the troubles of handling sulphide ore. We, at the Granby Company's plant, have had considerable experience along this line, and I for one would deprecate any attempt to roast our ore for treatment in a leaching plant in the hope of producing copper at the market prices.

MR. J. A. DAWSON: Some of the remarks made this morning almost bring tears to my eyes with regard to the chemical aspect of the question. As to getting sulphuric acid from sulphide ores, no doubt the Consolidated Mining & Smelting Company's representative could tell us of their success in manufacturing that acid from the sulphur dioxide obtained in smelting sulphide ore. The acid is used in their process by which they separate five different metals electrolytically. They also produce copper sulphate and hydrofluoric acid. I am also informed that the sulphuric acid works at Barnet near Vancouver were using sulphide ores instead of sulphur during the war. With capable chemists carrying out research work, there is no doubt that the problem that Mr. Greenawalt states could eventually be solved, It reminds me of the statement of the chemist Haber before the war regarding the production of ammonia from nitrogen and hydrogen. After repeated failures, Haber said that he still felt that the thing could be done. And it was finally accomplished. The war would probably not have lasted long if the Germans had not been able to obtain nitric acid from the nitrogen of the air by Haber's process. Also the manufacture of 99.9% pure zinc by the electrolytic method as worked out at Trail would encourage us in this problem of the separation of copper from solutions containing ferric iron. One of the chief difficulties in the electrolytic refining of zinc was the presence of very small proportion of arsenic. This was removed by agitating the solution with zinc dust upon which the arsenic separated in metallic condition. Dr. McIntosh, Professor of Chemistry at the University here, collaborated with the chemists at Trail in solving this and other related problems.

The problem of removing ferric iron from a copper solution by an economic process would seem more difficult. There are a number of methods used in the laboratory of separating iron from copper. Ammonia precipitates iron and copper as hydroxides, and an excess of ammonia dissolves the copper hydroxide but not the iron hydroxide. Possibly, it might be found unnecessary to remove the iron; instead it could be kept in the ferrous condition by introducing a reducing agent such as sulphur dioxide.

MR. S. S. FOWLER: One matter in connection with the sulphur supply. You are aware that Trail has put a tax on sulphur, a maximum of $3 per ton at the rate of 30 c. a unit.

MR. GREENAWALT (*Communication to the Secretary*):—The costs of roasting for smelting as stated by Prof. Thomson are evidently those that would obtain at large smelting plants fairly conveniently situated in respect of labour, fuel, and transportation. I had more particular reference to moderate

sized roasting installations located at the mines. Under those conditions, the comparison may need some modification. But even if the entire cost of $1 a ton for roasting is charged to leaching and electrolysis, it would not seriously affect the issue in the case of the independent miner and shipper, or of the independent smelter treating ore that is amenable to leaching.

It is true, as Prof. Thomson indicates, that by roasting at a high temperature, practically all the copper can be made insoluble in a leaching solvent. This is equally true in the roasting and leaching of ores of other metals. Some years ago, when I was employed at the Portland mill, at Colorado Springs, 400 tons of Cripple Creek ore was roasted daily. A high temperature roast made even a reasonable extraction impossible, and an excessive temperature in the roasting made practically all the gold insoluble in leaching; nevertheless we had very little difficulty in maintaining the roasting temperature within the desired limits to get an exceedingly high extraction with the leach solution.

Referring to the point raised by Prof. Daniels, concerning sulphuric acid supplies, it is reasonably certain that in leaching and electrolysis of sulphide ores, the process can be made self-sustaining in respect of acid by using the sulphur gas from the roasting furnace as a reducing agent for the ferric salts formed by the electrolysis. With a little care, much, if not most, of the copper can be rendered soluble in water, by roasting to the sulphate. This copper sulphate, in connection with iron salts and sulphur dioxide regenerates much acid, and this acid can then be applied to the ore to dissolve the copper, which is insoluble in water but soluble in dilute acid. It is safe to say that from 2.5 to 3.0 lb. of acid can in this way be regenerated for every pound of copper deposited. This will usually fulfil all requirements. It should be remembered, in this connection, that in the roasting of sulphide ore, and especially a sulphide concentrate, the consumption of acid is not greatly in excess of the theoretical equivalent. In roasting, most of the compounds in the raw ore which would consume acid, will be converted into the oxide or sulphate, which would be quite unaffected by dilute acid.

I would not advocate hydrometallurgical methods under all conditions, especially where smelting plants are installed and the ore is chiefly most amenable to smelting. On this account, as also on account of the excellent smelting operations conducted at the Granby Company's plant, Mr. Campbell is quite justified in his position not to venture on leaching operations in connection with copper sulphides. The idea, however, of leaching and electrolysis supplanting smelting, at least under some conditions, is not altogether visionary. A large copper company in Africa, that has been producing between four million and five million pounds of copper monthly for some years by smelting, is now installing a leaching and electrolytic plant, costing considerably over $750,000. This plant is merely a first unit, as operations on a much greater scale are contemplated and may even displace smelting entirely. This company spent much time and money in preliminary investigations, and must have considered the outlook for leaching and electrolysis very hopeful to warrant the expenditure on this account instead of on the enlargement of the smelting plant. A number of other large copper companies, owning and operating smelters, are seriously investigating the possibilities of substituting leaching sulphide ores and concentrate in the place of smelting. In view of this my "inevitable conclusion," objected to by Prof. Thomson, may not be so wide of the mark when taken within the limitations intended in my paper.

Mr. Campbell's statement that the leaching operations at the New Cornelia are the largest in the world, needs modification. They are large, but not the largest. The largest leaching and electrolytic plant is at Chiquicamata, Chile, which has a capacity of over 15,000 tons a day, and it is said that plans have been completed to increase this to 50,000 tons a day directly the copper market warrants the increase. The question of smelting or of leaching the New Cornelian sulphide concentrate

introduces an interesting problem. The same financial interests that control the New Cornelia also have a large smelting plant at Douglas, Arizona, and the treatment of this concentrate by a wet process at Ajo, before the oxidized ores are exhausted, would necessitate the installation of an additional hydrometallurgical plant there. Then, too, it is not impossible for the metallurgists at the New Cornelia to change their minds about smelting the sulphide concentrate, especially after the oxide ores are exhausted.

Mr. Dawson, in his reference to electrolytic zinc, may have had in mind the prediction some years ago of the editor (himself a zinc expert) of one of the leading mining papers, that electrolytic zinc would never be a serious factor in the world's zinc market. This opinion would now need some modification, and possibly, in the future, it may need inversion.

In the leaching and electrolysis of copper ores, perhaps it might be of interest to discuss very briefly the lines of attack which to me seem most hopeful. It will not be advisable to go into the history of past efforts; suffice it to say, that ever since the time of Body, the first patentee in copper leaching and electrolysis (in 1886), the matter has been under continuous investigation. The underlying difficulty has always been the same—namely, the disastrous effects of the ferric salts formed by the electrolysis. If this difficulty had been adequately surmounted, the metallurgy of copper would have been a different story in the past, and when it is overcome, it will create a profound revision of the metallurgy of copper in the future.

As little as 0.5% ferric iron in the electrolyte makes the deposition of the copper highly inefficient, and 0.75% is quite sure to make it unprofitable. Diaphragms have been disappointing. The precipitation of the iron preparatory to electrolysis seems hopeless. At Ajo and at Chiquicamata large quantities of solution are wasted daily to keep down the total iron in order to minimize the effects of the ferric iron.

As I see the matter, the right solution of the problem lies in making the iron useful—in making it work with the process instead of against it; in using a high-iron electrolyte instead of eliminating the iron from the leach solutions to a low percentage.

Since Mr. Campbell referred to the operations at the New Cornelia, it may not be amiss to analyze the problem in the light of their data, as presented by Tobelmann and Potter.[1] The copper solution, at the New Cornelia, passes successively through seven leaching vats, and then through three pairs of scrubbing towers, and from the towers to the tank room, or cells, and from the cells back again to the leaching vats. The solution circulates through the towers at the rate of from 1,005 to 1,324 gallons per minute. The average ferric content of the solution entering the tank house is from 0.21% to 0.354%; leaving the tank house, from 0.80% to 0.916%. The average copper content of the solution entering the tank house is from 3.06% to 3.12%. Leaving the tank house, from 2.621% to 2.71%. This solution is returned to the leaching vats, but in being returned, a certain amount is diverted for chemical precipitation, and wasted. It will be seen that in depositing only 13%, or 8.5 lb. of copper out of the solution containing 61.8 lb., in its flow through the tank room, the ferric iron is increased to 0.85%, and this is practically prohibitive. In the deposition of copper from high-iron electrolytes, free from ferric salts, it has long been known that the ampere efficiency closely approximates the theoretical. What, therefore, must be the efficiency at the beginning of the deposition when the ferric content is low, and what at the end when the ferric content is high, in order to yield an average of 0.8 lb. of copper per K.W.H.? It must be very evident that the efficiency at the beginning must be very high and at the end very low. Suppose that the ferric salt could at all times be kept below 0.25%, what would happen? Evidently the efficiency would be quite high all the time. In view of this inquiry, the tower data at the New Cornelia is most interesting, especially in the light of my own lines of investigation:

[1]Trans. A. I. M. E., Feb., 1919.

ANALYSIS OF SOLUTION IN AND OUT OF REDUCING TOWERS

	Ferrous Iron Per cent.	Ferric Iron Per cent.	Total Reduct'n Per cent.
Neutral advance entering towers......	1.61	0.79	....
Solution leaving first pair of towers....	1.95	0.45	49.5
Solution leaving second pair of towers.	2.14	0.26	27.5
Solution leaving third pair of towers...	2.25	0.15	16.0
Solution leaving settling tank.........	2.30	0.10	7.0

It will be observed that the solution, especially in the early stages of electrolysis, is fairly quickly reduced. The average time in the towers would probably not exceed a few minutes, when flowing through at the rate of 1,000 gallons per minute.

Suppose now (and this is important) that instead of passing the solution through the circuit as described, the reduction is made continuously in pools of the electrolyte alternating with a series of cells, and the rate of flow through the cells is regulated so that the ferric salts shall not exceed a predetermined limit, say, 0.25%; what would happen? Manifestly the efficiency would be greatly increased; and this is one of the lines fairly well developed by me. Instead of treating the copper solution momentarily with SO_2, as in scrubbing towers, I have a series of covered pools of the electrolyte under continuous treatments with the gas, while maintaining a continuous stream of reduced electrolyte from the reducers to the electrolyzers containing close to 0.00% ferric iron, and a continuous stream of oxidized electrolyte from the electrolyzers to the reducers containing, preferably, not over 0.25% ferric iron, while at the same time advancing a portion of the electrolyte progressively through the series of reducers and cells, and finally returning it to the ore. In this way the copper may be deposited out to as low as 0.50%, against 2.6% at the New Cornelia, and a low copper solution is more effective in dissolving the copper from the ore than a high-copper solution. Similarly, a high-iron solution will retard the solution of iron from the ore. Both time and temperature are vital factors in the reduction of the ferric salts with sulphur dioxide, and both of these conditions can be fully met by continuously treating large pools of the electrolyte with the uncooled gas from the roasting furnace. The reduction of the ferric salt, with sulphur dioxide, of course, yields an equivalent of acid. (See for example, U.S. Patents, 1,314,742 and 1,353,995.)

The next serious difficulty is encountered in the elimination of the foul solutions, and the recovery of the copper therefrom. All solutions wasted, represent a loss of acid, and as the copper is precipitated from these solutions with scrap iron, it also represents a waste of iron. At the New Cornelia it takes approximately 2.0 lb. of scrap iron to precipitate a pound of copper from the waste solutions. During the years 1917-1919, there were produced at the New Cornelia, 88,813,390 lb. of electrolytic copper, and 29,616,788 lb. of cement copper; or about 25% of all the copper produced was in the form of cement, which was shipped, smelted and refined, as in the case of ore or concentrate. This method of disposing of the foul and waste solutions not only requires a considerable amount of scrap iron, but much of the work connected with handling both the scrap iron and the cement copper involves much hand labour.

The best method, in my opinion, of overcoming this difficulty is to maintain an electrolyte of high iron content, as already described, so that the waste solutions can be greatly reduced, and as they contain only 0.50% copper coming from the tank room, this small remaining portion of copper is easily precipitated with hydrogen sulphide. The copper sulphide precipitate is then separated from the foul solution in a Dorr thickener, and washed in a similar apparatus, and this washed sulphide precipitate, without filtering or drying, is then, as an easy flowing sludge, applied to the electrolyte. This step serves two very important purposes: the ferric iron in the electrolyte, in the later stages of the electrolysis, is almost immediately reduced with the formation of copper sulphate, and this copper is then obtained in the regular way, as the pure electrolytic metal, with the simultaneous regeneration of an equivalent of acid. No scrap iron is used in the process. Only a comparatively small portion of the copper is precipitated chemically, and this, in the regular operation of the process, is obtained

as the electrolytic metal. The entire procedure can be carried out mechanically and practically automatically. (See, for example, U.S. Patents Nos. 1,340,826 and 1,357,495.)

The third important consideration, while perhaps more problematic than the two steps just described, offers great possibilities. This is the depolarization at the anode during the deposition of the copper. Lead anodes are not as well suited for depolarization as carbon anodes. The ultimate life of carbon anodes in sulphate solutions is somewhat uncertain. In pure sulphate solutions they are quickly disintegrated. In sulphate solutions, under effective depolarization, their use seems to be quite within reason. This position is supported by other investigators along these lines. In using lead anodes, the E.M.F. between the electrodes will average about 2.0 volts. If carbon anodes are used, the depolarizing action may reduce this to about 1.0 volt. This would mean a saving of 50% of the power both in installation and operation. It is very probable, although not certain, that this can be done in large scale commercial operations. It is evident that it must be done through effective depolarization, but, manifestly, effective depolarization can only be accomplished through effective reduction. As I have already described my preferred method of getting effective reduction, the general idea will be quite evident. There are other difficulties to be worked out in connection with the use of carbon anodes, but these do not appear to be insurmountable. For the purpose of depolarization it is best to work with warm electrolytes, but, in leaching hot roasted ore, and applying the hot roaster gas to the pools of electrolyte, the electrolyte will not only be warm but usually quite hot, so that this condition can easily be met. If the use of carbon anodes in electrolyzing leach solutions can be made practical on a large scale, it should be quite within reason to get 2.0 lb. of copper per K.W.H., and at least 5.0 lb. of acid. With power at 1c. per K.W.H., the cost of power for the deposition of 1.0 lb. of copper and regeneration of 2.5 lb. of acid, should not exceed 0.5c. It is reasonably certain that at least 1.0 lb. of copper, with the regeneration of 2.5 lb. of acid, can be realized per

K.W.H., even with lead anodes. If a mining enterprise should fail under these conditions, it would certainly not be due to the electrolysis, which at present appears to be the moot step.

With the conditions outlined, and with electrolytic copper as the end product, a very serious question would be raised whether many of the smelters now operating could compete in the face of such results, in the treatment of ores easily amenable to leaching. Where new metallurgical plants are contemplated, and the ore is amenable to leaching, it would not seem wise entirely to ignore such a promising field, especially in view of the fact that all essential factors entering into the entire matter of roasting, leaching, and electrolysis, can be quite accurately determined in advance with a small laboratory plant, and at a nominal expense. Such a plant will clearly forecast the conditions in a commercial installation of any size on the same material, if the operations are continued long enough to get complete cycles, or, say, continued for several months. Smelting has practically reached the limit of its technical possibilities. I know of no major problem in connection with smelting that has not been fairly well solved. Leaching and electrolysis, while old, is yet new, and many of its major problems still remain to be solved, or demonstrated in large scale commercial operations, and yet, even with its present limitations, it offers many evident advantages over smelting under certain conditions, and these conditions are very common.

I hope I will not be misunderstood as advocating leaching and electrolysis as a cure for all metallurgical ills. I wish to emphasize, however, that there are great possibilities, and that the possibilities are greatest where smelting is impractical, or at least unattractive, both in reference to the nature of the ore, and the location of the mines relative to transportation, fuel, and power.

THE DOMINION IRON AND STEEL COMPANY'S NEW SHIP-PLATE ROLLING MILL

By H. E. RICE

Annual General Meeting, Toronto, March, 1920.

This latest type of mill, erected at a cost of $5,000,000, and put into operation in February, 1920, is the result of negotiations initiated during 1918, between the Canadian Government and the Dominion Iron and Steel Company. The original contract terms and their later modifications are no doubt familiar to all interested in the matter and need not be discussed. The undertaking is one of national importance, for upon its success depends, to a considerable degree, the extension of the Canadian shipbuilding industry.

Site.—Immediately upon the conclusion of the negotiations the clearing of the site was undertaken. This involved the removal and re-location of seven large structures and numerous small buildings, the re-location of railway tracks, and the transference of immense quantities of miscellaneous stores material. The grading to yard level necessitated the removal of 200,000 cubic yards of rock and earth, and the earlier part of this work was carried out under winter conditions which made it especially difficult. (See Fig. 1.)

Foundations.—These occasioned the placing of 18,500 cubic yards of concrete, both plain and reinforced, for buildings and machinery foundations. The concrete for the heavy machinery foundations for the furnaces, motor and mill machinery was distributed from a 90-ft. tower having steel chutes, designed and erected on the site for the prosecution of this work. On numerous occasions, with this tower, over 400 yards was placed per day of 10 hours, using a ¾-yard mixer.

It is worthy of remark that the placing of concrete was carried on continuously during the winter months. This was accomplished by pre-heating the materials, by covering freshly placed concrete with tarpaulins, and, when required, by moderating the temperatures in the vicinity by means of steam and

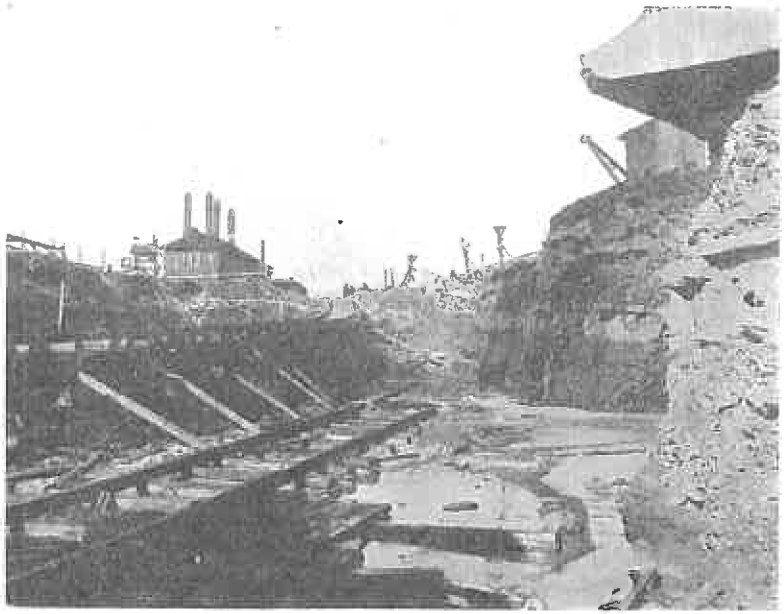


Figure 1.

open fires. It was clearly demonstrated that if concrete were placed when warm and subsequently protected for about 12 hours, trouble thereafter from frost need not be apprehended.

Sewers.—It was necessary to provide for an extensive sewer system. The depth of the excavations for the sewer were about 22 ft., and, wherever possible, were dug 'neat,' to avoid the use of outside forms. Collapsible forms were used for the centre, and were made so that they could be readily removed and re-used as the work proceeded. The walls of the sewer were of concrete 8 in. thick. The sewer itself was constructed of re-inforced concrete, 3 ft. by 2 ft. and 1,200 ft. in length. This drains the furnace and mill foundation sumps; all other machinery pits are drained by 12-in. pipe leading to the same sewer.

Buildings.—These cover a total area of 240,000 square feet, or 5½ acres, and are of the most modern mill type construction, embodying concrete foundations, structural steel frames, brick walls, and 'slow-burning' roof. A roof of this type is built of

2¾-in. tongue and groove plank and covered with 'Barrett specification' tar and gravel roofing. The steel frame work involved the erection of 3,500 tons of structural steel which was supplied and erected by the Dominion Bridge Company.

Simplicity and substantiality of construction was the aim of the designers of the buildings, and it was carried out in the erection. Reinforced concrete sills and lintels were moulded on the site and lifted into place by a locomotive crane, thus avoiding slow and expensive arch work over doors and windows. The erection of the buildings entailed the laying of over 3,000,000 bricks in the walls, the placing of 60,000 square feet of windows, and of 5,000 square feet of doors.

All the buildings are practically fireproof, are of a most permanent character structurally, and include very desirable architectural features; their cost was remarkably low, ranging from five dollars to ten dollars per square foot. The former figure represents the cost per foot of the shear building, the latter figure the cost per foot of the furnace and mill buildings.

Building and Mill Equipment.—A slab yard and ingot storage, having an area of 80 ft. by 200 ft., adjoins the north end of the mill buildings and is served by a 10-ton 3-motor E.O.T. crane. Narrow gauge tracks from the open hearth department pass under this crane and extend into the furnace building, so that hot ingots may be taken direct from the open hearth to the furnaces; or the ingots may be stored in the slab yard and later taken on cars to the heating furnaces. (See Fig. 2.)

The heating furnace building is 140 ft. by 227 ft. 4 in., and contains six gas-fired regenerative side-door heating furnaces 11 ft. 7½ in. wide by 51 ft. long, inside the buck stays; the width of the furnaces, inside of the brickwork, is 9 ft., and the length of the hearth is 34 ft. Gas and air regenerative chambers of ample proportions are placed beneath each of the furnaces, each of which is provided with a steel draught stack. the diameter of which is 4 ft. 6 in. inside the brick lining. The height

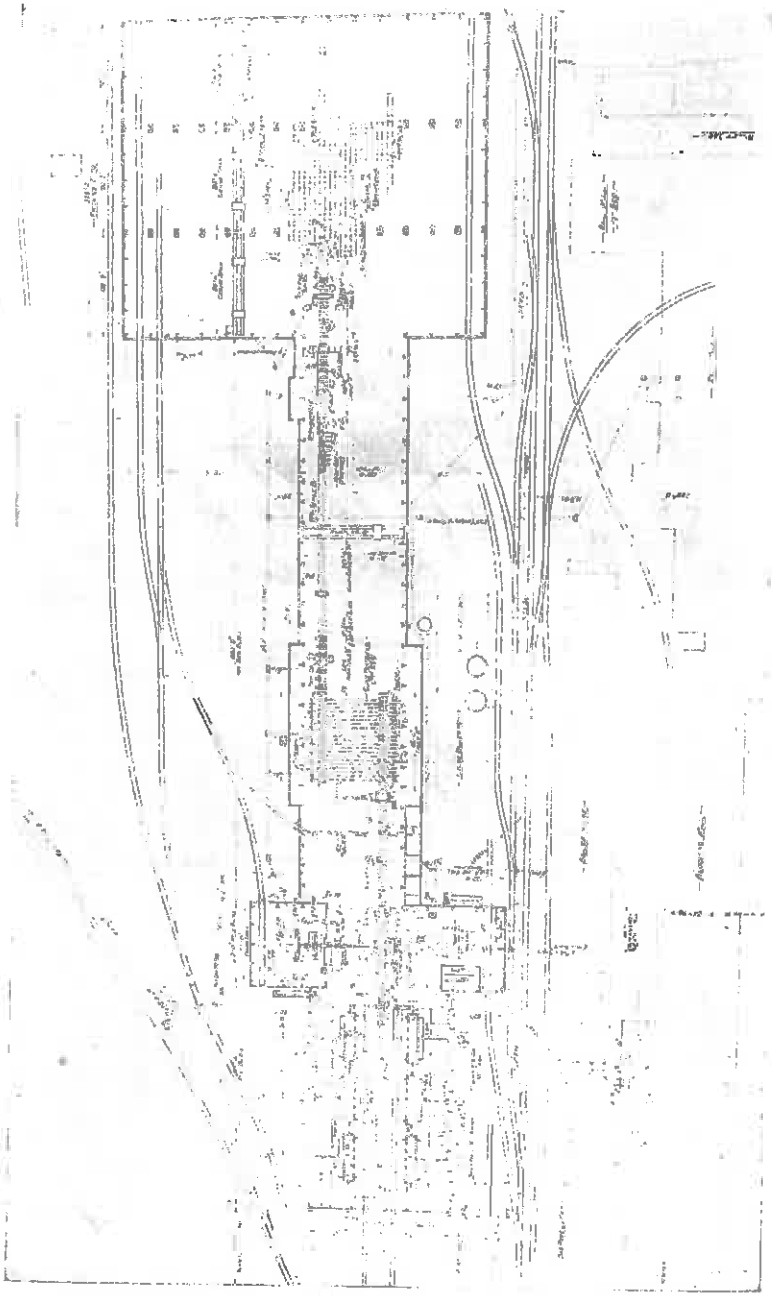


Figure 2.

of the stacks is 130 ft. Hand operated reversing valves are used.

It was decided that the quickest and cheapest way to erect these stacks was to assemble, rivet, and paint them on the ground, and then hoist them into place in one piece. In order to do this, an A-frame was built from stock lumber. When completed, the hoisting frame, or set of shear legs, was 85 ft. high. The stacks weighed about 24 tons each. (See Fig. 3.)

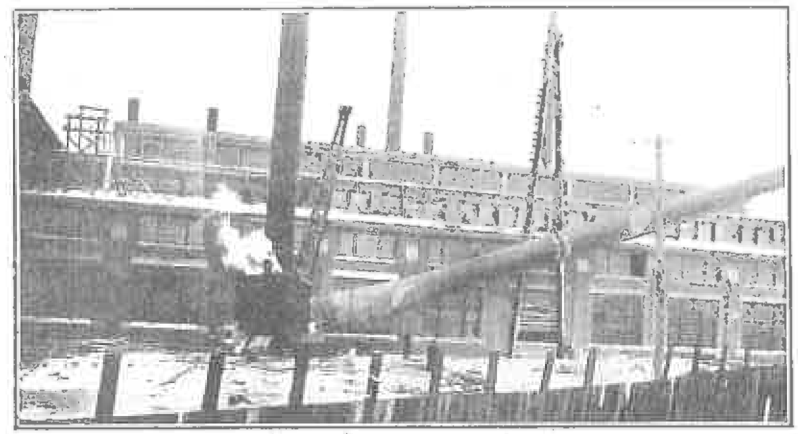


Figure 3.

The tackle employed comprised a ¾-in. steel cable, rove through two 3-sheave steel blocks, a locomotive being used for the main lift and a locomotive crane employed to take the weight of the base until the main hoist had lifted the stack to a vertical position. The stacks were reinforced inside and out, at the point where the sling was attached with wood bracing. This point was about 65 ft. from the base, or approximately 10 ft. above the centre of gravity. The workmen were much pleased with this method, and displayed their enthusiasm by reducing the time of the erection in each instance, and the last stack was lifted and bolted in place in the remarkably short time of twenty-two minutes.

Debenzolized coke oven gas, having a thermal value of 530 B.T.U., is used for fuel. This gas is brought through a 24-in.

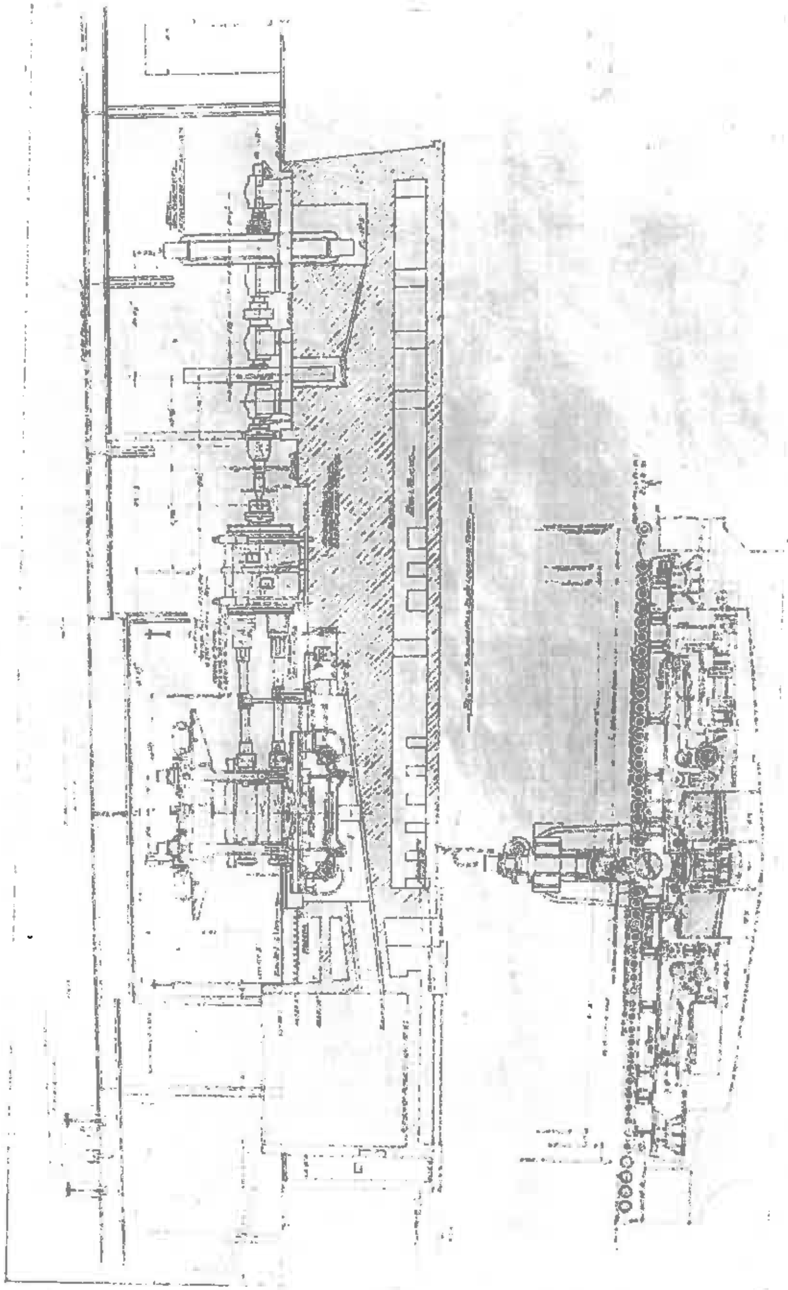


Figure 4.

main from the coke oven plant a distance of half a mile; and, as it is not preheated, both air and gas regerenators are used for preheating air. However, the arrangement of valves, flues, and ports is such that both gas and air may be regenerated in case coke-oven gas should be temporarily unavailable and it should become necessary to use producer gas for fuel. No gas producers have, however, been installed to date. Each furnace has four brick-lined cast iron doors 8 ft. 6 in. from centre to centre, the width between the door jambs being 6 ft., and the clear height from foreplate to skewback 4 ft. The furnace doors are individually operated by motors acting through a worm-geared crank motion. The water-cooled door jambs and skewbacks are of welded steel—the Blaw-Knox patent.

For average operation it is estimated that the coke-oven gas consumption will be 9,000 cubic feet per ton of ingots heated, while the water consumption per furnace for jambs and skew-backs will be about 120,000 imperial gallons per 24 hours. Venturi meters of the indicating, integrating and recording type are used for measuring both gas and water on these heating furnaces.

Two 10-ton, 5-motor revolving slab chargers, having a span of 55 feet, are arranged to serve the six heating furnaces in conjunction with an electrically operated ingot chariot arranged on the centre line between the furnaces. This ingot chariot and all of the twenty-four furnace door hoists are controlled from a pulpit at the north end of furnace building. It will be noted that all heating furnaces are arranged in plain view of the mill operator—a very desirable feature from the standpoint of operating efficiency.

The mill building proper is 80 ft. 10 in. by 250 ft. in plan, and houses the mill motor and mill proper, together with its tilting tables and auxiliaries, operating pulpit, roll shop, roll rack, salt bin and scale pit. The entire building is served by a 5-motor E.O.T. crane equipped with 50-ton main hoist and a 10-ton auxiliary hoist. A brick wall separates the motor room from the mill proper. This wall extends from floor level nearly to the crane girders, a fire-proofed canvas curtain being arranged to close the opening between the top of this wall and the roof, so that by raising this curtain, the 50-ton crane may serve the motor room. Standard gauge railroad tracks are arranged under each end of the 50-ton crane runway.

Mill.—This is known as a 110 in. x 36 in. three-high plate mill and is of the Lauth type. (See Fig. 4.) It consists of one stand of rolls served by tilting tables and is driven by a 4,000 h.p., 82 r.p.m. motor through a set of cut herring-bone pinions. The middle pinion is driven by the motor, while the upper and lower pinions are connected to the upper and lower mill rolls respectively, through leading spindles and coupling boxes. The pinion gear reduction is about 10 to 6 so that the mill is driven at about 49 r.p.m. The top and bottom rolls are 36 in. in diameter and the middle roll is 24 in. in diameter; the length of all rolls is 110 inches between necks, and the horizontal distance between mill housings is about 113 inches. The widest sheared plate that it is commercially practical to produce is about 96 inches. The mill screw-down drive is on the top of the housings and consists of two 100 h.p. motors connected to a common shaft, driving the screws through a worm-wheel reduction. These motors are electrically connected for series-parallel operation. The front and back tilting tables are each about 30 ft. in length and each supports twenty-one live rollers. Two 60 h.p. motors drive the live rollers on each tilting table. The power for raising and lowering the tilting tables is supplied by two 150 h.p. motors acting through a gear reduction and a crank motion. The tables are balanced by a closed hydro-pneumatic system acting on the crank motion through hydraulic cylinders and a hydraulic accumulator. The top roll is hydraulically balanced; the middle roll is mechanically balanced and electrically raised and lowered by means of two 150 h.p. motors.

Main Drive.—The mill motor is 88 pole, 4,000 h.p., 82 r.p.m., induction, 3-phase, 60 cycle, 6,600 volts. (See Fig. 5.) The flywheel is, direct connected, is mounted between its own bearings, and weighs approximately 155,000 lb., inclusive of

shaft. The diameter of this flywheel is 22 ft. 7 in.; the diameter of the motor rotor is 21 ft. 4 in. The total WR_2 of the rotating parts is approximately 20,000,000. The total weight of motor and flywheel is about 659,000 lb. The maximum running torque in pounds at 1 foot radius is 700,000. A 1,000-kilowatt motor generator set is provided to furnish direct current at 230 volts for all other motors used for the mill and its auxiliaries.

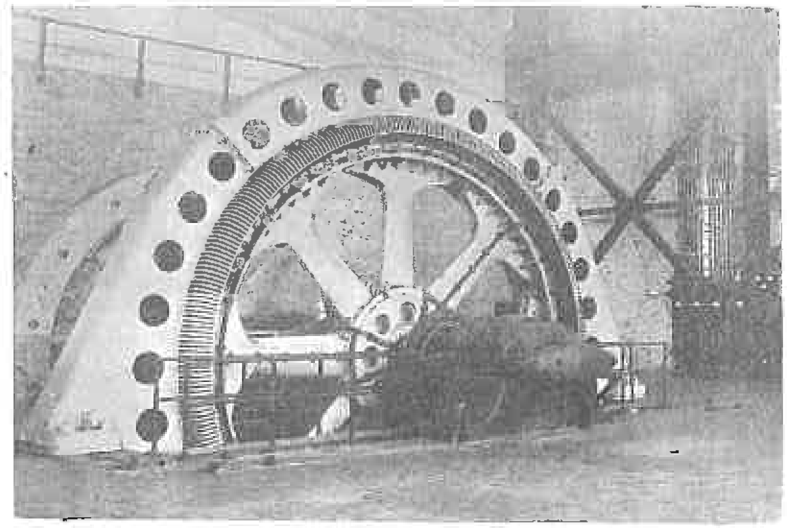


Figure 5.

The most important item, perhaps, in the building of this mill was the construction of this huge motor, which may be briefly described as follows:—On account of the enormous size and weights involved, it was necessary to assemble the stator frame, rotor, and spider, to press in the shaft, to build the laminated cores and to wind both stator and rotor, after the motor was placed in position on the base at the site. The stator and rotor core each contain over 60,000 laminated sheets of 14 mils thickness; these sheets were put in position one at a time and were so placed as to break joints, thus making, as nearly as possible, a solid core. The cores are 26 in. in width, thus making it necessary to press several times during construction in order to place the proper amount of sheets in each core. This work was accomplished by means of a special pressing outfit.

The stator winding is what is known as open slot construction. Each coil is a complete unit in itself and, in the event of any coil being injured, it can be readily removed and another substituted. These coils are held in position by fibre wedges, driven into grooves in the top of the coil slots. In addition to this, the projecting ends of the coils are tied down to a heavy supporting ring, thus preventing any chance of coil vibration. From time to time as they were placed in position, the coils were tested to ensure that they were in perfect condition. The testing was done under double voltage, namely 13,200 volts. After the coils were in position, but before the three phases were connected together, a double voltage test was made between the phases.

The rotor winding is what is known as the semi-open slot construction. The windings are held in position by wedges tightly driven under the projecting parts of the slot teeth; the projecting ends of the windings are clamped down to supporting flanges. This is accomplished by means of sectional binding bands. In the event of a damaged coil, these bands can be readily removed and a new coil substituted. These coils were also tested at double voltage (13,200 volts).

The pressing in of the rotor shaft was accomplished by means of a hydraulic jack; the pressure required was 370 tons. The rotor shaft is 20 inches in diameter at the bearings, which are 50 inches long.

The control equipment for this motor consists of a master controller and six automatic current-limit accelerating points and one plugging point, both forward and reverse, six of which may be hand-controlled from the master controller. In case of failure of voltage while running, the master controller must be returned to the off position before the equipment can again be started. Sufficient resistance is connected in the slip ring circuit to give approximately 15% slip at full load.

The flywheel consists of two sections of hub and spider and four sections of rim. The total weight of the flywheel rim is 120,000 lb., and the total weight of the motor and flywheel is 659,000 lb. The peak loads of the motor are absorbed by the flywheel effect of the rotating parts.

The motor is built to withstand an overload of 125% for two hours and a momentary overload of 250%. The installation of the motor and its auxiliary equipment was carried out under the supervision of the General Electric Company's construction engineering staff.

During the test, when this motor was first started, after shutting off the current, the motor drifted for an hour and forty-five minutes before it came to rest, a sufficient evidence of the splendid balance and alignment that had been attained by the erection engineers.

Over its entire length, the hot bed building, which is about 100 ft. wide and 560 ft. long, is equipped with a crane runway for a 10-ton, 3-motor, E.O.T. crane. A standard gauge railway track is arranged to enter this building underneath the crane runway. The plate mill machinery housed in this building consists of the approach table to plate leveller, plate leveller proper, hot bed, inspection turn-up, layout chain conveyor, and rotary shears and tables.

The shear building is 300 ft. by 350 ft. in plan and is equipped for its entire length with three 96 ft. 6 in. crane runways, each supporting a 10-ton, 5-motor, double trolley, E.O.T. crane. This building houses the 108 in. x 1¼ in. cross-cut shears, 156 in. x 1¼ in. trimming shears, 144 in. x 2¼ in. trimming shears, 5 scrap shears, and two 10-ton dial scales. The castor bed is arranged for the easy handling of plates from all shears to the scales. There are also excellent track provisions to facilitate shipment of the product, as well as extensive plate storage space which is so necessary for the economical operation of plate mills.

All cranes and mill machinery are motor-driven. Auxiliary apparatus, such as shear plate grinder, test piece milling machine, air compressor, hydraulic pump, etc., are also motor driven.

The supply of fresh water required for this mill amounts to about 1,500,000 imperial gallons per 24 hours. The cooling water for the heating furnaces is returned to a cooling pond and re-circulated, while the cooling water for the mill, which is contaminated with mill scale, grease and salt, is run into the sewer.

Power for the operation of the plate mill is supplied by a 5,000 K.W. turbo-generating unit installed in No. 3 power house (erected at a cost of $1,000,000), and is transmitted about a mile at 6,600 volts.

In order to provide room for bottom casting the slab ingots as required for plate mill consumption, extensions to the open hearth building were necessary. This auxiliary feature of the main installation involved the expenditure of approximately $600,000, and, without considering the necessary extension to the electrical power plant, brings the total cost of the plate mill to $5,000,000.

The work was entirely supervised by the Steel Company's engineering and construction departments, and the erection of all buildings and machinery, with the exception of the structural steel and the main motor, was directed by these departments. It is a remarkable fact that, of nearly 4,000 bolts in the machinery and building foundations, not one had to be changed; this testifies to the accuracy with which the drawings were prepared and the work executed.

THE BY-PRODUCT COKE OVENS OF THE GRANBY CONSOLIDATED MINING, SMELTING AND POWER COMPANY, LIMITED

BY W. A. WILLIAMS

Western General Meeting, Vancouver, November, 1919.

The Granby Consolidated Mining, Smelting and Power Company are the largest producers of copper in British Columbia. The consumption of coke at their smelter at Anyox averages 250 tons a day. Bee-hive coke was formerly shipped in to supply this need, but as these shipments were often considerably delayed, great inconvenience was caused in the operation of the smelter. To secure a more reliable supply, the company decided to produce their own coke. To carry out this plan a colliery, equipped in the most modern manner, was opened at Cassidy, near Nanaimo on Vancouver Island, and a coke plant was built at the smelter. The coal is screened at the mine, the marketable sizes sold, and the slack washed and shipped to the coke plant at Anyox.

Coal Handling.—Barges, each holding about 2,000 tons, convey the washed slack to the coke plant where it is unloaded by a steam hoist operating a one-ton hoisting bucket, having a capacity of 120 tons per hour. The hoisting bucket empties the coal into a hopper, whence it is carried by a 30-inch belt to the top of a 12,000-ton storage bin, where it is distributed by another belt travelling along the top of the bin. The storage bin, built entirely of wood, is 300 feet long and 51 feet wide, with a V-shaped bottom. From the bin, the coal is taken by conveyor belt and elevator to the crusher, which is of the swing-hammer type, and of a rated capacity of 50 tons per hour. The coal is crushed to a fineness that will permit the passage of at least 90% through a ¼-in. screen and 80% through a ⅛-in. screen. A bucket conveyor takes the coal from the crusher to the 500-ton bunkers over the ovens. The coal is drawn from the bunkers into a lorry, provided with four cone-shaped hoppers, the combined capacity of which is 13 tons, or just sufficient for an oven charge.

Ovens.—The battery consists of 30 ovens—capacity (24-hour coke) 270 tons—and was designed by the Gas and Coke Oven Corporation of America, a special feature being the vertical instead of horizontal flues. The dimensions of the coking chambers are 37 ft. 4 in. by 9 ft. 10 in., 18 in. wide at the pusher side, and 21 in. wide at the coke side, with a capacity of, approximately, 13 tons. A regenerator is situated directly under each oven and is separated from the adjacent regenerators by a heavy wall, upon which the heating wall rests. The latter consists of thirty vertical flues, every two adjacent flues forming a complete heating unit. Beneath these vertical flues are two gas ducts, which feed gas to them through removable nozzles. These horizontal gas ducts are so divided that fourteen of the vertical flues receive their gas from the coke side and sixteen from the pusher side. The regenerators are connected to waste heat flues paralleling each side of the battery, and are so divided as to furnish air to, and receive the products of combustion from, those flues, receiving gas from the same side.

The regenerators are divided into an upper and lower section by means of a horizontal partition extending from the front of the regenerator chamber to within eighteen inches of the back wall. Thus any tendency of the combustion air and products of combustion to short circuit is avoided and their rate of travel is accelerated. The efficiency of the regenerator system is greatly increased by placing this regenerative chamber, through which combustion air is being drawn, between two regenerators, through which the products of combustion are escaping, thus effecting what might be called a combined regenerative and recuperative system.

Battery Equipment.—The pusher was built by the Atlas Industrial Car Company, of Cleveland, Ohio, and comprises door extractor, pushing ram, and levelling bar. The coke guide and door extractor on the coke side are hung from an overhead track. The quencher car, also built by the Atlas Company, is provided with air-operated side-dumping doors. When the quenching car, with its load of red-hot coke, reaches the quenching station the water is automatically turned on, and is likewise automatically turned off, when the car leaves the station. After quenching, the coke is taken directly to the screening station where it is separated into furnace and breeze coke and loaded directly into cars. The handling of the coke is greatly facilitated by the fact that owing to a difference in elevation between the quencher and the loading tracks, the cars are loaded by gravity.

The gas is reversed every half hour, burning in one flue of each unit for one half hour and in the other flue during the succeeding half hour. Thus we have combustion taking place in every alternate flue throughout the length of the oven wall. The size of the opening connecting the two flues that comprise a unit is regulated by a slide brick, thus controlling the air supply for that particular unit. Above each flue is an inspection hole through which the condition and temperature of the wall may be determined. The openings leading from the regenerator to the stack flues are provided with dampers so that the draft to each regenerator may be regulated. The incoming combustion air is regulated by the use of finger bars. In each stack-flue is a damper at a point immediately below where these flues unite into a single flue at the stack end of the battery. This single flue, which leads to the stack, also contains a damper. The stack is built of radial brick, is 177 ft. high, 9 ft. 8 in. in diameter at the bottom, and 7 ft. at the top.

The gas is conducted from the oven chamber through 14-in. cast-iron ascension pipes to the collecting main which is 4 ft. 4 in. wide, 4 ft. 9 in. deep at one end, and 5 ft. 8 in. deep at the other, and has a semi-circular bottom. The off-take main is 28 in. in diameter; it continues to just beyond the bunkers where it enters the down-take (which is 24 in. in diameter), and thence along the ground to the by-product building. A hand valve to control pressure in the collecting main is in position just before the down-take. The temperature of the gas is considerably reduced in the collecting main and the heavier particles of tar and liquor condense and escape as a liquid. To keep the main free from thick tar and pitch, fresh tar is continuously circulated through it. This flushed tar drains off through a sealed overflow so arranged that any solid deposited may be raked out and removed.

On reaching the by-products building, the gas enters the primary coolers (of which there are three), each containing 594 lap-welded steel tubes, which are 17 ft. in length, and have an outside diameter of 3 in. The water flows through the tubes and the gas circulates between them. The temperature of the gas is here reduced to 32° C., and most of the tar and liquor is deposited. The gas then enters the exhauster, the function of which is to maintain an even pressure in the collecting main and to force the gas through the remaining apparatus.

The exhausters, three in number, were built by the Connersville Blower Company. They are driven by 10 by 10-in. vertical self-oiling Troy steam engines and are equipped with Huntoon float governors, arranged so as to maintain a pressure of 1 to 2 mm. in the collecting main. The exhausters have a displacement of 17 cubic feet per revolution and are designed to operate at 250 r.p.m. against a total plus and minus pressure of 2½ lb. per square inch.

After leaving the primary coolers, the gas still holds tar in suspension in the form of minute globules called 'tar fog.' The removal of these last traces is accomplished by means of a P. & A. tar extractor, which consists of a series of perforated steel plates, so arranged that the gas passing through the first plate will impinge upon the unperforated part of the second plate. The impact causes the very fine particles to coalesce and run down the plate.

Sulphate Recovery.—From the extractors the tar-free gas goes to the saturators (which are large lead-lined cast-iron containers), where it is caused to pass through a bath containing five to seven per cent sulphuric acid. The ammonia in the gas reacts with the acid to form a white crystaline salt, ammonium sulphate, which settles to the bottom of the saturator and is removed to the drain table by means of an air ejector. From the drain table the salt is conducted to a centrifugal

dryer where it is washed with water to remove the free acid, and then dried.

The liquor and tar which have condensed in the various stages of the process are pumped to a separating tank of 45,000 gal. capacity, where, in consequence of their difference in specific gravity, the tar and liquor separate. The liquor overflows into a 50,000-gal. liquor storage tank, and the tar is drawn off from the bottom to the 200,000-gal. tar storage tank. The tar produced averages eight imperial gallons per ton of coal. The liquor, which contains about one-fifth of the total ammonia produced, is treated in a continuous still where the ammonia vapours are liberated. These vapours are also passed through the saturator, which is so constructed that the waste gases which accompany the ammonia vapour from the still, may pass into the main gas stream leaving the saturator, or be conducted to the atmosphere.

The free leg of the ammonia still is composed of three 21-inch cast-iron sections and one 15-inch section, all resting upon the liming chamber which is 6 feet high. The internal diameter of all sections is 4 ft. 3 in. and the total height of the free leg is 15 feet. The fixed leg consists of two 21-inch sections and one 15-inch section of the same diameter as those comprising the free leg. Each section is provided with a vapour passage through the bottom covered with a hood with serrated edges and an internal overflow for liquor passing to the next lower section. Each section is provided with hand holes by means of which access can be had to every part of the interior so that the apparatus may be readily inspected and cleaned. The liquor enters the top of the still and drops from section to section. The steam which enters at the bottom, as well as any liberated vapours, passes up through the vapour passages and bubbles through the liquor at the edge of the hood. The boiling liquor, from which the free ammonia has been expelled, is mixed with milk of lime in the liming chamber and caused to flow into the fixed leg, where the remaining ammonia is liberated. The escaping liquor contains 0.003% to 0.007% ammonia.

The saturator is provided with a mother liquor heater which may be used to maintain the temperature of the bath in case the still is not working, or to heat up the bath when starting a saturator. Under normal working conditions the heat from the ammonia vapours from the still and the heat of dilution of the sulphuric acid are sufficient to maintain a temperature of from 45° to 50° C. The salt produced averages more than 20 lb. per ton and contains at least 25% ammonia and less than 0.2% free acid.

Final Coolers.—From the saturators the gas passes into the final coolers which are 40 feet high and 10 feet in diameter and are of the direct contact type, the water not only cooling the gas but mechanically washing out a large portion of the napthalene.

Light Oil Scrubber.—To recover the light oils, the gas is scrubbed with straw oil (a petroleum product) in a hurdle washer 75 ft. high and 14 ft. in diameter. The slats composing the hurdles are 5 in. wide, ¾ in. thick, and spaced ¾ in, apart; thus the gas in passing through the scrubber flows in thin streams over surfaces which are continually wet with the absorbent. The travel of the gas and oil is counter current. The oil is pumped to the top of the scrubber and flows in zig-zag fashion down over the hurdles, while gas enters at the bottom and leaves at the top; thus the partially de-benzolized gas is brought into contact with the fresh wash oil which is able more efficiently to remove the small amount of light oil yet remaining, and likewise the partially benzolized oil is brought in contact with the rich gas and its light contact oil increased. The amount of wash oil circulated depends upon the number of cubic feet of gas produced and its light oil content, and is of such an amount as to keep the light oil content of the benzolized wash oil between 2% and 3%. The benzolized oil is stored in a 3,000-gal. tank, from which it is pumped to the stripping still.

Heat Exchangers.—The de-benzolized oil leaves the still at about 130° C. and before it is re-circulated, its temperature is

reduced by a system of heat exchangers and coolers to about 20° C. The de-benzolized wash oil leaving the bottom of the still and the light oil vapours leaving the top of the still each contain a considerable amount of heat which is transferred to the incoming benzolized oil. The benzolized wash oil is pumped from the storage tank to the vapour-oil heat exchanger which consists of a cylindrical steel shell containing a number of tubes. The cold de-benzolized wash oil flows through the tubes, while the hot light oil vapours from the top of the still pass around the tubes. There are two vapour-oil exchangers so connected that one can be by-passed without interfering with the operation of the other. From the vapour-oil exchanger, the partially heated benzolized oil enters the oil to oil heat exchanger, where its temperature is increased by the de-benzolized oil leaving the still. After leaving the oil to oil heat exchanger, the benzolized wash oil enters a superheater where its temperature is raised to from 140° to 150° C. This superheater, furnished by the Alberger Heater Company of Buffalo, New York, is 8 ft. long and about 20 in. in diameter.

Stripping Still.—The hot benzolized oil now enters a stripping still which is continuous in its operation. The still comprises 17 sections, each section being 12⅜ in. in height and 4 ft. 4 in. inside diameter. The hot oil enters the eighth section from the bottom and passes down the still rapidly, giving up the light oil it contains. Steam is admitted in the bottom section and passes up through the still, bubbling under the sealing bells of each tray and carrying upward the light oil in the form of vapour.

As noted above, the de-benzolized oil, on leaving the still, passes through the oil to oil exchanger where it gives up some of its heat to the incoming benzolozed oil, then to the wash oil coolers. There are two wash oil coolers, a tank cooler containing eighteen 2½-inch steel pipes 22 ft. long, a spray cooler consisting of sixty 2-inch steel pipes 19 ft long. From the cooler the oil goes to the de-benzolized oil tank and is ready for re-circulation.

Fractionating and Purifying Stills.—From the 23,000-gal. light oil storage tank, the light oil is pumped into the crude still, which consists of a horizontal cylindrical kettle holding 3,800 gal. and a fractionating column of 14 sections resting on a separate foundation. Heat is supplied by means of indirect steam through sixteen 1½-inch extra heavy pipes 13 ft. 10 in. long, placed in the bottom of the kettle so that direct steam may be used when desired. The presence of live steam lowers the boiling point and is necessary when the toluol and solvent fractions are reached

The operation of the raw still depends upon whether motor fuel or pure products are to be produced. The first portion to distill over is often discarded because of the carbon bisulphide it contains. When making pure products the crude distillate is divided into 90 benzol, 90 toluol, and solvent naphtha. There will always be a residue left in the kettle consisting of wash oil, naphthalene, etc.

These crude products are then transferred to a lead-lined agitator where they are washed with sulphuric acid for about 30 minutes, after which another 30 minutes is allowed for the used acid to settle. The acid is employed to separate the unsaturated hydro-carbons, principally olefines. After the sludge is run off the benzol or toluol is washed with water, and any acid remaining is neutralized with caustic soda.

The washed products are now ready for rectification in the pure still. The pure still is similar to the crude still in construction but greater care and skill are required in its operation. Each 1,000 gal. of light oil will produce about 500 gal. C.P. benzol, 100 gal. C.P. toluol, and 100 gal. refined solvent naphtha.

When making motor fuel, the benzol, toluol, and such higher portions as will give a finished product, which will distill to complete dryness at not over 135° C., are caught in the same receiver. After the crude motor fuel has been run off, there still remains a solvent naphtha fraction boiling at from 130° to 160° or 170° C. The crude motor fuel is also washed with

sulphuric acid but not so thoroughly as in the case of pure products. This washed motor fuel is then passed through the pure still in order to remove the last traces of water and any foreign substances which do not settle after washing.

The products made are as follows: coke, gas, tar, ammonium sulphate, benzol, toluol, solvent naphtha, and naphthalene.

The plant has a boiler house containing two 350-h.p. boilers that operate the by-product and benzol plants as well as the unloading apparatus on the coal docks. There is also a well-equipped office and laboratory. All the buildings are of steel and concrete, and the plant is well constructed throughout.

DISCUSSION

MR. F. W. GRAY: The installation of the first by-product plant on the Pacific coast is a matter of considerable significance. Some years ago, I remember the first by-product ovens that were installed in England, near Sheffield. Since that time, the growth of the by-product industry has been very great, and this year for the first time, I believe, the amount of by-product coke in the United States exceeds that of any other coke.

The prejudice against by-product coke has been hard to overcome. The tendency of by-product coke ovens at the present time is towards narrow coke chambers and shorter coking time. The Dominion Steel Company of Sydney have finished what is probably the finest plant in North America. It is fully as fine as any in the United States though probably not as large. The coking chamber in this plant is about seventeen inches wide which is as narrow as can be used with advantage. The coking time—about 17 hours—is very short. When you compare that with the former coking time of 72 hours, it means very much quicker operation. In Sydney, about 4,000 tons of slack coal are coked each day, and it is all used in the Dominion Iron and Steel Company's blast furnace operations.

The Steel Company of Canada at Hamilton have just completed a new plant. The Algoma Steel Company have also finished a new plant. At the present time we have four by-product coke plants in Canada, at Sydney, at Hamilton, and now at Anyox.

MR. S. S. FOWLER: This installation tends toward the conservation of waste, which is in the right direction and I trust will be extended. Mr. Gray has referred to this plant as being the first of its kind west of the Rocky Mountains. As a matter of fact, however, in 1895 a French company, known as the Western Canadian Auxiliaries, built a by-products plant at Lille, Alberta, three miles north of the Crowsnest railway. They had a very good plant, most modern in type, but commercially the enterprise was an unqualified failure for the simple reason that they could not dispose of their by-products, and all they did sell was coke, which went chiefly to the B.C. Copper Company's smelter at Greenwood, B.C. The coke was of very excellent grade, but the point is they did not do anything, so far as I have been able to learn, with their by-products. Mr. Williams has not explained what the Granby Company does with its by-products, but it may be assumed that a market has been found for them.

MR. W. R. WILSON: The ovens referred to by Mr. Fowler could not be classed as by-product ovens, for I understand that they were merely plain rectangular ovens discharged by mechanical means. There is no question in my mind whatever about the propriety of constructing by-product ovens. The principle of making coke by this form of oven is theoretically correct under justified circumstances. Before venturing to make investments of this nature, however, companies should have a reasonable assurance that they will be able to dispose of the basic products of the ovens in sufficiently regular quantities to protect the capital risk.

The statements made by the author of the paper on this subject set forth some of these principles in a very deserving manner. The inference is that there is a terrible waste in the production of coke by the old system of ovens. I am full in accord with this point of view under given circumstances.

Forty years experience in the manufacture of coke under specific conditions has led me to this belief. During the last four years, I have had the necessary opportunities afforded me for fully investigating some of the different types of by-product coke ovens now in operation in several of the large manufacturing centres of the United States. These investigations were made with a view of ascertaining whether or not such installations might be expediently installed at Fernie, where the opportunities for disposing of coke are not only very irregular but oftentimes very unsatisfactory.

I, however, desire to concur with the views of Mr. Williams that the conditions at Anyox offer reasonable justification for the installation of a by-product plant at that point.

The installation of the new plant at Anyox, and the benefits that the Granby Company will derive therefrom will to some extent be at the expense of Fernie companies. The coke that was taken from Fernie to Anyox will now be produced at the new plant under much more favorable conditions to the Granby Company. This change will finally, I hope, be beneficial to all concerned. The curtailment in Fernie production will have an awakening effect upon our organization, and, I hope, prove a further stimulus to the efforts of our Company.

Association is an excellent thing; opposition in business is also an excellent thing. It is largely through these stimulating forces that our vision becomes broadened and cleared.

THE GRANBY CONSOLIDATED MINING, SMELTING & POWER COMPANY'S COLLIERY AT CASSIDY, VANCOUVER ISLAND

By R. R. Wilson

Western General Meeting, Vancouver, November, 1919.

The colliery is situated at Cassidy about eight miles south of Nanaimo, on Vancouver island. It was acquired and opened mainly to ensure a supply of coke for the Granby Company's copper smelter at Anyox. At Cassidy, a seam of coal about 10 feet in thickness outcrops in the bed of the Nanaimo river, the seam being known as the Upper Douglas, from which the first coal was mined on Vancouver island in 1852 by the Hudson's Bay Company.

Preliminary Developments

The townsite having been cleared, a start was made in opening the mine, and in June, 1918, the first coal was hoisted from the main slope. The main slope was sunk through gravel and quicksand, which necessitated the use of tongued and grooved spiling. A railway spur, three-quarters of a mile long, was then constructed to connect with the Esquimalt & Nanaimo Railway at Cassidy, and a temporary loading plant was installed so that coal could be shipped as development work progressed and until the permanent tipple and washery could be constructed.

The school is to be erected close to the athletic field, and playground equipment is to be installed so that the school children will be enabled to take advantage of all the facilities provided for clean, healthy sport.

The company has planned to provide ideal living conditions for its workmen, and to this end has arranged that they shall have ample opportunities for both physical and mental relaxation. Thus there is a recreation hall, and in connection therewith a gymnasium, a dance hall, a library, a reading room, and a billiard and pool room. The town is within a short distance of bathing beaches, is situated in a first-class game country where

pheasants, grouse, deer, wild duck and other game are plentiful, and is within a few hundred feet of the best fishing waters on Vancouver island.

The homes are neat and commodious, the architecture varied, and each house is equipped with modern conveniences. The streets are lined with shade trees and lighted with electric lights.

The rooming house for the accommodation of single employees is a granite structure built in the form of a double L. It contains about 80 rooms, all of which open to the outside veranda or balcony. The rooms are heated by steam and lit by electricity, and each room has hot and cold water. The floor is a patent material 'Raccolith'; and the rooms can be washed out with a hose when necessary. On the verandas and balconies are window boxes of flowers.

The Townsite of Granby

The area set apart for a residential district comprises about 80 acres of bench land overlooking the Nanaimo river to the north, Halsam creek to the south, and is sheltered on the east and west by a wooded ridge which is being preserved as a park. In the background can be seen Mounts Buttle, Tyee and other mountains. No more beautiful industrial townsite could be found anywhere.

The townsite was carefully planned to present a pleasing appearance. The streets are boulevarded and the houses surrounded by fresh green lawns and flowers.

The colliery has one of the finest athletic parks in the country. There is a baseball diamond, football ground, tennis courts, bowling green, and quarter-mile track. The furniture and bedding are supplied by the Company.

Mess House.—The mess house or dining hall is also a granite structure. The men enter the building through a lobby equipped with wash basins and running hot and cold water so that they may enjoy a refreshing wash, hang up their hats and then proceed through a pretty vine-covered pergola to the dining hall. At the entrance to the dining room, there is a drinking fountain. The dining hall is bright and comfortable, cool in summer and steam-heated in winter. Each table accommodates six men. No enamel dishes are used.

The kitchen is equipped with every labour-saving and modern device, including electric dishwashing machine, vegetable paring machine, tables heated by steam coils to keep dishes hot, large bake oven, and refrigerating plant. Living accommodation is provided upstairs for the help. In order that there may be no waste, the scraps from the mess house are fed to pigs. A vegetable garden will furnish all vegetables for the mess house.

The Change House.—The change house is equipped with steel lockers, and with shower baths and large lavatories.

MINE BUILDINGS AND EQUIPMENT

Between the change house and the manway portal are the powder house in which the stock of explosives is limited to one day's supply (the larger magazine being on the opposite side of a hill from the town), the timekeeper's office, lamp house and mine rescue station. The lamp house is equipped with 300 Edison storage battery electric lamps. The mine rescue station is equipped with Gibbs apparatus, lungmotor, smoke chamber, etc., and has a large lecture room for first aid or mining classes.

The above mentioned buildings are all heated with exhaust steam from the power house.

Tipple and Washery.—The tipple is equipped with Fairbanks scale, rotary dump, Marcus screen and loading booms. The railway cars are handled with Fairmount car retarders. The track scale is a Fairbanks standard, is of steel and concrete, and has a capacity of 100 tons. The rock cars are handled with a special Wilson gravity rotary dump.

The washery is equipped with two 2-compartment jigs, each having a capacity of 40 tons per hour. The tipple and washery were designed by Roberts & Schaefer, Chicago. The washery is equipped with sludge recovery and uses the same water over and over again.

The washed slack is used in the new by-product plant at Anyox in making coke for the copper smelter. The lump, nut, and some pea coal are sold. The bone coal is burned under the colliery boilers.

Power House.—The boiler plant at present consists of two Badenhausen water-tube boilers, 260 h.p. each, fired by type E mechanical stokers. The ashes are removed by washing and fluming to the dump. The feedwater is heated with a Webster feedwater heater, and forced draft is used.

The brick stack is 8 feet in diameter and 125 feet high. The boilers and steam pipes are insulated with asbestos and magnesia to prevent loss of heat. Venturi meters are used to check the quantity of water at the pump station and at the boilers.

The compressor is a Rand of the cross compound condensing type, and has a capacity of 2,000 cu. ft. of air per minute. The air is used for running the underground drills, pump, and hoists.

Electric power is supplied by an Allis-Chalmers 450 K.W. generator (2,300 V. 3 phase, 60 cycles, 360 r.p.m.) and also by an auxiliary unit 250 K.W. (2,300 V., 3 phase, 60 cycle, 450 r.p.m.), both direct connected to vertical high-speed engines (Goldie & McCollough). The remainder of the electric equipment is of Westinghouse make. The power-house is equipped with the Bowser oil handling system. A Worthington fire pump (capacity 1,000 gal. per minute, size 18″ x′ 10″ x 12″) is ever in readiness in case of emergency.

The entire plant is heated with exhaust steam, the condensation being returned to the boilers.

Carpenter, Machine, and Blacksmith Shops.—The shops are well lighted, and will be connected with the mine tracks. The carpenter shop is fitted with a rip saw, band saw, planer, boring and mortising machine. The machine shop is equipped with a large lathe, small lathe, planer and shaper, pipe threading machine, drill press, emery wheel, etc. The shafting is all well guarded. The master mechanics' office adjoins the machine shop. The blacksmith shop is fitted with two forges, steam hammer, and swing crane. Adjoining the blacksmith shop is a special tool house where miners' picks are kept after sharpening. All scrap iron is sorted out and stored in pockets provided for the purpose. Racks are provided for storing stocks of iron and steel.

THE MINE

The seam dips at about 18 degrees; the coal varies in thickness from 5 to 20 feet, the average thickness being about 10 feet. The roof and floor are shale and subject to rolls. The roof is fairly regular, most of the rolls occurring in the floor.

The mine is opened on the dip of the seam, the main slope having been driven to a depth of approximately half a mile. This slope is being driven 7 ft. x 14 ft. in the clear to allow for a double track and is timbered with 12-in. to 14-in. framed sets spaced at 4-ft. centres. A separate mainway is provided as a travelling road. Employees are not allowed to use the main haulage way in passing to and from their working places. The mine; which is worked on the pillar and stall system, is being divided into relatively small panels as a precaution against mine fires. Large pillars are left along all main haulage roads and permanent airways, the aim being to extract a maximum amount of coal at least cost rather than to take out cheap coal for a few years to the final detriment of the mine as in the case of so many of the mines on Vancouver island and elsewhere.

So far as practicable, the coal will be delivered from the face to the main haulage system by gravity. Storage battery locomotives are employed on the levels underground. No horses or mules are used.

The drainage system has been carefully planned so that surface water entering the mine from the gravel will drain by

gravity, and water from workings below the drainage level will gravitate to a central sump.

Ventilation is secured by a Sirocco fan, having a capacity of 150,000 cu. ft. per minute. The mine is provided with a double intake and return airway throughout, and the workings are planned so that the air can be taken to the face where required with a minimum loss.

The fan house is a concrete fireproof structure. It also houses the telephone exchange, as well as the motor generator set for charging storage battery locomotives. The fan is driven by a 150 h.p. Westinghouse electric motor.

The main hoist is a Vulcan (18 in. x 36 in.) double drum second motion hoist.

HAULAGE BY ELECTRIC STORAGE-BATTERY LOCOMOTIVES

By J. A. McLeod

Northern Alberta Branch, Edmonton, December, 1919.

Storage batteries are sometimes called secondary batteries in order to distinguish them from the ordinary type of battery, which is called a primary battery. A secondary battery is one that is used as an accumulator or container to store up electricity, and it must be charged by an electric current to be of service. The primary battery is one that generates its own electricity or current by the action of chemicals on plates of metal or other substance.

The two types of batteries in most common use in electric locomotives are, the lead-acid type and the Edison nickel-iron-alkaline type, the latter consisting of a combination of iron and nickel elements in a non-acid electrolyte.

In the lead-acid type, the positive plates are made by mixing red lead with sulphuric acid to the consistency of a paste, and pressing it into lead grids or frames. The negative plates are made of metallic lead ground into a powder and pressed into similer grids or frames. The plates are then assembled into batteries, using wood or rubber separators to keep them insulated. Sulphuric acid is then poured in and they are allowed to stand for ten or twelve hours before charging in order that the sulphate may be absorbed by the plates. The batteries are now put on charge, care being taken to ensure that the positive terminals are connected to the positive wire of the charging current and negative terminals of battery to the negative wire of the charging current. As the current flows through the battery, the sulphate from the plates passes into the water, converting it to sulphuric acid, and when this reaction is completed the batteries are fully charged. When the battery is put into use, a reverse action takes place until it is discharged.

In charging storage batteries, only direct current can be used, as alternating current would spoil the batteries. For

companies which have alternating current only, it is a simple matter to install rectifiers on the generator sets and thus convert alternating into direct current. In charging the lead-acid type of battery it is important to insure that the current is not in excess of the ordinary rate, as an overcharge will cause the plates to buckle and 'sulphate.' This type of battery, therefore, should have the best of care. The Edison batteries we are at present using differ from the one just described. In the first place, they may be termed 'fool-proof,' as they cannot be injured by overcharging, undercharging, or by connecting them to the wrong terminals. To become effective, however, the battery must be charged in the proper manner. The gases given off when this battery is being charged are inflammable but not noxious. If ordinary care is taken with this type of battery, exceedingly good results will be obtained. The electrolyte should be changed every nine or ten months.

Determining Size of Battery Locomotive to be Used

Although storage batteries have been in limited use in mines for the past eight years, it is only recently that they have come into general use, and as yet many operators and mine superintendents do not understand them sufficiently to decide on the proper size to employ. It may happen that a superintendent has seen a locomotive of a certain weight hauling the amount of coal which is required to be hauled at his mine, and consequently he orders the same type of locomotive from the manufacturers, only to find, after using it for a short time, that it will not haul the same amount of coal as the locomotive on the strength of whose performance he placed his order. Conceivably, he will lay the blame on the manufacturers, whereas the failure may be attributable to a number of causes, for the haulage capacity depends upon the following factors: weight of locomotive, tires, speed, H.P. rating of motors, grades, weight of pit cars, type and condition of wheels on the pit cars, condition of track, length of haul, number of cars per trip loaded, number of cars per trip empty, and number of trips per day.

A locomotive that is not large enough to do the work required of it is a failure, while one that is too large is a waste of power and money. Hence the first information required is the weight of locomotive to draw-bar pull, and although this does not necessarily mean that all locomotives of the same weight will have the same draw-bar pull, those of the same weight, construction and type will have the same pull. At the present time there are several different types of locomotives on the market. The draw-bar pull to move mine cars varies greatly with the condition of the track and the rolling stock, and may be from 15 to 50 lb. per ton. An average of about 30 lb. per ton is usually taken. On grades, 20 lb. additional per ton is required for each per cent of slope, independent of the friction of the wheels. The maximum draw-bar pull on a locomotive in pounds (on a clean, straight rail) is equal to one-fifth of the weight of the locomotive for iron wheels, and one-quarter for steel-tired wheels, for greater pulls excessive slipping of the wheels occurs.

The H.P. developed by the motors equals:

$$\frac{\text{Tractive effect speed of locomotive in feet per minute}}{\text{33,000 mechanical efficiency of locomotive}}$$

For example, with a 5-ton locomotive with steel-tired wheels, having a speed of 7 miles per hour:

$$\text{Tractive effect} = \frac{2{,}000 \times 5}{4} = 2{,}500 \text{ lb.}$$

$$\text{Speed in feet per minute} = \frac{5{,}280 \times 7}{60} = 616 \text{ ft.}$$

Assuming the mechanical efficiency as 90%, then

$$\text{H.P.} = \frac{2{,}500 \times 616}{33{,}000 \times 0.90} = 51.5$$

Again in the case of a 5-ton locomotive with a maximum tractive effect of 2,500 lb., the total draw-bar pull on various grades is as follows:

Draw-bar pull on level = 2,500—30 x 5 = 2,350 lb.
" " on 1% grade = 2,500—(30 x 20) x 5 = 2,250 lb.
" " on 2% " = 2,500—30 x (2 x 20) x 5 = 2,150 lb.
" " on 3% " = 2,500—30 x (2 x 20) x 5 = 2,050 lb.
" " on 4% " = 2,500—30 x (4 x 20) x 5 = 1,950 lb.
" " on 5% " = 2,500—30 x (5 x 20) x 5 = 1,850 lb.

The number of tons this locomotive will haul on different grades is as follows:

$$\text{Level} = \frac{2{,}350}{30} = 78.33 \text{ tons.}$$

$$1\% \text{ grade} = \frac{2{,}250}{30 \times 20} = 45.0 \text{ "}$$

$$2\% \text{ "} = \frac{2{,}150}{30\ (2 \times 20)} = 30.7 \text{ "}$$

$$3\% \text{ "} = \frac{2{,}050}{30\ (3 \times 20)} = 22.6 \text{ "}$$

$$4\% \text{ "} = \frac{1{,}950}{30\ (4 \times 20)} = 17.6 \text{ "}$$

$$5\% \text{ "} = \frac{1{,}850}{30\ (5 \times 20)} = 14.4 \text{ "}$$

Assuming that pit cars weigh 2,000 lb. and have a capacity of 5,000 lb., the number of cars and tons of coal this locomotive is capable of hauling on different grades is as follows:

Grades	No. of cars	Tons of coal per trip
Level	22	56
1% grade	13	32
2% "	9	22
3% "	7	17.5
4% "	5	12.5
5% "	4	10

It is vitally important to keep locomotives in good repair and running order. Nothing is more annoying than to discover after a locomotive has started on its run that something is wrong; this discovery may cause a delay of an hour or more before the repairs are made, and may be the cause of considerable loss in output to the section that the locomotive is serving. Possibly, moreover, at some places there will be only one or two cars left at the end of the shift, with the result that the machine-runner will be unable to cut these places, and the loaders on the following day will only have the loose coal to load. This is a serious loss to the loader as well as of output. In order to prevent such occurrences, our electrician examines every machine as soon as it comes in; and at the end of the shift the motorman is required to fill in a report under the following heads:

MOTORMAN'S REPORT

Date. . . . . *Loco. No.* . . . .
Motor defects . . . .
Loco. or car off track . . . .
Time . . . .
Place . . . .
Cause . . . .
Time lost . . . .
Time fixed . . . .
Location bad track . . . .
Location dirty road . . . .
Location supplies on road . . . .
Location low or bad roof . . . .
Remarks . . . .

Foreman will initial line marked "Time Fixed"

Signed Motorman . . . .
" *Pit Boss* . . . .
" *Manager* . . . .

In this way, if a motor goes wrong at the beginning of a shift, reference to the report will indicate on whom the blame should be placed if a defect was reported and not repaired.

At the present time we have in operation four storage battery locomotives and five sets of batteries. Our method of mining is room and pillar, and we are hoisting about 400 cars averaging about 2.2 tons of run of mine coal per day of 10 hours' hoisting, or an average of 80 cars per shift per locomotive, one being operated on the afternoon shift.

Some of our locomotives are pulling more coal than others, this difference being ascribable to the varying conditions under which they have to work. Also a great deal depends upon the motorman and switcher for successful operation.

The rate of wages paid to a motorman is \$3.79 plus .92, or \$4.71; and to a switcher \$3.47 plus .92, or \$4.39 per 8-hour shift. We employ five motormen and five switchers, whose wages aggregate \$45.50 a day, making the total cost for gathering 5.17 cents per ton.

STORAGE-BATTERY LOCOMOTIVES AS MAIN HAULAGE MOTORS

By J. Shanks

At a meeting of the Rocky Mountain Branch of the Canadian Mining Institute, held in April, 1916, Mr. R. Green, of Blairmore, Alta., read a paper on "Horse Haulage Versus Compressed Air Haulage—A Comparison of Costs."[1] This paper gave rise to considerable discussion. The present writer sent, at that time, a communication[2] to the Secretary in support of the conclusions of Mr. Green, viz., in that after a certain ton-mileage has been reached, horse haulage is much more costly than compressed air locomotive haulage. The actual experience of Mr. Green at Blairmore, and of the writer at Nordegg, with horses on the main haulage roads during the spring of 1916, showed a cost per ton-mile as follows:—

Horses at Blairmore—$0.109 per ton-mile for 14,450 ton-miles per month.

Horses at Nordegg—$0.093 per ton-mile for 21,160 ton-miles per month.

Compared with compressed air high-pressure locomotives at Blairmore, of $0.078 per ton-mile for 70,260 ton-miles per month.

In the Spring of 1906, the writer was advising his directors to introduce mechanical haulage to replace horse haulage on the main haulage road. The main levels at Brazeau Collieries are driven on a grade, in favour of the loads, of from 0.60 to 0.70 per cent.

The coalfield at Brazeau is slightly rolling. This causes the main haulage road (being driven to a fixed grade) to curve considerably, so much so, as to put the adoption of rope haulage out of the question when considering mechanical haulage. The choice of haulage was therefore confined to motors of the locomotive type. The Coal Mines Act forbids the use of electric trolley locomotives and gasoline locomotives, so that the choice of motors was confined to compressed air and storage-battery locomotives.

[1]Trans. C. M. I., Vol. xix, page 247.
[2]Ibid, page 251.

The writer had had considerable experience with compressed air locomotives and knew their advantages and disadvantages. Of storage-battery locomotives as main haulage motors, little information could be obtained, except the experience of the McGillivray Creek Coal & Coke Co., at Coleman. Their storage-battery locomotives had been entirely successful as main haulage motors, and full information in this connection was obtained from this Company. Manufacturers and experts were consulted, with the result that two 5-ton electric storage-battery locomotives, each with a spare battery, were procured from the Canadian General Electric Company.

These locomotives were put into operation in February, 1919, and have (with the exception of trouble with one motor, due to its having been incorrectly connected at the factory) given great satisfaction. One of the main factors considered in deciding on the installation of storage-battery locomotives was the low first cost as compared with that of a compressed air high-pressure plant.

At the Brazeau collieries the situation is somewhat similar to that found at many collieries in the West, viz., the boiler plant and power-house units are working almost at maximum capacity during the day, and working to between 30% and 50% of their capacity only during the night. This spare power during the night is available for charging storage-batteries to be used on locomotives during the day shift. This arrangement allowed the use of the power plant to its maximum capacity during 24 hours, and saved the expense of installing extra boilers, which would have been required if compressed air locomotives had been installed. The cost of operation of the storage-battery locomotives in question, with two spare batteries, is tabulated below. In the table, to facilitate the comparison of costs, the figures have been arranged in a similar manner to those in the paper by Mr. Green and in the discussion thereon. Wages costs are based on the wages in the 'Tentative Agreement.' The war bonuses and the 14% increase are not taken into consideration, as these were not in force during 1915 and 1916, when former estimates of costs were compiled. The month of October, 1919, is selected, and costs for January, 1920, are also given. During these months the outside temperature ranged from 40° below zero to 15° above. No. 2 Mine locomotive worked almost at capacity during day shifts, and on a test run in zero weather, hauled 70 empties (tare, 1,100 lb.), an average distance of 1½ miles, supplied four sidings with empties on the in-run, and gathered the loads from four sidings on the out-run. The number of loads hauled out to the mine mouth was 70, coal tonnage was 100, power consumed was 50 amperes at an average of 100 volts for the in-trip with empties, and 10 amperes for the out-trip with loads; the rear car carried two sprags to steady the trip. The time taken for the return trip on the test was 75 minutes. Allowing for usual delays incident to operations, this locomotive can haul 100 tons 1¼ miles in one hour, or its working capacity is equal to 125 ton-miles per hour.

Each locomotive enters the mines, Nos. 2 and 3, at 8 a.m., runs four trips before 12 noon, changes batteries and runs four trips between 12 noon and 4 p.m. Fully charged batteries are put on the locomotives at 4 p.m. and these batteries last the whole of the afternoon shift, as operations on this shift are mainly confined to loading the empty cars and hauling timber. The locomotives return to the round house after midnight, and all four batteries are charged to capacity ready for the following morning. No coal is loaded on the night shifts. If each locomotive shift was a full capacity one, the haulage units installed (two locomotives and four batteries) are capable of at least 3,200 ton-miles per day, of 16 operating hours, or, say, 80,000 ton-miles per month of 25 working days. The following are actual costs of operation for October. The mines worked only 22½ days out of a possible 26. The total output was 32,000 tons.

Storage-Battery Locomotive Haulage

Total tons mined and hauled during the month October, 1919	32,000 tons.
Days haulage worked	22.5 days
Average distance hauled (say 1¼ miles)	6,750 feet.

Monthly Expenses

Wages:		
4 Motormen, 22.5 days at $3.79	$341.10	
4 Conductors, 22.5 days at $3.47	312.30	
1 Tracklayer, 12 days at $4.08	48.96	
1 Helper, 12 days at $3.47	41.64	
1 Battery charger, 22.5 days at $3.66	82.35	
1 Asst. electrician, 12 days at $3.66	43.92	
Total wages		$870.27
Materials:		
Electricity, 3,825 k.w. at 4½c.	$172.12	
Motor repair material	25.00	
Dry sand and oil	73.00	
Repairs to mine cars, etc.	520.00	
		$790.12
Plant and Equipment:		
2 5-ton storage-battery locos, with two spare Edison batteries	$31,000	
Switch board	2,000	
Motor shed	2,000	
Spare parts	2,000	
500 Mine cars, etc., at $80.00	40,000	
	$77,000	
Depreciation at 11% on $77,000 for one month		705.83
Total monthly expenses		$2,366.22
Total ton-miles		40,000
Total cost per ton-mile		$0.059

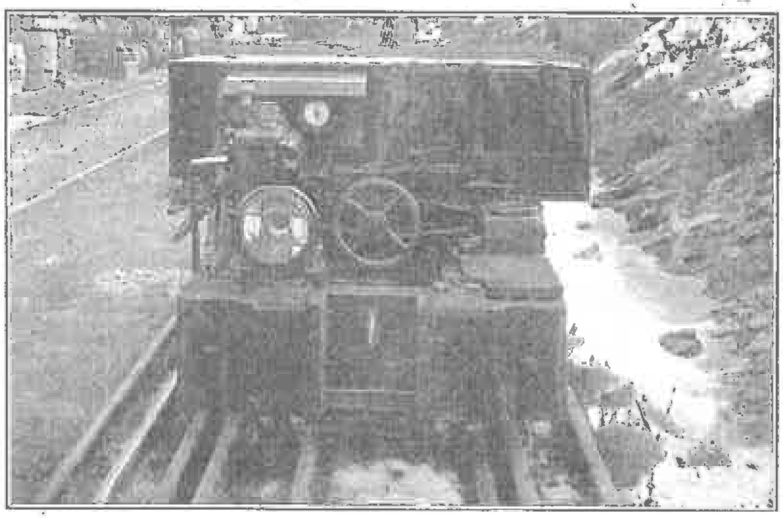


Fig. 1.—Storage-battery locomotive, Brazeau Collieries, Ltd.

DESCRIPTION OF EQUIPMENT

Locomotives:

Weight (including battery).........	5 tons (approx.)
Rated draw-bar pull on level track..	2,000 lb.
Speed at rated D.B.P.............	3½ M.P.H.
Gauge.........................	30 inches.
Over-all length (not including couplers)	12 ft.
Over-all width (over battery).......	58 inches.
Height over battery compartment from rail....................	42 "
Wheel base.....................	50 "
Wheel dia......................	20 "

Motors:—(Totally enclosed and air tight) 10 H.P., 2 G.E. 825 special series wound, casing guaranteed to withstand an internal pressure of 150 lb. per square inch.

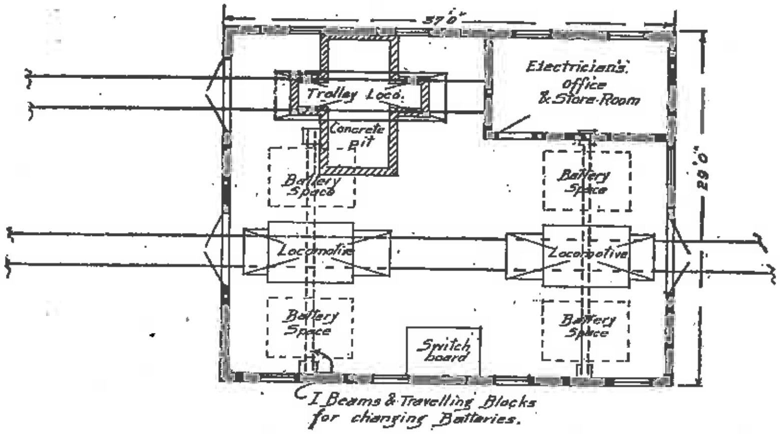
Controllers:—Type S.-56 A. (Airtight.) This type of controller does away with wasteful and dangerous resistances.

On the first step of the controller, the motors and fields are all connected in series and the batteries in parallel. There are two sets of batteries in each box, each set consisting of 42 type G-14 Edison cells. This means that the motors will have only 25 volts approximately across the terminals of each motor, which of course starts the motor slowly without sparking on the commutator. The next step of the controller connects the motors and fields in series and the batteries in series, which puts approximately half of the voltage (about 50 volts) across the terminals of each motor; this further increases the speed of the locomotive. The third step of the controller puts one motor and its field in parallel with the other motor and its field and the batteries in series which still further increases the speed of the locomotive. The fourth step of the controller connects the motors in parallel with the half of each of their fields in parallel, which gives the maximum speed, and in this case each motor has approximately the full voltage (say 100 volts) across its terminals.

Batteries:—The batteries are built up of Edison cells, type G-14 having a capacity of 375 amperes. The cells are grouped in trays, each tray containing six cells. This facilitates removal. Seven trays are connected in series, making a complete battery of 42 cells; each steel battery box contains two series—connected batteries of 42 cells each, which when connected in series through the controller gives a battery of 84 cells, having a rated discharge capacity of 350 ampere-hours. Each steel battery box has attached to it a steel, airtight box containing cartridge fuses and a cut-out switch. There is also attached to the battery box an ampere-hour meter for recording the amount of charge and discharge of the battery. This instrument is of great assistance in the care of the battery and in recording the consumption of electricity per ton-mile hauled.

Signals:—Incandescent (non-glare) head lights are mounted on each end of the locomotive; a foot gong is placed on the driver's platform.

Housing and Charging Arrangement:—The round house is steam heated by coils, and pipes lead parallel to the rails to thaw



LOCOMOTIVE ROUNDHOUSE

FIG. 2.

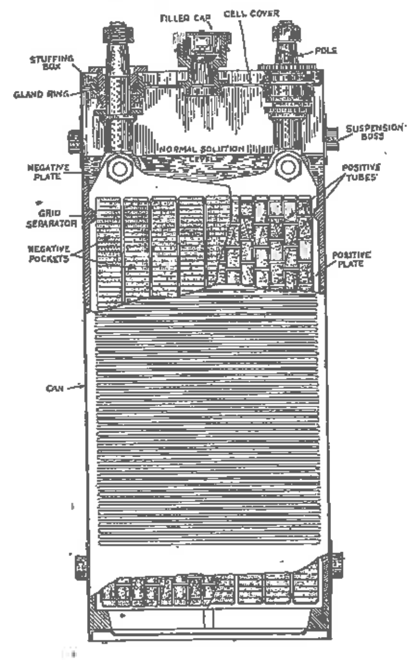
out thoroughly all bearings, etc. (See Fig. 2.) The batteries are lifted by overhead sets of travelling lifting blocks. It requires about five minutes to change batteries.

Charging Arrangement:—The batteries can be charged in series or parallel from a switch board. The board is supplied with electricity at from 250 to 300 volts, and the same motor generator is used which supplies the Jeffrey outside locomotive with power. Single batteries can be charged by using a regulated resistance, either at normal charging rate or by giving a 'boost' charge of four times normal rate for 10 minutes or three times normal rate for 20 minutes. To date, we have had little or no occasion to give 'boost' charges; we use the constant potential or tapering method of charging.

CONCLUSIONS

The writer, from experience gained with this equipment, considers that storage-battery locomotive haulage will be the underground haulage of the future. With ordinary precautions the locomotives are absolutely safe. They should of course in gaseous mines be confined to operation in intake air, and perhaps it would be a wise precaution in gaseous mines not to allow them nearer a working face than, say, 300 feet.

The Edison battery is all that is claimed for it. The cells are very strong in construction; they will stand overcharging and undercharging and almost any kind of abuse without being harmed; in fact they are almost 'fool proof' and require only a reasonable attention. The temperature of the electrolite (sodium hydrate) may rise to 115° F. without doing any harm, and sulphation and other troubles are impossible. The battery also may be left idle, either charged or discharged, for long periods without injury. The positive plate is made of steel tubes and the negative plate of steel pockets; the positive tubes contain metallic nickel and nickel hydrate in layers, and the negative pockets contain iron oxide. The cells are carefully encased in a steel container with an alkaline solution or electrolite. (See Fig. 3.)



EDISON CELL IN SECTION

FIG. 3

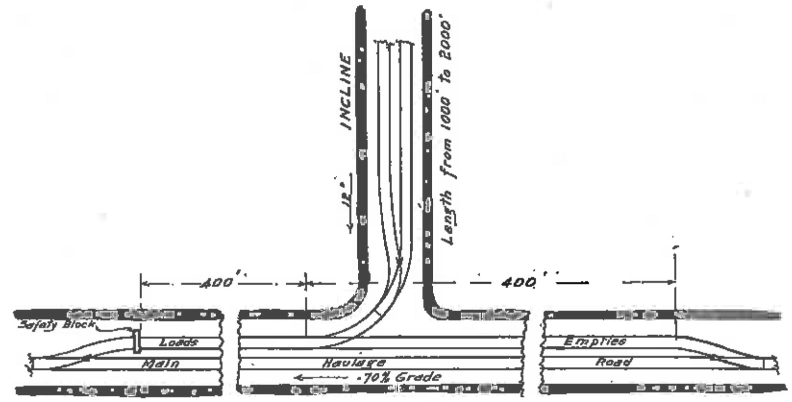
All that is necessary to maintain the battery for twelve months is to add pure distilled water to the specified height in the cells. The normal strength of the sodium hydrate solution is about 1.170 as measured by the hydrometer, but it may, at times, be as high as 1.190. When the specific gravity of the solution, tested after a full charge, drops to less than 1.130, it is time to refill with fresh solution. Our cells are type G-14, having 84 cells to a locomotive battery; this maximum voltage required for charging is 155 volts, and the normal charging rate 105 amperes. The charge is made according to the discharge of the battery; the full charging time for fully discharged battery is 4¾ hours. We have not yet had to make any renewals or repairs to the batteries.

The writer has had no experience with the oxide battery but the Brazeau Collieries' electrician has had considerable experience with that type, and he is convinced that the Edison battery is well worth the extra cost. At the same time, he admits that the oxide battery is capable of giving a better performance in the way of uniform speed and ability to discharge at high rates, but sulphation of the plates, lack of durability, and erosion outweigh, in his opinion the output advantages.

Locomotive Wheels:—A so-called expert in the east over-ruled the writer's specification for steel-tired wheels, and chilled iron wheels were sent with the locomotives. They lasted five months, developed 'flats' and nearly shook the locomotive and batteries to pieces before new wheels could be obtained to replace them. The writer advises intending purchasers, therefore, to insist on steel-tired wheels; they are easily re-tired and have a better adhesive effect on the rails.

Mine Car Lubrication:—The ampere-hour meters on these locomotives gave the writer some interesting information respecting car lubrication. A locomotive that is hauling the same number of cars over the same grade and distance every day should consume the same power, other conditions being equal. Common black car oil was found to be effective as a lubricant (on Hadfield wheels) only for two days after oiling,

whereas Numidian grease kept the cars lubricated at all temperatures for from seven to ten days. Therefore, the writer advises the use of good oil on mine cars—it pays. In fact, common black car oil, as it has been manufactured lately, can hardly be called a lubricant.



INCLINE LANDINGS & MAIN LEVELS.

FIG. 4.

Fig. 4 shows the Brazeau Collieries' system of sidings at the foot of the gravity inclines. The locomotives fill up the empty sidings on the in-trip, and haul out the standing loads on the out-trip by means of a short chain, without actually stopping. The size of the trains hauled by locomotives is from 65 to 75 cars, car tare 1,100 lb.. coal capacity of each car 3,000 lb.

The actual work of which the equipment described is easily capable in a month of 25 working days with motors working nearly at their capacity for 16 hours per day, is 80,000 ton-miles. The computed cost per ton-mile for 80,000 ton-miles in a month of 25 working days would be:—

Labour	Supplies	Depreciation	Ton-Miles	Cost per Ton-mile
$0.0119	$0.0110	$0.0088	80,000	$0.0317

Table of Comparisons

Cost per ton-mile hauled—from actual operations.

	Labour	Supplies	Deprecia-tion	Ton-miles	Total cost per ton-mile
Horses—Blairmore......	$0.072	$0.022	$0.015	14,450	$0.109
Compressed air.........	0.046	0.017	0.015	70,260	0.078
Horses—Brazeau.........	0.062	0.016	0.016	21,160	0.093
Storage-battery loco-motives—Brazeau:					
October, 1919..........	0.0217	0.0197	0.0176	32,000	0.059
January, 1920..........	0.0198	0.0183	0.0163	43,125	0.054

Note:—The figures given for the haulage costs with storage-battery locomotive are conservative, as there is about 1,200 tons of rock (cap rock and brushing) hauled to the outside per month and this is not included in the ton-mileage in the calculations.

HOISTING WATER BY TANKS.

BY T. MORDY

In these days of complicated machinery, and highly developed and scientific methods of handling modern problems, it seems strange that we should consider even for a moment any of the old-fashioned methods with a view to adapting and applying them as possible solutions to present day problems. Yet, hoisting water by tanks is merely a development, on a large scale, of the old-fashioned water bucket and windlass, and has proved its value as a means of handling large quantities of water under conditions where pumps are either too expensive to maintain or too apt to breakdown to be relied upon entirely.

In the north of England and in Belgium there are large and heavily watered areas drained by shafts, specially sunk for that purpose. These are connected by drainage levels with the mines of the subscribing companies, and are equipped with large and elaborate hoisting plants entirely automatic in their operation, requiring only a regular visit once a day from an inspecting engineer to examine and oil the machinery. These plants have been designed and installed to handle huge quantities of water at low cost of maintenance and a minimum of breakdowns. This purpose they successfully achieve.

At No. 6 mine of the Canadian Collieries (Dunsmuir) Ltd, Comox Mines, the problem is of a somewhat different nature. This mine is not hoisting coal, but it is connected through the lower seam with No. 5 Mine where coal-hoisting is in progress all day until 11 p.m., while material is lowered throughout the greater part of the night. Under these circumstances it is in every way much more convenient to use No. 6 shaft as a water shaft. The appended analysis of the water it is required to hoist will explain why pumps have been discarded here.

Analysis of Water at No. 5 and No. 6 Mines, Comox, Canadian Collieries (D), *Ltd.*

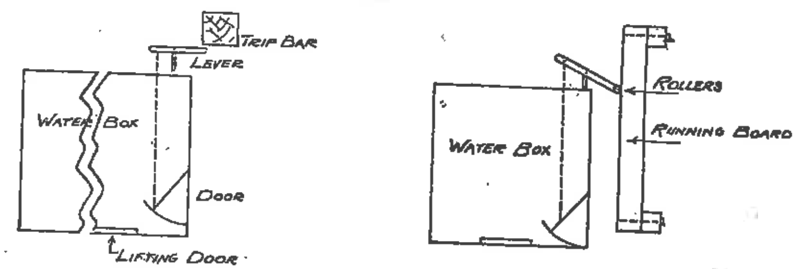
	In the Shaft.	In No. 2 Dip.
Colour	Yellowish Brown.	Yellowish brown.
Appearance	Turbid with heavy yellowish brown sediment.	Turbid with heavy yellowish brown sediment.
Odor	Earthy.	Earthy.

	In the Shaft. Per cent.	In No. 2 Dip. Per cent.
Solids (by evaporation)	.3096	.4484
Mineral matter (non-volatile)	.2151	.2727
Organic, volatile (ignition loss)	.0945	.1757
Sulphates (as CO_3)	.14857	.15088
Chlorides (as Cl)	.0120	.0102
Silica (as SiO_2)	.0046	.0035
Iron (as ferric sulphate)	.13187	.18913
Lime (as calcium sulphate)	.0949	.0733
Magnesia (as magnesium sulphate)	.0273	.02717
Acidity (as sulphuric acid)	.19677	.18292

The corrosive action of the water as practically demonstrated on the pumps at No. 5 shaft is sufficient to eat away a suction pipe and foul the valves to such a degree in a few days as to put the pump out of commission, and better results have been obtained by keeping these pumps in good repair to be used as reserves in case of breakdowns or during repairs to the hoisting plant at No. 6. This practice has the further advantage of providing a capacity reserve against any extraordinary influx of water with which the hoisting plant at any time is unable to cope.

Two years ago, a wooden tank, having a holding capacity of about 300 gallons sufficed to hoist all the water it was required to lift at that time. This box or tank was hung under the cage by four heavy chains, each 25 feet long, and was balanced by a coal-car half filled with scrap iron, placed in the other cage. The 25-ft. extension below the cage reached the water below the hoisting level and was delivered 25 feet below the tipple level into a spillway. The extension of pillar work in No. 5 mine, however, gradually tapped a larger and larger water-bearing area, and in consequence, the balance box was displaced by a second water box, the capacity of each being increased to 400 gallons, and subsequently to 485 gallons, giving a net discharge of about 350 gallons per box.

Up to eighteen months ago this proved a fairly satisfactory 'lay-out,' but the quantity of water continued to increase, and so means had to be devised to cope with it. The boxes were emptied through front valves lifted by a chain attached to a lever, which caught against a trip-bar in the old-fashioned way. This was displaced by a lever with rollers on the end, and the bar supplemented by a vertical bar, 10 ft. long, up which the rollers could travel. This enabled the box to travel up 10 feet and return the same distance with the valve open so that it discharged uninterruptedly. In this way the hoistman did not have to 'come up' slowly to stop within a few inches of his mark; he could 'come up' at good speed, and start back again directly he had stopped, the bottom tank filling and the top one emptying continuously. This saved time, and relieved the motor of a large portion of its



dead lift. The increase in the number of hoists was immediately noticeable at the heaviest stage of the water, as many as 2,700 hoists in 24 hours at an average of 350 gallons per hoist, making a total in one day of 945,000 gallons of water hoisted.

A reading of the bachograph for one average day showed 21¼ hours actual hoisting, 2,361 hoists made, 911,346 gallons of water hoisted, 3,000 kilowatt hours used, and 303.8 gallons hoisted per K.W.H.

This, however, proved in the long run, more than the motor could handle without injury. It is a 250 h.p. motor, running at 480 revs. per min., 58 amperes per phase, 3-phase system, hoisting from a depth of 276 feet. The shallow depth gave it no chance to reach its full speed, and the heavy current induced by the overload, heated the coils with the result that they frequently burnt out.

An alteration in the gearing and drums of the engine increased the power by one-third, but this was partly offset by substituting circular tanks (4 ft. 6 in. dia. by 6 ft. 6 in. deep, having a capacity of 608 gallons) for the old wooden boxes. The change did, however, enable the motor to gain speed sooner, and the number of hoists remained as before, two per minute.

The weight of tank........ = 6080 lb.
" " " unbalanced 1000 " (approx. due to tank filling slowly and loss of weight in water.)
Total load on the motor 7080 lb.
Speed of motor........ 720 ft. per min.
Motor working two-thirds power for duration of wind:

$$\times \frac{7080 \times 720 \times 3}{33{,}000 \times 2} \times \frac{100}{70} \text{ (70\% efficiency)}$$

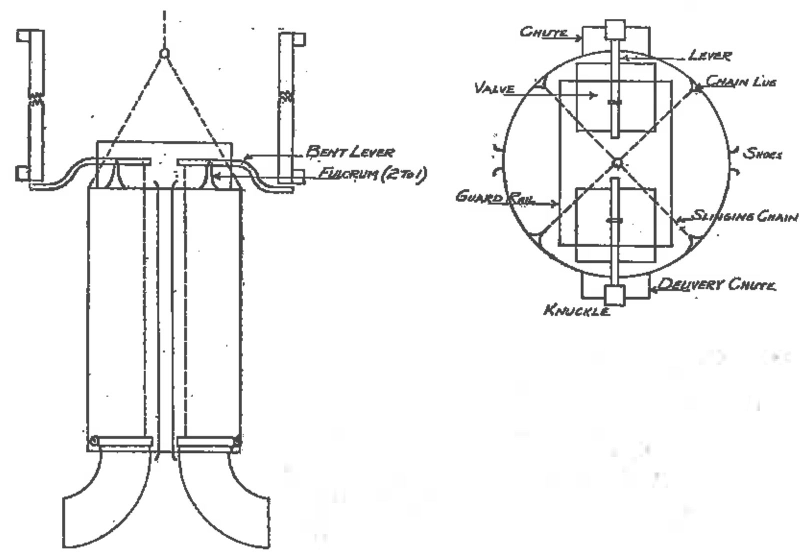
$\times$ 330 H.P., when unbalanced and
284 H.P., when balanced.

A test of the power cables revealed an excess amperage that varied from 38% to 71½% above the manufacturer's specification, and thus developing an actual horse-power of 394. At present the tanks operate twice a minute, delivering 600 gallons per tank through a distance of 276 feet. They are provided with two levers each to equalize the striking on the trip board, and the rollers are displaced by a knuckle which doubles on the return journey, thus permitting of an easy clearance for the lever. The leverage is two to one, the lever being bent to give an additional throw clear of the tank side. A guard rail is put over the levers to prevent slack chain from getting entangled in them.

These iron tanks, ropes, chains, etc., are, of course, subject to constant corrosion, which necessitates regular and frequent rope oiling, re-coning of rope, and repair of tanks. To facilitate

the latter, an overhead trolley-way has been installed, which permits of either tank being readily picked up, slung out through a portion of the guides hinged for that purpose, and lowered into a small car underneath where a spare tank is in position to be picked up and slung into place. The wasting effect on the rivets and bolts is particularly marked, the threads on both nuts and bolts being 'gutted' clean out, and although this is partly prevented by heavy painting with superlastic preservative paint, yet the tanks still require changing and repainting every two or three weeks. As, however, the mine is only used for water hoisting, no work is stopped thereby, and on such occasions the shaft pumps at No. 5 mine come into play and justify their existence.

There is, of course, a certain amount of unavoidable leakage and drip from the tanks that tends to corrode the ropes, but liberal applications of boiling rope-dressing every two weeks does much to prolong their life.



DISCUSSION

MR. E. B. WILSON (*Communication to the Secretary*): Mr. T. Mordy's paper on hoisting water at the Comox mines of the Canadian Collieries Ltd. covers not one but three interesting problems which have a bearing on the subject—mechanical, chemical, and economical.

As auxiliaries to pumping plants, water tanks have time and again demonstrated their value. We have seen them recover pumps in drowned mines, and again have seen them hold the water which was gaining on the pumps. We have also known them to hold the water until broken-down pumps could be repaired. Those managers who have tested their usefulness are slow to 'scrap' these old friends; consequently one may find them occasionally at mines where more modern devices are also installed.

Several large water-bucket hoisting plants have been installed in the anthracite fields of Pennsylvania, and have, since the introduction of the multi-stage centrifugal pump, been held in reserve as auxiliaries to the pumps, which is the reverse of the plan followed at the Comox mines. The Keyser Valley water hoisting plant at Scranton was the last word in this sort of installation, it having been electrically equipped so that it hoisted, discharged, and lowered the bucket without the aid of an attendant, although a watchman was in the engine house to oil the machinery. The object of this plant was to do away with several pumps at mines in the vicinity and thus economise in pump repairs, attendance, and power. The object was attained, but it was discovered later than even this plan could be improved by making use of the centrifugal pump.

Economic Considerations.—A simple calculation will explain the economy in power that would take place at the Comox mine were it possible to make use of a centrifugal pump instead of the water-buckets. At this mine the shaft is 276 ft. deep—the tanks which run twice a minute deliver 600 gallons each and the efficiency of the apparatus is stated to be 70%. According to Mr. Mordy, the tanks, when hoisting in balance, required

284 h.p., and if it is assumed that the power costs 3 cents per K.W.H. it would amount to $8.52.

Assume now that a multi-stage centrifugal pump had to do the work, it having the same efficiency.

$$\frac{600 \text{ x } 10 \text{ x } 2 \text{ x } 276 \text{ x } 100}{33{,}000 \text{ x } 70} = 143.4 \text{ h.p. or practically one-half}$$

the power needed for the tanks and with a power cost of $4.30 per K.W.H. The decrease in cost is sufficiently large to attract attention.

The Chemical Problem.—While the water at the Scranton plant may not be as corrosive as that at the Comox mines, it is, nevertheless, sufficiently corrosive to make it advisable to use pumps of phosphor bronze and, as well, suction pipes of the same material. With this equipment the corrosive action of the water is comparatively unimportant.

It would seem from the analysis of the water from the Comox mines, that its corrosive action might in a great measure be reduced. It is evident that ferric sulphate will not corrode the pipes directly, and that some chemical action occurs to furnish free sulphuric acid for the purpose. If the chlorine is in the form of magnesium chloride the following reaction would probably occur:

$$3MgCl_2 + Fe_2(So_4)_3 = Fe_2Cl_6 + 3MgSO_4$$

If free sulphuric acid is present, hydrochloric acid would be formed, according to the reaction.

$$Fe_2Cl_6 + 3H_2SO_4 = 6HCl + Fe_2(SO_4)_3.$$

Hydrochloric acid has more affinity for iron than sulphuric acid, but in this case it is the H_2SO_4 that produces HCl. Further, the secondary combination of iron and chlorine will (if more sulphuric acid be present) form more hydrochloric acid, thus working the chlorine extra shifts to accomplish the destruction of the pipe line. The problem of discouraging corrosion is a chemical one which involves the neutralization of the H_2SO_4.

About two years ago a similar problem was presented to the writer which he solved by converting the sulphuric acid into calcium sulphate. He is not prepared to state that this is applicable to every case, although it is possible that the condition and position of the mine may at least warrant its trial.

It is evident from the large amount of solid matter in the water at the Comox mines that its mixture with sulphate of lime would produce a material that would fill up the column pipe unless the velocity of the water was such that it would not precipitate on the pipe. In the anthracite mines, such material has almost closed up column pipes in two or three weeks. Therefore, the remedy, while simple theoretically, may meet with a set-back in its practical application, for which reason it cannot be considered as of universal use.

THE SUGGESTED APPLICATION OF HYDRAULIC STOWING TO UNDERSEA COAL WORKINGS, WITH SPECIAL REFERENCE TO THE SYDNEY COALFIELD

BY WALTER HERD

Annual Meeting, Mining Society of Nova Scotia, Glace Bay, May, 1920.

Although the hydraulic stowing of mine workings has long since passed the experimental stage and is to-day used with success in many European coal mines and in South African and Australian gold mines, yet, with the exception of a few American thick-seam mines, the English speaking countries generally have been very slow to adopt what has proved to be the best method of filling the space left by the extraction of a coal seam so as to cause the minimum of subsidence.

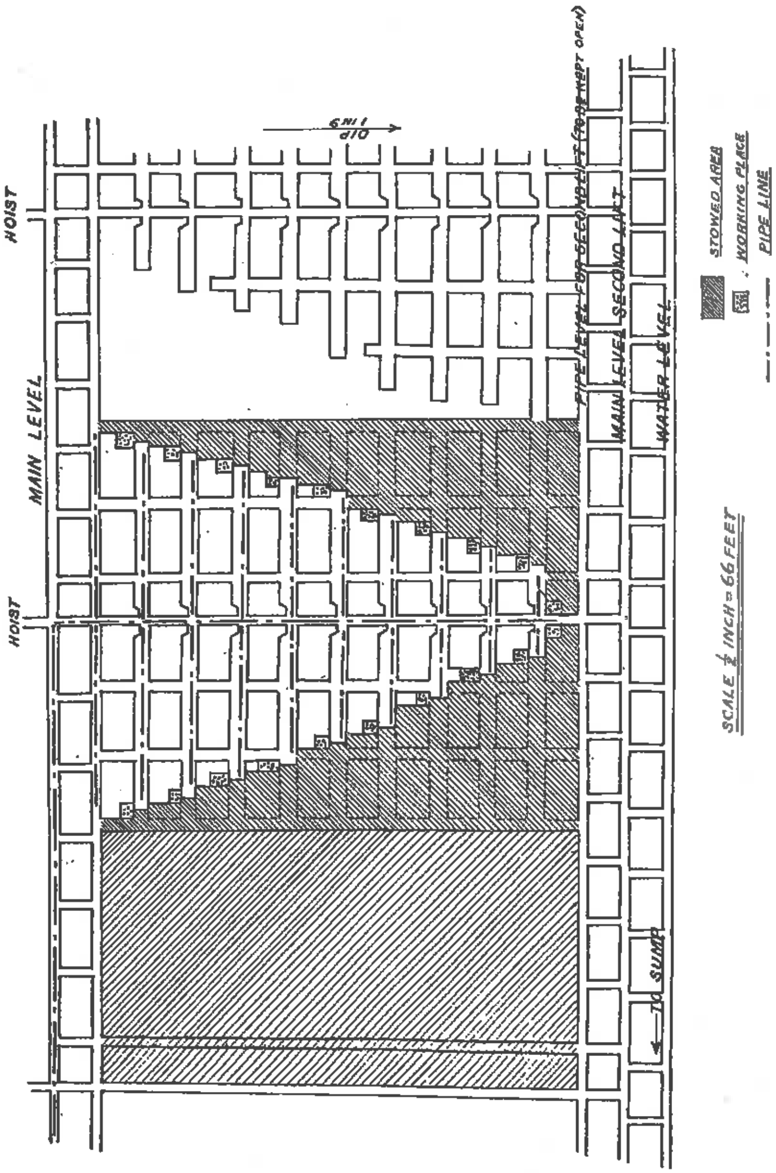
Hydraulic stowing was first attempted in Pennsylvania, but to Upper Silesia belongs the credit of first having demonstrated its practicability on an economic basis. Prior to its adoption in that coalfield about 20 years ago, seams of from 20 to 40 feet in thickness were being worked; this caused great surface damage and there was a large loss of coal through the difficulty of taking out the pillars in these thick seams. At the same time spontaneous combustion added to their troubles, often necessitating building off large areas of coal. Since the adoption of hydraulic stowing, practically the whole of the coal is extracted with the minimum of surface damage, and gob fires are almost unknown. Very much less timber is required and accidents have been considerably reduced.

Although these conditions do not exist in the Sydney coalfield, there is the condition of large areas of coal lying under the sea at comparatively shallow depths where it would be imprudent to extract the whole of the seam. To be more definite, this applies to seams lying under the sea and having a cover of from 200 feet to 800 feet of solid measures. Already, between these depths, the greater portions of the best seams in the coalfield have been formed into pillars, representing at least 50% of the seam left to support the roof. No doubt in the past, when considerable areas of these seams remained to be worked on the land area, the loss in leaving these pillars did not seem so apparent as it does to-day, when conservation commissions are bringing home to most of us the necessity of husbanding our natural resources. This should apply particularly to coal, which is a wasting asset. The Sydney coalfield undoubtedly contains a large tonnage, but it has not the illimitable resources popularly supposed. The workings in the thicker and best seams extended for a considerable distance seawards, and the necessity of conserving coal suitable for metallurgical purposes is very apparent.

By hydraulic stowing, it ought to be possible to recover the many millions of tons of coal left in pillars having from 200 to 800 feet of cover. The cost of recovering all these pillars now, after the lapse of many years since they were formed, would in some cases be prohibitive, due to the roof in the rooms having fallen and the difficulty in collecting the stowage water. Many will have to be left until coal has a greater value than it has to-day, but there are many pillars that could be economically recovered at the present time. However, the plea the writer wishes to put forward is not so much for the recovery of pillars that have been formed in the past, as the need for guarding against a repetition of the same procedure in the future.

In the future working of seams underlying or overlying those already formed into pillars and which extend under the sea, the writer would suggest that the area in these seams, down to 800 feet of cover below the sea bottom, be blocked off into panels of suitable size, which in the case of the thicker seams would be formed into pillars to be extracted as soon as the broken work in the panel is completed, the pillar coal pulled up hill to the top level and the space left stowed by hydraulic means, as shown in Figure 1. In the case of thinner seams, the panel would be worked out by retreating longwall, the space left being filled as in the case of the pillar extraction in the thicker seam.

— FIG. 1 —



STOWING MATERIAL

Generally it may be stated that the economic success of hydraulic stowing depends upon the existence of suitable stowing material near at hand; to a lesser extent the distance the material has to be transported underground and the head against which the return water has to be pumped must be considered. Of all the substances tried as a stowing material, sand has proved to be the best, both from the point of view of cost and for forming a densely packed stowed area with the minimum of shrinkage. Less water is required to flush sand than other substances tried and less material is held in suspension in the return water, thus reducing the cost of renewal to pump parts and pipeline. Experiments in recent years have shown that in some cases a 10% mixture of clay with sand produces a better filling material than sand alone. This experience, not being general, may possibly be explained by some of the clays tried being more or less of a cement nature, which would bind together the particles of sand after the water had run off. Where sand is not procurable, pit-refuse heaps, boiler ashes, and granulated blast-furnace slag have been used with success. The last named material is, however, very hard on pipes and is of too porous a nature to be used alone. In one large European installation special quarries have been opened to supply stowing material, the whole of the stone being crushed before being sent to the mines. As much as 4,000 tons of stone per day has been sent from these quarries to the various mines that are supplied by them.

SUPPLY OF STOWING MATERIAL

The Sydney coalfield is fortunate in having within a reasonable distance of the mines an adequate supply of sand. This sand could be extracted by means of suction dredges, which should work backwards and forwards across the various beaches and sandbars in the vicinity of the mines. The sand would be delivered from the dredges to railway cars for transport to the mines, or, where the sand has only to be taken a short distance, an aerial ropeway would probably be the cheaper form

of transportation. One hundred and fifty tons per hour can be economically carried by this means.

Where sand is employed as the stowing material it should be possible to extract the entire seam up to the present legal limit of 180 feet of solid cover between the seam and the bottom with practically no risk of the sea's breaking in. It is not possible entirely to replace the original seam with stowing material as there will always be a certain amount of shrinkage consequent upon the drying of the stowage, but at least 90% and possibly 95% of the excavated space is filled. It is a generally conceded fact that the better an excavated area is stowed the further is the line of fracture thrown ahead, making, in the case of 95% stowing, a line that forms an angle of about 60 degrees with the vertical in a seam dipping one in nine. Consequently, an area that is not stowed at all will make an almost vertical break to the surface. In other words, the better the stowing the further is the extension of the 'draw,' with, of course, proportionately less subsidence. Consequently, the risk of the sea's entering through a break with a line very little removed from the horizontal is much less than through an almost vertical break.

DESCRIPTION OF PLANT

A short description of the procedure in hydraulic stowing may be of interest to those who have not seen it in use. The stowing material, whatever its composition may be, is conveyed from a storage-bin by a spiral conveyor into a hopper in the shape of an inverted cone about three feet in diameter at the top. The reason for using the spiral conveyor is that the quantity of material to be delivered can be accurately gauged to suit the water supply. The storage material is met at the bottom of the hopper with several jets of water and a little lower down the main jet enters, ensuring a thorough mixture of the material with water. This water flushes the material down a pipe to which the hopper is attached. This pipe may either be placed in a shaft or slope from the bottom of which it is continued into the workings, or it may be placed in a borehole sunk close to the sea-shore and as near as possible to the workings it is desired to stow. A simple arrangement is fitted on the hopper that stops the supply of stowing material should the water supply fail; otherwise the pipe would fill with dry material, which would soon choke it up. Branches are put in from the pipe to the various areas to be stowed and blank flanges instead of valves are used to divert the stowage to the required area.

UNDERGROUND LAYOUT

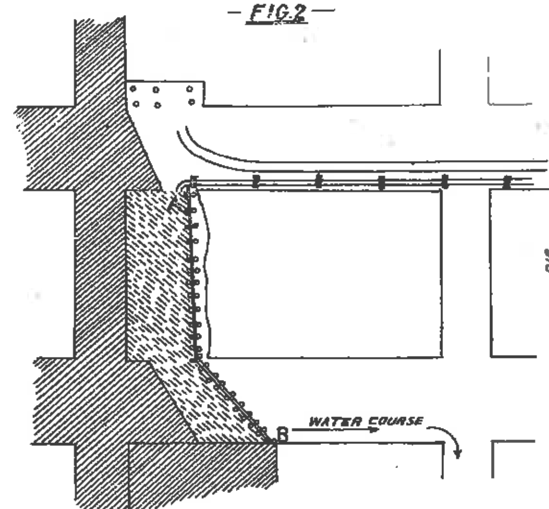
In Figure 1 the writer has endeavoured to show a standard panel formed into pillars in the usual way. These pillars are half-out and the sketch shows the panel at its maximum production. The next panel inbye is being split into pillars, the lower one being ready for extraction to replace the almost extracted lower pillar in the outside panel. These panels are designed to give a maximum output of 200 tons per day in a six-foot seam, and an average output of 150 tons per day. The tonnage of sand required per panel would, consequent on its having a greater specific gravity than coal, be a maximum of 300 tons and an average of 225 tons per day to replace the coal extracted in pillars, but as the pillars only represent about half of the coal originally in the panel area the quantity of sand required daily will, on the average, be double these figures. An 8-inch pipe will flush 120 tons of sand per hour under the conditions existing in this coalfield, so that this size of pipe would be sufficient to stow the output from two panels in one shift.

It will be observed that the stowing-pipe is taken into the main level and down the headway, branches being put in to each room. The pillars are extracted by a series of upward slices about 15 feet wide, and when a cut is through it is immediately stowed. When a cut runs between the stowing and an old crosscut the latter should be stowed before the cut is begun.

The depth of the panel, viz., 700 feet between main levels, may seem excessive in comparison to the width of 500 feet, but the reduction of the narrow work to the minimum has been

kept in view, also the fact that after a cut is through and has been stowed it will be two days before the next cut can be commenced, as the stowage will require about that length of time to dry out, and a few extra places must be maintained to give employment to the men in these circumstances. It will be noticed that, after the first lift, three levels must be driven for future lifts, viz., a water-level, main haulage-level, and pipe-level. The latter level is necessary to take out the pillar to the rise of the main level; otherwise the stowing would have to be forced uphill. Under certain conditions of good roof this level might be driven room width. For ventilation, and also as a waterway, it is necessary to keep open the main headway after the pillars have been extracted. It could be stowed as the pillars are extracted to a minimum width of five feet and left that way.

— FIG. 2 —



— SCALE 1 INCH = 25 FEET —

AREA PREVIOUSLY STOWED

AREA TO BE STOWED

Figure 2 shows an enlarged view of a cut-through about to be stowed. The stowing material and water pour into the excavated area at the point 'A.' From there to the point 'B,' two parallel rows of props are set about two inches apart with three feet between each couple. One-inch boards are placed between the two-inch space forming a wall from roof to pavement, and this wall is backed with brattice-cloth. As the stowing material and water flow into the space, the solid material gradually settles and the water drains off through the brattice-cloth and spaces in the boards, leaving, after a day or two, a hard compact filling, through which roadways can be driven. After a few days the props, boards and brattice are withdrawn and are ready for further use. The water will run through the various crosscuts until it reaches the water-level and finally the sump, from which it is pumped back to the surface. In the case of using sand very little trouble is experienced in clarifying the water. All that is necessarv is to lead it into the bottom of a wooden tank and allow it to overflow into the sump. The tank can be cleaned out after flushing ceases. Where material of the nature of clay is used for stowing, a series of silt-recovery boxes would be required.

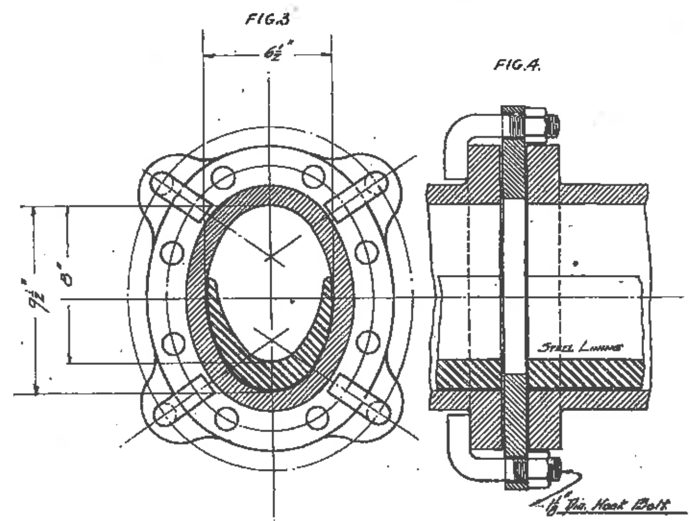
It will be noticed that the stowing is not flushed tight against the next cut. If this were done it would not be possible to drain the water. At the same time the leaving of a space ensures good ventilation and an open end for the next cut. Moreover, the coal does not become mixed with the stowing material.

All stowing should be done on the night shift, and telephonic communication should be established between the surface and the men in charge of stowing below ground. Before the commencement, and after the completion, of stowing operations, water only should be run through the pipes for a few minutes to make sure that there are no obstructions.

WATER

The quantity of water required varies with the stowing material used. In the case of sand, volume for volume is sufficient; that is, from six to seven gallons of water will flush a cubic foot of dry sand into the workings. Twice this quantity

of water would be needed if material of a clay-like nature was used. This quantity of water is based on the assumption that no stowing material has to be forced into the workings to the rise, as the water required increases very rapidly when a head is put against the stowing material.



PIPES

The choice of the type of pipe through which to flush stowing material is a very important matter. In the early days of hydraulic stowing the cost of renewing pipes was serious; plain cast-iron pipes were used and were given a periodic quarter-turn to ensure even wear. Owing to the scouring action of the stowing material, many wore out completely before 100,000 tons was passed through them, and this to some extent accounted for the scepticism with which hydraulic stowing was regarded for a number of years. Attention was then given to various forms of lining such as earthenware, porcelain and wood, all of which proved comparatively successful. The type of pipe that of late has seemed to meet with the most general approval is the oval-shaped steel or cast-iron pipe fitted with a tapered steel or cast-iron lining as shown in section in Figure 3.

The diameter of the long axis is about eight inches and the tapered lining when made of steel has a maximum thickness of one inch. These linings are made in from 3-ft. to 4-ft. lengths, and from 500,000 to 1,000,000 tons of stowing material can be passed through the pipe before the lining requires to be renewed. In order to prevent these linings slipping out in the event of replacing a broken pipe in a column set either in a shaft or in an incline, loose flanges are inserted between the fixed flanges as shown in Figure 4. These loose flanges have a space area equal to the internal diameter of pipe and lining combined. Special bolts keep this loose flange fixed to the flange of the pipe above so that when the lower pipe is removed the linings are held in place.

The greatest wear takes place in bends, and a simple and effective method of counteracting this wear is to have a series of ribs a few inches apart and at right angles to the flow as shown in Figure 5. The material collects between these ribs and protects the metal from the excessive water.

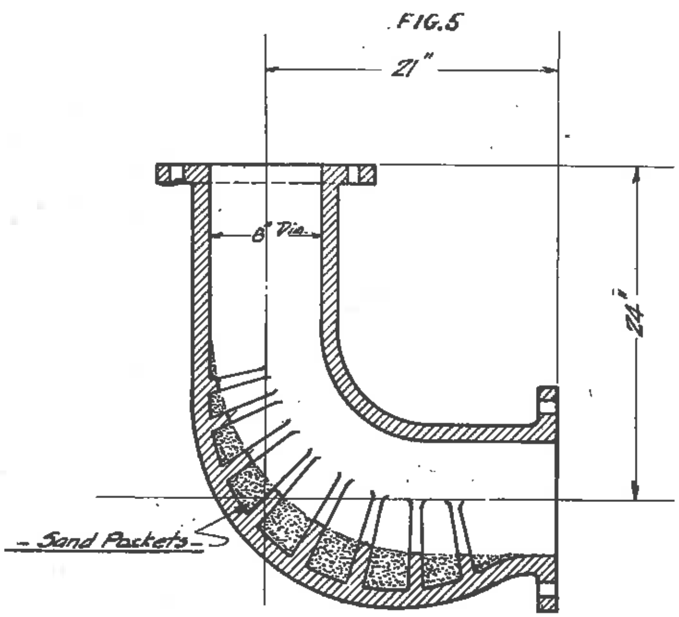


Figure 5.

COST

It is often argued against hydraulic stowing that it will put a prohibitive cost on the coal mined. In some cases this would be so. Below, the writer has attempted to give an approximate estimate, based on present-day prices of the cost of installing a plant in this district under average conditions; the additional cost per ton of coal produced is also computed. The estimate is based on one installation supplying two panels similar to those shown in Figure 1, each producing 150 tons of coal per day for 200 days in the year. The writer does not consider it economically possible to continue hydraulic stowing during the winter months in the Sydney coalfield. During this period the formation of panels could be undertaken and in the thicker seams the pillars formed.

COST OF AVERAGE INSTALLATION

Two lined boreholes 600 feet deep, 8 and 6 in. in diameter respectively	$ 7,000.00
2,000 feet of lined piping (laid) at $2.50 per foot	17,000.00
Storage-bin, hopper and water-supply pipe	10,000.00
Return-water pipe	3,000.00
Pump and motors complete	5,000 00
Total cost of installation	$42,000.00

This, reckoning interest and depreciation at the rate of 20% per annum, entails a yearly charge of $8,400.

Approximately one and a half tons of sand will be required to replace every ton of coal taken from the panels; but at the time of stowing, in the thicker seams where the panel is first split into pillars, approximately three tons of sand is required. In arriving at the cost, the total amount of coal in the panel is considered.

ESTIMATED COST OF STOWING TWO PANELS PRODUCING 300 TONS OF COAL PER DAY

450 tons of sand at 50 cents per ton =$225	=75 cents
Interest, and depreciation on plant	=14 "
Pumping	= 2 "
Labour	= 7 "
Total	98 cents

In the case of pillars already formed, the cost would be practically double this amount.

This estimated cost is no doubt higher than would actually be the cost in comparison with the coal won from the lower levels of the mine where hydraulic stowing was unnecessary, as, being nearer to the outlet, haulage charges and upkeep of roadways would be less, ventilation would be much simpler, and more timber could be recovered. Again, the pillars left are in the thicker and more profitable seams, and their extraction, using hydraulic stowing, might not be actually more costly than certain seams now being worked. Also it must be remembered that these pillars will constitute an extra tonnage to the mines as presently developed, making possible an increase in output that would further reduce overhead charges, for which credit must be given.

CONCLUSION

In conclusion the writer would suggest that some form of hydraulic stowing be adopted in the remaining seams to be worked under the sea down to 800 feet of cover. If this is not done, the seam could be left intact until a cover exceeding 800 feet is reached, with the exception of driving winning places through this area to reach the coal having a cover greater than 800 feet. Beyond this cover there ought to be no danger in extracting the whole of the seam without solid packing. A Crown Lease permits of the total extraction of a seam 11 feet thick after a cover of 810 feet is reached. It may, however, be necessary to carry hydraulic stowing beyond the 800-feet cover mark in the case of working superimposed seams simultaneously; in fact, hydraulic stowing would allow of this simultaneous extraction, an operation that without hydraulic stowing should be highly condemned, being the ruin of many mines.

The writer does not suggest that the introduction of hydraulic stowing into the Sydney coalfield is a simple and easy matter. There are many difficulties; but it well deserves the careful consideration and co-operation of those interested in the ultimate recovery of all the coal.

THE THEORY AND PRACTICE OF FIGHTING MINE FIRES FROM PRACTICAL EXPERIENCE GAINED IN PICTOU COUNTY.

By F. E. NOTEBAERT

Annual Meeting Mining Society of Nova Scotia, Glace Bay, May, 1920.

A large proportion of the Pictou county coalfield is composed of very thick seams. Amongst these are:

The Foord seam..........	40 ft. thick
The Cage Pit seam........	18 ft. thick
The Third seam..........	14 to 17 ft. thick
The McGregor............	The thickness of which at certain places reaches over 22 ft.

These four seams are known as the seams of the Stellarton district, and have been worked, some of them, for over a century. To these seams a series of thirteen new seams underlying the four mentioned, was added, when, in the winter of 1915, their discovery was made by means of a borehole, drilled by the Acadia Coal Company.

As in the upper seams, some of these new seams are of unusual thickness; thus there is one 21-ft. seam; one 28½-ft. seam; one 20-ft. 4-in. seam; one 24-ft. 2-in. seam; and one 23-ft. seam. The drilling also disclosed the existence of thinner seams, with thicknesses, for example, of respectively 3 ft. 6 in.; 5 ft.; 6 ft. 2 in., and 4 ft. 11 in.

Undoubtedly had the borehole been extended further down, other seams would have been discovered, and eventually it would have entered the Westville coal measures. These very often have been considered as being the same seams as those of the Stellarton district, although thrown in a southwesterly direction by the fault of great amplitude known as the 'McCulloch fault,' the existence of which, however, has never been proved. Indeed, it is fairly safe to abandon the old conception of the 'McCulloch fault' and to accept the theory that the Westville seams are the regular continuation of the series of Stellarton seams, thus adding an enormous quantity of coal to that already known to exist in the Stellarton district.

Having thus outlined what may now be properly called the main coalfield of Pictou county, the chief features of this field will next be enumerated.

The angle of dip may be called steep in comparison with the great majority of seams in the Cape Breton coal basin. At the southern end of the field, the seams outcrop with an angle of from 21° to 23°, dipping northerly until they reach a basin line; then they rise in a northerly direction at various inclinations, from the horizontal up to 90°; and in certain disturbed sections of the northern portion of this field the seams are completely reversed, the footwall resting on top and the hangingwall lying underneath.

The angle of the dip, the thickness of the seam and the fact that the space left open by the working out of the seam is not flushed or filled, imply almost necessarily a method of working by 'Bord and Pillars,' retreating from the limit of the field towards the main slope and leaving only a crush behind the working faces.

The immediate consequence of such a system is the unavoidable abandonment of some coal in the gob. This coal, being subjected to the heavy pressure of the roof, is crushed, and provided that the oxygen of the air is allowed to come in contact long enough with this 'loose coal,' great heat is bound to be generated and ultimately provoke a 'gob fire.'

These gob fires are naturally frequent in the seams of the Stellarton district, or more correctly, would be quite frequent if special precautions were not taken. These precautions comprise:

(1) Reducing the quantity of combustible matter left crushed in the gob, the presence of which is the original cause of the fire. (2) Reducing the prolonged contact of air (oxygen) and crushed coal in the gob. (3) Emergency measures, consisting of heavy stoppings which allow us to isolate and to seal off any sections or part of a section whenever this section is threatened or affected by fire.

These emergency measures are naturally very expensive, but our experience has proven to us that they are, after all, the cheapest and safest means of preventing or fighting gob fires.

The amount of combustible matter left in the gob, when attempting to work out such thick seams, can only be reduced to an unimportant quantity by the adoption of the 'flushing system,' also called 'hydraulic filling,' but at this stage of the coal industry in Nova Scotia, it is very questionable if in long slopes with an abnormally high cost of labour as compared with the selling price of the coal, the time has yet come when 'hydraulic gob-flushing' can be introduced into Nova Scotian mines with financial profit. However, as just stated in the case of the Pictou county mines, the 'flushing system' is, and will be, the only system by which (all the coal having been recovered), the gob fires may be completely suppressed. Incidentally, it might also be mentioned that with the introduction of hydraulic flushing, dust explosion will become a thing of the past.

To reduce the prolonged contact of air (oxygen) with the crushed coal left behind in the gob, the most efficient method is to advance the working faces as rapidly as possible, so as to bring down the roof; also to avoid any filtration of air through the gob. This can best be done by retreating towards the slope, also by ascensional ventilation, the air being allowed in at the bottom of the pillar section and exhausted at the upper end of the section in a direction opposite to the gob.

To reduce a prolonged contact of the air with the gob implies also that the gob resulting from the working out of a pillar section be properly and completely sealed off, so as to avoid, as much as possible, filtration of air through the gob and oxidation of the crushed coal left therein.

In the prevention of gob fires, without resorting to the 'flushing system,' the above in general are the points that should always be kept in mind in the extraction of pillars in a thick seam dipping at a steep angle.

The nature of the seams and of their dispositions has been mentioned, and it may be of interest to mention that the coal seams of Pictou county, and especially of the Stellarton district, are the least subject to spontaneous combustion; this is attributable to their very low percentage of sulphur, which in most cases does not exceed 1%.

But even when all precautions are taken, gob fires are apt to take place. Usually the first indication of fire is only a very slight odour of distillation of coal. According to the nature of the seam, and to the general dispositions that are causing the fire, the odour will persist for hours or for days, and cases are known where it has persisted for weeks without showing any increase in temperature or any sign of smoke, which are the next indications of a gob fire.

At this stage when smoke appears, conditions are always very serious, because even if the seam does not liberate explosive gases, the distillation of coal with a restricted amount of air will generate carbon monoxide, which is highly explosive. It is at this stage that good judgment must be used and a quick decision made.

When smoke has made its appearance, the tendency is usually to continue the dispositions that had been adopted in the earlier stages of the gob fire, when its only indication was the odour of distillation. In many cases this course may result in apparent success, but very often the fire has already been allowed to smoulder too long, heat has increased and flames have appeared. Such conditions are highly dangerous and alarming, since all the elements required to provoke an explosion are present. To combat the fire at this stage there are few dependable means, and the success of any of them is affected by a very great 'factor of chance.' The natural remedy against flames is water, applied by sprinkling or by flooding. The flooding of the section will usually take a long time, during which the fire will continue to progress, kept alive by the suction of the fan. Moreover, flooding against fire can only be adopted as a very extreme measure. As a matter of fact, it is almost

always worse than the complete and forced abandonment, unless the flooding can be restricted to a small area, because without controlling the fire quickly one adds destruction from the flood to the damage and risk of the fire.

As regards sprinkling, it will not commonly be effective and will almost invariably amount to failure, since the water will only reach the outside manifestation of the gob fire (the flames) without reaching the real seat of the fire, which is in the gob itself. Cases are known of flaming fire having been successfully extinguished by sprinkling with water, and the officials that were supervising the fight are to-day still wondering why it was that the water put out the fire. A closer study might have shown that steam (generated by the water on a very hot fire) and carbon dioxide were the decisive factors. The fire will, in most cases, continue to gain in violence and make its appearance at other places in the district, until conditions are so alarming that the mine has to be abandoned. This abandonment means the stopping of the fan. The natural ventilation, which will likely be reinforced by the heat of the fire zone, will carry a current of air through the mine and in the vicinity of the fire. The next logical step is to seal off the mine at the main intake and main return, and to await developments. The moment is naturally an anxious one, since, in view of the conditions now obtaining, a destructive explosion is imminent.

It is in order to avoid a situation such as this that the control of gob fires at their initial stages is strongly urged. Even then conditions are dangerous, but in the writer's opinion, it is at this stage that radical decision should be taken without hesitation, and the mine should be sealed off as tightly as possible at the main intake and return. The purpose is the total suppression of oxygen, not only in the fire district, but in the whole mine, and its replacement by a high percentage of other gases, explosive or not, the mixture of which will after a short time be inexplosive, due to its lack of oxygen.

As an illustration, some analyses of mine gases obtained at the slope mouth of the Albion mine, of the Acadia Coal Co., after it had been closed for a few days only, are here presented:

	Per Cent.
Carbon dioxide	7.2
Carbon monoxide	0.0
Oxygen	3.1
CH_4	31.4
Nitrogen	58.4

A few days later the percentage of oxygen had decreased to 0.86% and 0.42%.

The Albion mine, ever since it was opened in 1881, has had numerous very serious fires, almost always attributable to one of the causes mentioned above. In this particular case (the fire that took place in 1917-1918), an old abandoned section in the Third seam had been entirely isolated by a line of very substantial concrete stoppings, which, unfortunately, owing to a special arrangement of the ventilation circuit, were subjected to a heavy water gauge, which, except for the stoppings themselves, would have allowed a short circuit of the ventilation through this old section.

Workings in an underlying seam, having disturbed the stoppings in the Third seam, odour of distillation of coal was first perceived at almost every stopping, proving conclusively that most of the section was affected by heat. This heat was generated by the oxidation of the coal, occasioned by the filtration of the air through some defective stoppings and also through the coal itself, as will be shown later.

The first step taken was naturally a general reinforcement of every concrete stopping, followed by further reinforcements of weaker stoppings. The odour of distillation would disappear for a few days or even weeks, to re-appear suddenly at some other place, until finally after about eight weeks of special watch, a slight appearance of smoke made its appearance in one of the bottom stoppings that was being reinforced.

The section that was affected had an area of 1,410,000 sq. ft., in which a considerable body of crushed coal had been left. Following the policy advocated above, it was decided without hesitation, at the very moment smoke was seen, to

suspend further reinforcements of the stoppings isolating this section, and to surround this body of smouldering waste, slowly yet undoubtedly developing into a flaming fire, with an atmosphere almost entirely deprived of oxygen.

The analyses of the mine atmosphere have been given above, and the writer will merely reiterate that after a very short time the percentage of oxygen in the mine was less than 1%, that the percentage of CH_4 was about 36% and the percentage of carbon dioxide between 7% and 8%. Such a mixture made the extension, or even the existence of any fire impossible.

But this mixture, effective as it was to suppress the fire and prevent its extension, could not kill the heat that had been generated by the first stage of the coal's oxidation. The extent and importance of this heat was not and could not have been known, since at the first appearance of light smoke it had been decided to seal off the mine for the reasons already explained.

It had been considered that the fire zone was of small extent, in that after the previous reinforcement of the other stoppings, the warm points could be found only very near the stopping where the first smoke had been noticed. Therefore, basing a decision on similar experience in the past, it had been decided that the mine being closed for over 36 days, the temperature of any heated point should have been equalized by the natural and much lower temperature of the strata and of the whole atmosphere of the mine. But apparently the area of the zone that had been affected by heat was far more extensive than had been estimated.

During the last six years in the Pictou county coalfield, the re-opening of mines abandoned either on account of fires getting beyond control, or because of explosions creating fires compelling the temporary abandonment of the mine, has been occasioned no less than five times.

In all our re-entering of mine workings we have never made use of the fan. The mine having several outlets all sealed up as tightly as possible, with the exception of one (the one left for re-entry), a current of air or of gas is bound to be set up, notwithstanding the fact that there is only one opening. The slightest leakage in the stoppings of the return air-ways, if the intake is open, or of the intake if the return is open, will cause the heavy, cold, fresh air to drive out the light, warm gas in the mine.

Supposing that all the openings could be kept absolutely tight, with the exception of the one through which the re-entry of the mine is to be made, a current will be set up. The cold air will travel down on the pavement of the slope chasing the warmer gas which will escape by following the roof of the slope. As a matter of fact, during the re-opening of the mine, which will be described later, an experiment was made. All the surface stoppings were reinforced, covering them with sand and clay, even covering certain portions of the surface with sand and clay, and then the main return-air slope, lying alongside the main air-shaft, was opened.

The first effect was the emission of a large quantity of gas from the full section of the slope and after a very short while a regular current was noticed, going in at the footwall of the slope and coming out at the roof. After several hours, the cold air gradually found its way down the mine, the whole section of the slope becoming an intake. In order to counteract this movement, a wooden stopping was built below the connection that existed between the slope and the air shaft in the main return slope. This had the effect of reducing the amount of fresh air going down, but did not stop the current.

In order to still further counteract this tendency, an opening of 16 sq. ft. was made in the fan shaft, in the expectation that the air, instead of keeping down towards the mine, would return up the fan-shaft. Of a total volume of 8,400 cu. ft. of air the greater part was going straight in the mine, and 'barefaced' men could reach the stopping that had been built in the main return slope.

Finally, as a last effort to prevent the air from going down, the steam-fan was started at 14 R.P.M. in order to draw the air up the fan-shaft. Of a total quantity of 5,148 cu. ft. of air at the slope mouth, 4,500 cu. ft. was rushing into the mine at a point 25 ft. below the cross-cut between the main slope and the main air-shaft.

This experiment serves to show the effect of natural ventilation, especially on pitching seams. It should be evident that a state of equilibrium does not exist when bringing light warm gases in contact with cold heavy air. The quantities involved being very large, with considerable differences in temperature and density, heavy exchanges in setting up currents or natural ventilation are bound to take place and to persist for a very long time.

Therefore, in every re-opening of mines, the question of natural ventilation will have to be taken very seriously into consideration, especially in cases where it is important that fresh air does not get ahead at places where fires may exist, because, under these circumstances, fires are apt to start again suddenly. In such cases, it will be most important to direct the natural current by short circuit or new circuit away from any point where the oxygen of the air could, and would, cause damage.

As an illustration, at the time of re-entering the Albion mine in January, 1918, the first inrush of air, entirely due to natural ventilation, was 50,000 cu. ft. per minute, both fans being inoperative, every opening being closed, with the exception of the main slope (through which the re-entering of the mine was made) and a few boards taken off the fan shaft.

In order to prevent this flow passing anywhere near the seat of the heated section, this amount of air must be deflected before it gets near the fire zone by short circuiting the main intake and the main return. In our case, this had been done when closing down the slope at the time of the temporary abandonment of the mine. The short circuit of the air current was established at a distance of 1,300 ft. from the seat of the fire. Only a short distance below this point there was a blanket of gas through which no person could pass unless provided with breathing apparatus.

The great bulk of the gas lying on the circuit that had been established at a point 1,300 ft. above the fire zone was then allowed to clear for three or four hours. The ventilation being then considerably reduced, due to the cooling effect of the circuit, and also due to the dilution of the gas in the return, the short circuit at this point was suppressed and established further down, closer to the fire zone at a distance of only 460 ft. from the fire.

The men, following the air circuit, reached the point where the light smoke had been seen at the time of the closing of the mine. At this point, a strong odour of distillation was detected and soon after was followed by a smell of coal smoke. This discovery was most disappointing, because it did not leave any doubt but that the smouldering gob fire, which had been surrounded by an atmosphere containing less than 1% of oxygen during 36 days, had not been extinguished.

A few hours after, smoke again made its appearance, and for the same reasons as before, it was decided to close down the mine and to let it fill again with gas.

The mine had been opened for only 18 hours and all the different phases of the re-entering had been carried out as scheduled. As stated, this attempt was somewhat disappointing, coming after four other successful re-openings, carried out exactly under the same plan. In each of these cases, one month had been considered as being fully sufficient to allow not only to extinguish the fire, but also for the surrounding coal and strata to cool. In some cases the waiting had been less than one month.

This apparent failure inclined some of the mine officials to question the effectiveness of the methods, as above described, and every possible cause for the non-success was considered, including the likelihood of the section being connected with some workings of upper seams that were on fire, or connections through the subsidence that might have taken place between this seam and the underlying seam, and so on.

Finally, after new consideration and study of the situation, it was decided to proceed according to the same method as that adopted previously, but to keep a closer control of the natural ventilation. Since the first attempt had shown that 36 days had not been sufficient to cool down the fire zone, the new attempt was made almost three months after the first one, and contrary to what had been done then, it was intended that the new attempt should be made in different stages.

The first stage included the establishment of a ventilation circuit to No. 4 level 1,900 ft., and from there upwards towards the surface in a separate ventilation slope, the idea being to keep the ventilation as far away as possible from the fire zone. Instead of starting with as large a quantity of air as during the first attempt, the air current given by the natural ventilation at the start was only 10,000 cu. ft. per minute, which ultimately was increased to 21,000. The composition of the mine atmosphere at the time of this start was:

	Per Cent.
Carbon dioxide	9.98
Carbon monoxide	0.89
Oxygen	1.00
Methane	42.6
Nitrogen	45.9

—in other words, highly favorable.

The rest of the programme of this first day included the opening of a door on No. 5 level 260 ft. away from the seat of the fire. This door having been previously left closed by mistake, meant that any ventilation passing below No. 4 level would pass on the fire zone. This part of the programme meant that in order to avoid sending the ventilation past No. 4 level, as this part of the mine had to be kept under gas, the door had to be opened first by a Draeger team. The distance down the slope from the air station to the door was 940 ft. The men went down to the door but found that a fall that had taken place prevented its opening. At this stage a sample of gas on the slope at No. 5 level, 2,800 ft., was quickly analysed, and showed 17% of oxygen. The mine having been opened for almost 10 hours, dilution of the gas in the air current was taking place. It was decided to end there this first stage of the operations and to let the mine fill up again with gas for a few days. This was done, and after a short while, a gas sample taken through the pipe at the slope mouth gave:

	Per Cent.
Carbon dioxide	10.9
Carbon monoxide	0.42
Oxygen	0.62
Methane	30.00
Nitrogen	59.00

A few days later the second part of the programme was proceeded with; this consisted in levelling off the fall that prevented the opening of the door at No. 5; also the closing, by a temporary wood partition, of a small ventilation head next to the fire stopping. Twenty thousand cubic feet of fresh air, forced in the mine through natural ventilation, was admitted in the slope, but only down to No. 4 level. From this point the work planned was done by the Draeger team in an atmosphere including only a small proportion of oxygen, working at 940 ft. from No. 4 level, which was the air station. This was practically all that was done that day. At the end of the day, the Draeger team had levelled off the fall, opened the door referred to and closed up the ventilation head. They also took a sample of gas almost against the fire stopping; this sample was most reassuring as it showed 5.5% of carbon dioxide, 10% oxygen, and 24.5% of methane, after the mine had been opened up for almost twelve hours.

The value of this information alone would fully justify the use that can be made of a well-trained and well-organized Draeger team. The information implied that all previous work had been successful in keeping an explosive mixture away from the fire zone; also that it was possible to work with restricted ventilation in the close vicinity of the fire stopping for about 12 hours, without allowing the diffusion of the air and of the methane to constitute an explosive mixture.

Further valuable information brought back by the Drager team was that nowhere close to the fire section could they feel any indication of heat. There was, therefore, every reason to believe that the period of three months' rest that had been given to the mine, during which the proportion of oxygen had been less than 1%, had been sufficient to equalize the temperature of the smouldering fire to the surrounding temperature of the mine. In other words, not only had the combustion been suspended, but the surrounding temperature was low enough to avoid excessive avidity of hot combustible matter for oxygen.

At this stage it would have been quite possible to complete the programme, which included the building of a temporary stopping outside the one where on two different occasions smoke had been noticed; but before proceeding to this last stage, the whole mine was once more allowed to fill up with gas.

Finally the last day came. Adopting the same procedure the air was allowed in the mine, and this time permitted to pass No. 4 level, travelling down towards No. 5. Barefaced men started, followed the air and built the temporary stopping mentioned above. The old stopping and all its surroundings were found to be perfectly cool, giving no indication of heat and no odour of distillation. The building of the temporary stopping was rapidly completed, and immediately after the building of the permanent concrete stopping was begun. We had been successful, and once more the mine officials felt that their confidence in this system of fighting fire was well justified.

In order to remedy the real cause of the fire, namely filtration of air through the coal surrounding the stoppings that sealed off the lower part of the section, a system of pipes connected to a natural supply keeps a head of a few feet of water all over the bottom part of the section. The overflow taking place through the coal itself is the best proof that the air cannot again enter to feed the gob with oxygen. All this occurred late in the year 1917 and early in 1918.

On this occasion, as before, it had been found that it was not only possible to suspend a gob fire by curtailing its supply of oxygen, but also that it was possible, if time were liberally granted, to equalize the temperature of the smouldering fire with the temperature of the rest of the mine by means of a restricted natural ventilation.

Our faith in this logical, safe, and efficient system of preventing an underground fire from getting rapidly beyond control had once more been fully justified.

LONGWALL MINING AND COAL CONSERVATION

By J. H. Cunningham

Annual Meeting Mining Society of Nova Scotia, Glace Bay, May, 1920.

A prominent mining engineer has remarked that when a coal miner is once inoculated with 'Longwall' he is forever immune against future attacks of 'Room and Pillar.' Although this is perhaps an exaggeration, the fact yet remains that a 'Longwall' miner when selecting a method of working will usually view the situation first from the 'Longwall' standpoint and then adopt another method only after finding that conditions are totally unsuited to his favorite system. This point of view may not be as prejudiced as it at first appears, because a man who has actually found a superior way of doing a thing, is undoubtedly the best judge of whether his new method is better than the old; and for this reason a miner who has once learned the numerous advantages of 'Longwall' when worked under suitable conditions, is very loath to change to any other system, unless the conditions make the use of 'Longwall' prohibitive.

The remarks which follow are not intended, however, to introduce a controversy on the two systems, 'Longwall' and 'Room and Pillar' (for both have their advantages under certain conditions), but are merely intended to indicate the considerations which should govern the use of the former and to emphasize the effect of its use on the conservation of the coal resources of the country.

The application of 'Longwall' mining is of course governed by certain natural conditions in the coal seam itself, and in the overlying strata. Any seam of a hard or medium hard nature, varying in thickness between 30 inches and 6 feet, with a good roof and floor, and lying between 300 and 3,000 feet below the surface is capable of being worked 'Longwall,' provided it is not badly faulted; and the success of the operation depends entirely upon how these conditions are utilized. Some operators prefer to add that the seam should not have more than a slight pitch, but there are so many examples of 'Longwall' worked successfully in seams having a steep pitch that such a statement does not appear to be warranted.

The requirements of a good roof for 'Longwall' are the same as for any other method of mining, but in addition it should be of a material that 'shoots' well and is suitable for building rock walls. The floor also should be fairly hard in order that it will not 'creep' readily when the pressure comes. Above the working roof the most suitable measures consist of alternate strata of sandstone and shale. If entirely of sandstone it makes a good enough roof, but if a break occurs, the fissures extend too far and there is too much opportunity for water to collect. Where beds of shale intervene, however, the fissures are kept from extending, and if any subsidence of the measures occurs, the beds of shale open out and help to cushion the weight. In this way the effects of the subsidence are not felt at any great distance above the coal seam.

The preliminary development of the seam need not be discussed here, since shaft sinking, opening out of the pit bottom and arrangement of the shaft pillar will be carried out in the same way, whether for 'Longwall' or any other method. 'Longwall' proper, therefore, may be commenced as soon as the main roads has reached the boundary of the shaft pillar, and there should be sufficient information available at that time to indicate how the work should be opened out. While the shaft was being sunk, the thickness and nature of the overlying strata have been ascertained, and while driving the various roads from the pit bottom to the boundary of the shaft pillar there has been an opportunity to find out the nature and thickness of the seam and the direction of the cleat. If the seam is hard, the direction of the face that is to be laid out should run nearly parallel with the cleat, and if it is of medium hardness, the face may proceed 'half on.' In some instances, it has even been found feasible to work 'Longwall' in a fairly soft seam by keeping the face going 'end on.' Another feature that helps to determine the direction of the face is the direction of the joints in the roof strata. The face, of course, should not run parallel with the joints.

In continuing the main roads after the shaft pillar is passed, two methods are followed. One is to protect these roads by pillars of a suitable size for the purpose of maintaining a tight air-course and insuring height for the roadways. The other is to begin extracting all the coal as soon as the shaft pillar is left behind and carry the main roads forward through the waste and depend upon the excellence of the roadside packs to secure a proper air-course. The latter method certainly entails more 'brushing' for height than the former, but it is contended that the advantage of having all pillars removed from the area of the 'Longwall' operations will in time more than offset the temporary disadvantage of higher cost for 'brushing.' It is also true that the first cost of the pure 'Longwall' roadway may not be any greater than the cost of driving in the solid, in spite of the 'brushing' entailed, since when the complete extraction method is followed, the main road is also a part of the working face and carries the same mining rate as does the regular face. In this way the high rate per yard for driving levels and headways is eliminated.

One of the most frequent mistakes made in 'Longwall' is that of paying insufficient attention to the contact zone between the solid coal and the 'Longwall' waste, and this is probably the best time to consider this side of the question, because the whole system of 'Longwall' has often been condemned as unsuitable, simply because proper precautions were not taken in commencing the work. As soon as a 'waste' is opened beyond solid coal, there is bound to be a subsidence of the roof. The amount will depend on how tightly the waste is stowed in the area included within the contact zone. If the method of complete extraction of pillars on main roads is followed, special care is necessary in building the pack walls next to the shaft pillar, and the waste should be stowed as nearly solid as possible. This will reduce to a minimum the breakage of the roof measures, and will also prevent air leakage that would otherwise occur at this point. On the permanent roads the pack walls should have a greater thickness than on the secondary roads; this serves a double purpose by making the permanent roads tight for an air-course and by maintaining their height for both haulage way and air-course for some time to come. What the thickness of the roadside packs should be will depend largely upon the depth of the seam below the surface, and the roadways as first laid off should be a little wider than required for a finished roadway, since the pressure will push the walls out a few inches into the road. When a double track is necessary, it is sometimes advisable to consider if two narrow roads would be better than a single wide one, and allow the outgoing traffic on one road and the incoming traffic on the other The upkeep of two single-track roads will in many cases be less than that of one double-track road.

In the case of a 'Longwall' road running alongside a pillar, the coal rib should never be used for one side of the road. Against the pillar there should be a well-built pack wall from 10 to 20 ft. thick, so that the road will be formed with pack walls on either side. This will cushion the weight on the road and throw the loose end of the broken roof in the gob, rather than directly over the roadway.

In turning off secondary roads, care should be taken to strengthen the turns. This may be done by using a timber pack wall if the building stone is not of the best, or if there is any doubt about the roof. If these roads are also to be used as haulage ways, similar precautions should be taken with them as with main roads, only to a lesser degree.

In breaking off the 'gateways,' or short roads leading directly to the face, the distance between centres will depend largely upon the thickness of the seam on the amount of building material available and on the method of dealing with the coal at the face. A few remarks on the origin of this system will throw light upon the practice to be followed.

'Longwall' originated in the north of England, and was devised as a measure of necessity. Many of the thick seams in that part of the country had been worked out and it became necessary to mine the thinner seams in order to maintain

production. No particular difficulty was found in mining these thin seams by the 'Room and Pillar' method, although it was found that, in order to do so, a considerable increase in both mining and datal labor was necessary: also, that during a portion of the day rock had to be hoisted instead of coal, all of which made it very difficult for the thin seams to compete with the thicker seams still being worked. The 'Longwall' system was therefore devised to increase the daily production of the miners and also to provide stowage for the rock instead of sending it to the surface. It was found in addition that the stowed rock could be used to support the roof and to assist in working the coal face properly.

'Gateways,' therefore, came to be broken off at distances to ensure that the waste would absorb all the rock 'brushed' down in providing the necessary height for the gateways. In seams varying between four and six feet in thickness it has sometimes been found advantageous to lay a temporary track along the face and load the coal directly into the mine cars, thus reducing shovelling to the minimum. In order to do this, a good roof is necessary, because the roof-supports must be kept back some distance from the face. If this system can be employed, 'gateways' can then be placed further apart and considerable saving made in 'brushing.' In thinner seams the same result is sometimes obtained by using face conveyors travelling between two consecutive 'gateways.' Nevertheless, the system most commonly employed is to load the mine cars as they stand at the face in the 'gateway.' In applying this system it is evident that the 'gateway' must be kept sufficiently close so that excessive shovelling will be eliminated. Where the direction of the face is on the 'level' it is customary to divide the face equally on both sides of the 'gateway' for loading the coal; or if the direction of face is inclined, it is divided so that the long shovel is on the high side of the 'gateway,' and the short shovel on the low side.

These are the preliminary considerations. The actual working of the 'Longwall' face may now be discussed. Mining may be either by hand or by machine; but owing to the comparative scarcity in this country of skilled 'Longwall' hand miners, it is probable that any development of the 'Longwall' methods will be by the use of continuous cutting machines. The post or other types of reciprocating machines are useful for some of the auxiliary work, but they are not suitable for steady cutting along the face. The selection of the motive power for the machines depends upon whether the seam is gassy or not. If gassy, compressed air should be used, but if not, electric power is more efficient and satisfactory. The type of cutter used on the machine depends upon the hardness of the material to be cut. If the coal is very hard the cutter of the disc type will be found the strongest. In some workings this type of machine is even used for mining ironstone. If the coal is of medium hardness the chain type cutter will probably give the greatest satisfaction; and in a soft seam (or one in which the coal is apt to settle and jam the cutter) the best type is the bar machine. The last mentioned has also proved very satisfactory in seams where the floor in uneven, since the bar is more flexible than that of the other cutters and will ride more easily over the irregularities.

The length of cutter bar used, or in other words, the depth of the undercut, depends upon the hardness of the coal. This may vary between four and six feet and should be graduated so as to prevent the coal from falling and clogging the machine before the cut is properly made. When used rightly, the depth of undercut can be made of great assistance in working the coal so that very little explosive will be needed. In many instances the coal will fall in large blocks without the use of any explosive, although to facilitate loading, light shots must be used to break up the blocks.

In undercutting a hard seam, especially where the roof also is hard, very little 'spragging' will be necessary, whereas in a medium or soft seam 'sprags' or 'breakers' should be set fairly close to the cutter as it moves along the face. There are two reasons for this: First, to prevent the coal from settling on the machine, and, secondly, to assist in breaking down the coal in blocks not too large to be handled.

The ease with which coal comes down after it is 'mined' depends also upon the nature of the roof. A hard roof does not 'give' immediately after the coal is undercut, and the face therefore must stand for a while before it is worked, in order to obtain the benefit of the roof pressure. On the other hand, a flexible roof will begin to bend and exert pressure almost as soon as the undercutting is completed.

For these reasons, the face should not be advanced as rapidly under a hard roof as under a soft one. otherwise complete advantage of the roof pressure cannot be taken. The essential feature is to advance the face rapidly enough to keep the roof bending and yet prevent it from breaking. 'Breaks' will occur from time to time, but they should be so controlled as to ease themselves in the gob rather than over the working places. When excessive pressure is felt in the roof on the gateways, it is sometimes beneficial to strengthen the roadside packs and leave a waste space between the packs without any support. This will tend to throw the 'break' in the gob and thus relieve the weight on the road. For this reason standing timber should never be left in the waste because it prevents regular subsidence and throws additional weight on the roadways and face instead of allowing it to come where it will not do any damage. It might be noted here that the distance between the packs and the face has a great influence on the working of the coal. If the roof bends easily and breaks the coal down freely after it is undercut, the packs should be kept close to the face, whereas if the roof is firm and therefore slow in exerting pressure on the face, the packs should be kept farther back in order to give the roof a better chance.

It will be seen from the foregoing remarks that a considerable amount of skill is necessary in working a 'Longwall' face properly, and each face must be made a subject of study. Probably backwardness in the adoption of 'Longwall' methods is attributable to this fact.

Another important subject is the application of 'Longwall' mining. Its application to the conservation of coal may be briefly pointed out. In the first place, it provides for complete extraction of the seam as compared with a system of partial extraction under the 'Room and Pillar' method. Under the latter system more coal, as a rule, is left standing in pillars than is taken out in the first working, and it is only in rare instances that these pillars are completely recovered. Even if they are ultimately recovered, there has been a continued loss in them year after year, on account of crushing and 'spalling off.' Moreover, when they are finally worked, there is a larger percentage of slack produced than in the first working. With 'Longwall,' there is no deterioration, for the reason that the face is kept fresh continually.

'Longwall' will produce a larger percentage of round coal than any other system, since there is always a loose end to work on; this permits a smaller quantity of explosive to be used, and by being able to make use of the roof pressure, the quantity of explosives emplnyed is still further reduced. A reduction in the amount of explosives means a lower mining cost.

When coal is being produced from a continuous face, rather than from a number of faces separated by pillars, it is possible to have a greater concentration of work than is possible with other systems. This feature reduces the cost of supervision as well as that of haulage. For example, a 'Longwall' headway with gateways broken off on the level, need only be half the length of a 'Room and Pillar' headway producing the same tonnage, provided that in the latter case half the coal is left in pillars. This will permit the use of less rope, fewer rails, smaller haulage engines and less rolling stock.

The problem of ventilation is also much simplified; and ventilation can be carried on more efficiently and with a smaller quantity of air. The nature of the face permits it to be swept continuously with a stream of fresh air reaching all parts of the face equally instead of having it travel up one side of a room, then back the other side, passing through a crosscut and travelling up one side of the next room before it

again reaches the face. In other systems, for seams of the same thickness, there is a greater friction loss in ventilation and, in the case of deep seams, the additional heat generated from the crushing of pillars gives an increased temperature which must be offset by increasing the volume of air. All of these losses lead us back to the economic use of power. Saving in power means a reduction in steam consumption, which also means a saving in the quantity of coal used. Additional saving of power is also possible in the operation of the mining machines, especially where the heavier types are used. These machines require power to load and unload them and power to transport them to the next place, whereas in 'Longwall' practically all the power they consume is for productive work. The removal of machines from one place to another entails other losses as well. If half the time in the shift is used in moving and setting up the machine, both men and machines are engaged in unproductive work during half the time.

In addition to the saving in coal, explosives, hauling equipment, and all the material necessary for the production of power, 'Longwall' methods effect a considerable saving in timber. With a good roof practically no timber is required overhead in the gateways, especially if the roof can be arched. The only timber necessary on the sides is at the turns of the road, and perhaps an occasional timber pack to stiffen up the rock wall. At the face, a certain amount of timber is required to protect it while being worked, but as fast as the pack walls are built this timber should be withdrawn. Usually, it can be used repeatedly and when no longer fit for face timber, it will serve for building packs. With the rapid depletion of our forests, timber will each year become more costly and more difficult to obtain.

In a locality where there is a large and cheap supply of timber available, it is sometimes possible to work 'Longwall' under rather unusual conditions. The writer has in mind a colliery in Western Canada where the thickness of the seam varied between two and ten feet and where 'Longwall' would not ordinarily be considered suitable because, in addition to the variation in thickness of the seam, the roof was not particularly good. The seam, in many places, also contained several thick layers of soft dirt which was not suitable for packs. On account of a plentiful supply of timber, however, the packs were built entirely of timber set skin to skin along the roadside, and the dirt from the seam was stowed behind the timber packs.

Another instance in which the adoption of 'Longwall' was feasible on account of conditions out of the ordinary may also be noted. In a seam where the roof is only fair, but where the floor contains good building material, it is feasible to leave the roof unbroken and support it on timber resting on top of the pack walls and then 'brush' the pavement in order to secure the necessary height and to provide sufficient material for building the walls.

There are also a number of other special conditions apart from the standard conditions required, which will permit the working of a seam by 'Longwall,' but space does not permit of them being referred to at this time. It may be noted, however, that there are two main varieties of 'Longwall' which differ only in general principles, and not in detail. These are 'Longwall Advancing' and 'Longwall Retreating.' The former, which has been the subject of discussion so far in this paper, is in most common use. In 'Longwall Retreating,' the main roads are driven through the solid to the boundary as quickly as possible, and the face is first opened out at the boundary instead of near the shaft bottom. This system will probably pay even better than the other in the long run, but it requires a much longer time to develop a large output. To compensate for this loss, however, the cost of production will be lower, because the roadways are easier to maintain and the wastes as they are formed are always being left behind. In 'Longwall Advancing,' when the workings are opened up clear of the shaft pillar, a large output can be produced in a comparatively short time. Sometimes both systems are adopted and one side of the pit is opened for 'Longwall Advancing' and the other for 'Longwall Retreating.' Another variation is sometimes intro-

duced by laying out the pit in large panels, each of which is surrounded by its own pillars and each panel is worked out individually by 'Longwall.' A very successful example of this system is found in Lanarkshire. The seam is between six and seven feet in height and contains a rock band about one foot in thickness in the centre of the seam. A section of the mine was laid out in panels, and the main roads and gateways were 'driven in' the full height of the seam. The bottom half of the seam between the gateways was worked 'Longwall' and the bottom bench of coal and the rock band were extracted in the first working, the top bench of coal being left for the roof. After the boundary of the panel had been reached, the top bench of coal was brought back by 'Longwall Retreating.' In the second part of the work one of the principal advantages was the ease with which the coal could be 'loaded out.' When retreating, the tops of the mine cars as they stood in the gateways were on a level with the floor from which the coal was loaded,

Hydraulic stowing is obviously feasible for either 'Room and Pillar' or 'Longwall' mining, but even without it, 'Longwall' can be practiced in undersea workings with very little danger, provided there is sufficient cover and the roof measures are suitable. What is probably the most successful 'Longwall' operation in the Dominion to-day is carried on in a submarine area, namely, the workings on the Newcastle seam in No. 1 shaft at Nanaimo, B.C. 'Longwall' was commenced in the bottom seam of this colliery in the year 1904, and to date the production has amounted to about 3,000,000 tons. This is in addition to the output from the Douglas seam, which lies above it and is much thicker. Practically all the 'Longwall' operations have been in a submarine area, with a covering varying in thickness between 400 and 600 feet. The subsidence recorded to date is only from 12 to 18 inches, and the seam has an average thickness of three and a half feet.

So far as the writer is aware, the best example of a seam that contains all the elements of successful 'Longwall' operation in this district is found at Sydney Mines, in the bottom seam of Jubilee colliery. Practically all the ideal conditions as noted in this paper are found there, and although the development is not yet very extensive, the preliminary work has been carried on in a very thorough and careful manner. Before long it will probably become one of the principal 'Longwall' operations in the country. When this is accomplished, the introduction of 'Longwall' methods in other seams of similar nature, in the Cape Breton coalfield will soon follow, and will conduce to a development in some of the submarine areas that will result in the saving of thousands of tons of coal annually that would otherwise be left standing in pillars and probably lost, not only to this, but to all future generations.

THE CONSERVATION AND DRAWING OF MINE TIMBER

BY P. T. PRENDERGAST

Annual Meeting, Mining Society of Nova Scotia, Glace Bay, May, 1920.

When asked to present at this meeting a paper on the conservation and drawing of mine timber, it was with a full realization of the fact that this subject had already been taken up by the Society on different occasions.

This matter has, heretofore, been discussed from the theoretical point of view; and while, no doubt, much valuable information is obtained from a discussion of that nature, it is proposed here to deal briefly with the subject from the standpoint of practical experience (by which is implied the writer's own personal experience) as acquired during the years spent in the mines of Cape Breton.

Conservation of raw materials has so taken hold of the public mind, and is regarded as of such vital importance by all who realize our responsibility to those who are to come after us, that it behooves everyone to use every available means to prevent wastage of timber in and around the mines. In recent years the item of mine timber has entered so largely into the cost of coal production that any measures taken towards the economical use of our available timber supply will prove a great investment in the years to come.

When, in 1893, the Dominion Coal Company was organized, the cost of the regulation 9-ft. prop was from 45 to 50 cents per dozen (laid down at the pit mouth) and the specifications in use at that time required that all should be of black spruce, six inches in diameter at the small end. This was in the days when the seams were being mined under comparatively shallow covers. To-day, the only available supply of timber is balsam fir, with a small quantity of spruce—and the cost is now four and a half times as much.

Mining at greater depths with an inferior wood to support the roof means that more timber will be required, and the manner of use of our present supply, and the steps taken to ensure the future supply, will have to receive much greater consideration in the future. The drawing of mine timber and the use of preservatives are two ways of conserving supplies.

Each successive year we are going deeper and deeper into the earth to obtain the coal for operating our industrial plants and for keeping our homes warm. This means that each successive year a much greater thickness of strata has to be supported. It will be found, where mining operations are carried on at any great depth, that even an increase in the size of pillars will not prevent the breakage of timber.

Where great weight of strata has to be reckoned with and where lateral pressures are troublesome, we find that on our roadways the roof is forced down on the timbers, causing them to lag and finally break. To overcome this breakage is one way of assisting conservation, as all broken timber must be replaced by new; and the writer submits that a great deal of overhead timber destroyed in this way might be saved by shortening the span between the supports. It may not be good policy to recommend the placing of props too close to the haulage roads, but the shortening of the span of overhead booms could be done in a great many cases with increased safety to employees, and at a saving in expensive timber, if certain methods are adopted with a view to conserving the timber supply.

For landings and turnouts where long booms are required, and where, on account of the length of the span and the area to be supported, extra heavy timbers are required, it will be found that even these timbers break very often, and it becomes necessary to replace them. Also, the thin end of the pillar is here continually breaking away, necessitating more timber and labour, adding greatly to the cost of production. This is an unpardonable wastage, as it not only means an unnecessary use of new timber, but also results very often in a decreased output of coal, both of which could have been avoided had the work been properly done in the first instance.

The building of neat packs in the 'V' of the road, on which would rest one end of the overhead timber, thus shortening the span, would result in a great saving of timber. This would not only be a safer method, which is in itself sufficient to justify its use, but would result in a great saving in cost, as much shorter and cheaper timber could be used. It would also serve the purpose of preserving the ends of the pillars, which are continually splitting off. These packs could be built of broken and discarded timber of which every mine has a fairly good supply. Hence, the only cost would be that of labour.

Not only would such packs conserve our supply of timber by lessening the amount of broken timber and protecting the weak ends of pillars, but it would be a satisfaction to the mine officials to have this work done properly along roadways and landings. To have long booms carrying a heavy load is by no means a nerve-soothing condition.

As indicating the increase locally in the cost and consumption of timber in eight years, it may be cited that in one mine 64,000 more pit props were used in the year 1919 than were used in 1911, and the cost of props increased in that period a hundred per cent. In 1911, there were four tons produced per prop, and in 1919 only 2.4 tons. This difference may be attributed partly to different conditions, but principally to the deeper workings and the maintaining of longer roadways and airways. The replacing of broken props along roadways is a heavy drain on our timber supply.

In the deeper local mines, where the edges of pillars cannot be depended upon as a safe support for booms, and where all booms must be supported by props or packs, such props are very often subjected to more stress by the heaving of the bottom than by the weight of the roof. Props are thus forced up until they cut through the overhead boom, thereby weakening the roof support, and requiring the use of new timber. This heaving of the bottom is usually from one of two causes: (a) where pillars are not left sufficiently large, (b) where the strata below the coal are of a weak and broken nature. The first of these causes suggests its own remedy, while the second presents a problem that is somewhat more difficult to solve.

Since heaving of the bottom results in a great wastage of timber through breaking, one might suggest that the most easily applied remedy would be to use a better quality of timber. With this conclusion all will doubtless agree, provided there is not a material increase in the cost of coal per ton. However, looking at this question from the viewpoint of conservation, and bearing in mind the fact that future mining developments must be carried on at a much greater depth than at present, it might be the proper course to conserve for future generations the more valuable woods, while the cheaper and faster growing woods might be utilized in sufficient quantities to meet present requirements.

While still dealing with the question of conservation of mine timber, there is another phase of it which is worthy of attention. At present, the wood that is to be used as timber in our mines is cut and handled in large quantities during the winter months, and, while the hauling is good, is hustled out of the woods and piled along the railways and highways. Some of this will not be used in the mines for a year, and in many cases for a much longer time. During all this time it is lying in close piles and deteriorating from disease and decay. As a result, when it is finally placed in the hands of the miner, its strength and life in many cases is reduced by at least 50 per cent. In timber that is closely piled the free circulation of air is prevented, and as the bark is continually moist, we have the most favourable conditions for fungoid growth.

A great deal has been done and said by mining men with regard to treating timber before it is used in the mine, and much good has been accomplished by such treatment to lengthen the life of mine timber; but to apply preservatives to wood that is already well advanced in decay is merely money and energy wasted. The proper time to apply the treatment is when the timber is green, that is, before the bacteria have had an opportunity to become effective, and not after they have had a start of many months.

The drawing of mine timber from worked-out parts of a mine is a problem that has been given a great deal of thought by those interested in coal mining. It has been contended on many occasions that timber cannot profitably be drawn in Cape Breton mines, as such work cannot be done by common labour. The writer agrees with this contention in so far as it applies to inexperienced labour. Experienced men must be employed at such work, and labour of this sort must be paid at a fairly high rate. The timber must then be carried to the roadways, from where it can be hauled to the working places.

In this country, where mine timber is principally of balsam fir, a large percentage, after being drawn, is not worth re-setting. The writer has had this work done at a fair profit by having the men take only the timber which could be drawn without too much labour and which would make good timber for re-setting. Having in mind only the cost of coal per ton without regard to the conservation of timber, the writer believes the drawing of timber in this way will be found most profitable. In places where the bottom is fairly strong, and does not heave sufficiently to break the timber, and where the roof is not too heavy, it will be found that a great many timbers can be drawn which will be fit for re-setting. This would probably amount to from 50% to 75%, particularly where such timber is of spruce.

The writer has also used broken timber in the making of road ties in the mine, and finds that these can be made at about the same cost as new ties, but it requires a man who has had some experience at this sort of work. It may be suggested that such work could be done better on the surface, if the wood was brought there; but the extra handling would increase the cost, and would probably result in the lessening of the output by interfering with the free handling of the coal.

When booms must be removed, because of their being broken, either through deterioration of the wood itself, or through some other cause, and when either portion of such booms is of sufficient length to make a prop, they may well be used for that purpose. Even when the fibre becomes brittle through deterioration, and the piece is of no further use as a boom, it can be used as a prop with fairly good results.

Rafter timber, or in other words, timber that has been soaked in water for some time until it sheds the bark, will stand underground conditions much better than wood sent into the mine with the bark attached. It might also be stated that timber cut after drying on the stump in areas that have been swept by forest fires has much better lasting qualities than wood cut when green. It follows then, that it might be better to have timber peeled when cut, and piled so as to give it a chance to dry, although this would of course add materially to the cost.

It may be argued that the additional cost would not be justified from the fact that the life of much of the timber used in the mine is of short duration owing to conditions already mentioned. However, it is recommended that a certain percentage of the timber should be prepared in this way, so that the mine officials could use it in travelling ways, airways, and such other places as are to be maintained for a longer period.

In the timbering of mine roads, where it is necessary to maintain such roadways for a number of years, and where such timbering must of necessity be of a more or less permanent nature, imported timber of a quality superior to our native wood is sometimes used. This is generally yellow hard pine. This wood does not seem to be affected by underground conditions if it is quite sound when put into the mine, and has not had too much resin extracted from it.

Steel girders are sometimes used where support of a permanent nature is required. The initial expenditure is fairly large, but, allowing for the cost, and having due regard to the permanency desired, any reasonable expenditure in this respect is justifiable, as it will ultimately prove much cheaper than if ordinary timber had been used.

It has been proven conclusively that, particularly where double tracks are required, main roads can be supported as cheaply by steel rails that have been condemned for ordinary railway use, as by round spruce timber, the reason being that many more spruce booms would have to be set for a given distance. With respect to durability, the life of a steel boom, as compared with one of spruce, might well be placed at 20 to 1, and the ratio may even be much greater. This refers to the ordinary 80-lb. steel rail.

Portions of haulage road boomed with these rails, say six feet apart, supported on cured spruce props, and having sufficient overhead lagging, would require, if spruce timber was used instead, three spruce booms to every rail so used, and would entail three times the labour in setting.

It is generally supposed when steel is used instead of wood for timbering, that the first cost is much greater, but this is not always so. For instance, where roof conditions require a steel boom every three feet, then at least two spruce booms must be placed in the same space. The average cost of a steel boom 14 ft. long is $5, the cost of two special spruce timber props is 30 cents each, and the cost of putting up is about $2.40, making a total cost per boom of $8. The cost of two spruce booms 14 ft. long is $2, of four props for supports $1.20, and of setting $4.80; this also gives a total of $8. Where it becomes necessary to set steel booms at less than three-foot centres, owing to the weight of the roof, the writer would not use spruce even if set skin to skin, unless propped in the centre—and centre props on haulage ways are always a source of annoyance and delay.

CONSERVATION IN COLLIERY POWER-PLANTS

By W. S. Wilson

Annual Meeting, Mining Society of Nova Scotia, Glace Bay, May, 1920.

A constant and steady supply of power is necessary for success in modern coal mining, and, to say the least, it is unfortunate that the equipment necessary for the production of power is often regarded as a necessary evil. So long as the power supply continues without fail, the power plant is commonly regarded as having performed its duties, thereby eliminating one of many worries for the oft-times sorely tried colliery manager.

When coal was being mined for 80 cents a ton, none cared very much what amount was consumed for power, and such a matter as conservation of fuel in the power-plant received scant attention. Coal was too plentiful and too cheap, and the commercial advantages of scientific operation doubtful. Even now, with the cost of production constantly rising, there remains that contempt for the home product of which a ready supply is always at hand. The manager might well forget the plant (which was probably designed by engineers and is operated by men of the same calling), if everything is running smoothly and it appears to be in good repair.

The time has come for a change in attitude. Under existing conditions with the prevailing high prices of coal, and with the Nova Scotian coal mines providing for little more than the home demand, it is of paramount importance to save every ton of coal possible. Although from the owner's standpoint, the production of coal is the object, and the production of power is only incidental thereto, it is safe to say that more coal is used for firing boilers in the coal mining industry than in any other industry (steel included) in Nova Scotia. About 500,000 tons of fuel was used in colliery boilers in this province during last year alone.

One can picture a power company with a plant of such a size as to use this amount of coal in a year employing every con-

ceivable device to generate power as cheaply as possible. It need not be emphasized here that it is to the colliery owners' benefit to generate their power cheaply in order to reduce mining costs and to have a greater surplus of coal for sale—every ton needlessly expended represents not only the loss of the price of winning that coal but also the revenue that might have been derived from its sale.

During the war, our neighbours across the line made noteworthy efforts to save 50,000,000 tons of coal per year. Their fuel conservation board had the authority to close any plant wasting coal. Great Britain made similar efforts with a view to conserving not only the coal brought to the surface but that left in the ground; and everyone is familiar with the unhappy circumstances to which France and Italy found themselves reduced. In spite of this, and of such a serious shortage of coal in this country, little has been done in Nova Scotia along these lines, and we have seen the continuation of inefficient plants as well as a serious reduction in outputs, the cause of the latter being well known to all. Warnings have already been sounded in Great Britain that in future it will be considered criminal to burn coal even for steam-raising purposes without first extracting its valuable by-products.

Considering the wonderful developments that have been made in recent years in improving the methods of power generation from fuel, it surprises those to whom the natural limitations of this process are not familiar, when they realize the small amount of the total energy of the fuel that is utilized in useful form. Even among large central plants generating electrical power there are very few in which as much as 12% of the fuel's latent energy is obtained and there are many obtaining as low as 6%. With all the latest equipment, the highest obtainable is about 18%; and this is so close to the theoretical maximum that this amount will never be exceeded unless entirely different methods, involving radical changes in power generation, are developed. Perhaps it is here that gas-engines have an advantage over turbines in that their output of power is generally about twice as much as that obtainable by steam sets from the same amount of available energy.

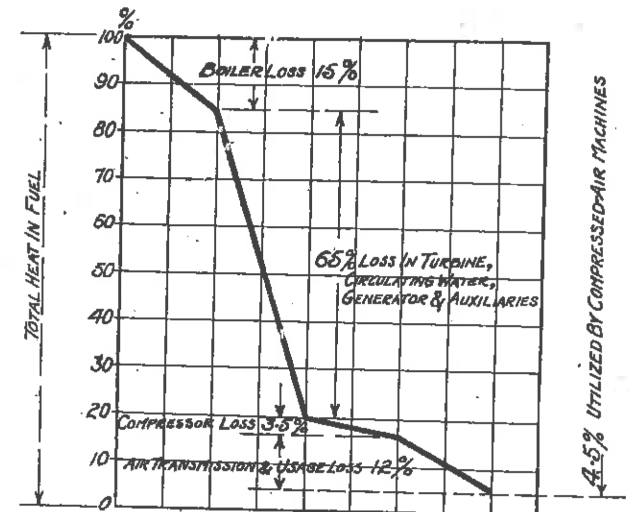


FIG. 1.

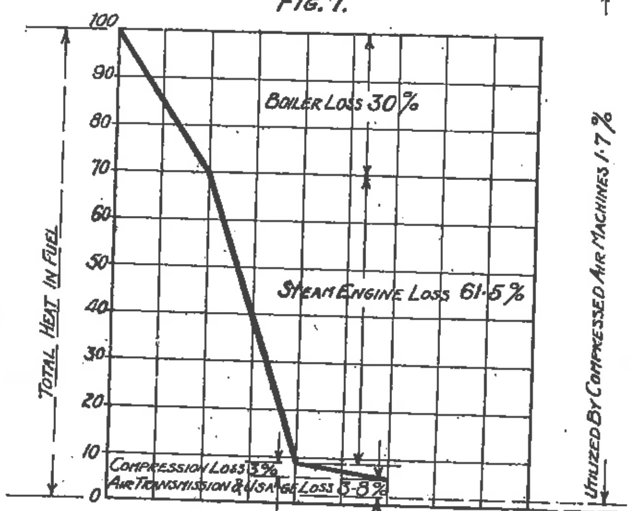


FIG. 2

A study of the distribution of the various losses that occur in the generation and use of power at collieries is particularly interesting. For the purposes of comparison, an electrically driven compressor in conjunction with a modern turbine power-station working under good loading conditions, an ordinary steam-plant with steam-driven compressors in good condition, and a similar plant in the poorer condition only too frequently met with, are shown in Figures 1, 2, and 3 respectively. The first is ideal. Results as shown by Figure 3 are easily obtained, surprisingly how easily, but those in Figure 2 can only be attained by constant and careful attention on the operators' part. All the care and supervision, however, are not required at the power-plant, as the distribution and use of compressed air in the mine may cause serious losses. These, as a rule, are more easily detected and remedied than losses in the power-plant, and it is here that the greatest effort should be made to maintain the efficiency of the system.

Generally speaking, the maximum thermal efficiency of a plant is fixed when the plant is installed. In other words, it is possible with all the different combinations of apparatus available and the constant attention of the operating crew to obtain a certain high standard of efficiency as shown, for instance, by Figure 2. Most of the variations in efficiency can occur in the boiler room. Results in the compressor house are fixed, to a certain extent, by the design of the machinery. All that is asked of the operator is to deliver a certain amount of steam at a given pressure and the machine automatically takes care of the rest unless, of course, it is allowed to depreciate in an unwarranted manner. A definite standard of efficiency can thus be set for a machine and this can be maintained without any extraordinary precautions. These efficiencies are rapidly approaching those theoretically obtainable, but even the best appear to be woefully wasteful, converting into useful form only about 23.5% of the energy of the steam. Yet these are known as very efficient units and this poor performance is unpreventable as far as known. Thermodynamic laws have set certain limits which cannot be exceeded but which are being rapidly approached by modern engines and turbines. All the refinements made in recent years by designers and manufacturers of steam engines and turbines may be completely overshadowed by improper operation in the boiler-room; for even when given the most up-to-date boiler plant, it is the easiest thing in the world to reduce its efficiency to that of a tea-kettle. It should, nevertheless, be borne in mind that enormous sums of money have been expended in boiler development for which the purchaser pays whether he takes advantage of it or not. The boiler-plant has no set standard, no self-maintained efficiency. This is dependent to a large extent on the skiil of the operators and on the interest taken by the owners.

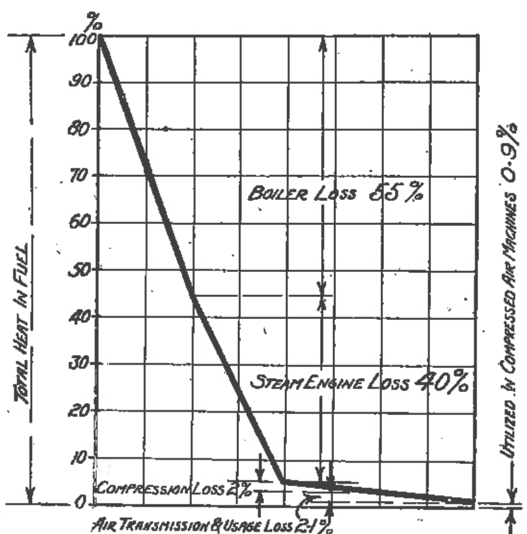


FIG. 3.

The losses shown in Figure 3 as attributable to the boiler-plant are shown in another manner in Figure 4. From this it can be seen that the most serious loss is that of the hot gases, or products of combustion, passing into the atmosphere through the stack. This is unavoidable to a certain extent in that the temperature of the gases cannot be cooled in an ordinary water tube boiler to within 100° F. of the steam temperature. Of

course, with a large amount of heating surface this can be bettered, but at too great a cost. There is a gas-fired boiler on the market now for which it is claimed that the difference in temperatures can be reduced to within 20° F., with a resultant high efficiency of over 90%. With the water-tube boiler, the heat in the waste gases is used to create draft, for without some means of moving the products of combustion and drawing the necessary air through the fuel bed, no combustion could take place. The intensity of the draft depends upon the height of the stack and the temperature of the waste gases, simply due to the difference in weight of a column of hot gases and cold air.

While this loss is unpreventable, it is possible, however, to reduce it materially. One pound of coal requires about 12 lb. of air to burn it completely. With perfect combustion, then, 13 lb. of hot gases are formed and the heat used in raising the temperature of the gases from that of the atmosphere in the boiler-room to that of the escaping gases is lost up the stack. It is not possible in practice to obtain perfect combustion and it is commonly found that about twice as much air as is necessary is used to burn the coal. Stated in another way, the more air used per pound of combustible material, the more waste there is going up the stack, the less the heat developed in the furnace; and as the rate of heat absorption by the boiler is proportional to the temperature of the gases and the fuel bed, the bad effects of too much air can readily be understood. Therefore, one of the most important points to watch in firing is the amount of air used per pound of coal, but as air is the only thing one gets for nothing it seems human to be greedy for it. Economise on the air and the coal will take care of itself. Unfortunately, Nova Scotian air is invisible and if there was only some simple method by which anyone or any outsider could observe this waste, the steam manufacturer would very quickly remedy matters. The only visible loss is black smoke, but this loss is generally not nearly as much as that due to excess air. Black smoke means incomplete combustion or air admitted in the wrong place, and it does not necessarily mean that there is a deficiency of air in the furnace. The underlying reasons for excess air are holes in the fire and leaky settings, and while it is not practicable to reduce the amount of air to the theoretical minimum, yet it is within the bounds of possibility to approach the ideal by fairly simple methods. Leaky settings can be detected readily by holding a lighted torch to the setting, and it would appear to be a comparatively simple matter to keep the fire bars well covered so that no holes appear in the fuel bed. If the fuel bed is too thick, a reduced boiler output is the result, due to the fact that the amount of coal burned is governed by the amount of air it is possible to draw through (no matter how much coal is shovelled on), and the increased resistance of the fuel bed reduces the supply of air and also forms troublesome clinker. The latter is due to the absence of cooling effect that a larger supply of air would give, although most clinker troubles are caused by needless stirring of the fuel bed so that fresh burning coal finds its way on to the fire bars; there it becomes mixed with the ash, which, as a result, is heated to the point of fusion. Light firing and often, replenishing only the thin spots, is the secret of good firing. A great deal more could be written on this subject but this would lengthen the paper unduly. With well-kept fires, a draft gauge set in the furnace is a great help in keeping tab on the condition of the fires, the variation in draft indicating extra holes in the fire or the formation of clinker.

Regarding the waste going up the stack, there are means of keeping check on this by waste-heat meters such as CO_2 recorders and pyrometers. The first named gives direct information as to the amount of waste gases and guess work is eliminated. It simple measures excess air in terms of carbon dioxide. It reveals to the manager at a glance what is happening without having to resort to examination of every detail, which he cannot, or has not time to do. It does away with guessing the qualities of the foremen and the efficiencies of the furnace. The pyrometer will show if the baffles are in good condition or if the boiler is free from soot. Finally, a steam-flow meter and a feed-water recorder are valuable instruments in learning

what returns are being obtained for the coal burnt. Unless a boiler plant is equipped with indicating instruments, it is impossible to learn what is happening merely by walking through the plant.

The other losses prevalent in boiler plants are due to unburnt fuel in the ashes, heat loss due to moisture in the coal, and moisture formed by the burning of hydrogen in the coal. incomplete combustion (generally in a furnace of faulty design), sooty tubes (soot is a splendid insulator), scale and dirt inside the tubes and radiation. (See Figure 4.)

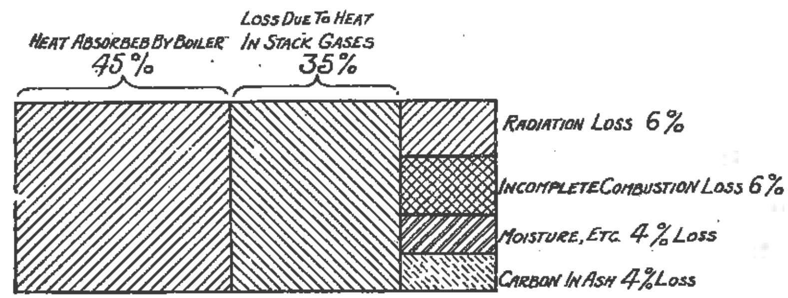


FIG. 4.

The loss due to unburnt fuel is easily detected and needs no further comment. The remedy is obvious. The heat lost through moisture is unpreventable. Incomplete combustion is usually the fault of the furnace or of insufficient combustion volume to ensure thorough mixture of gases. The loss due to sooty tubes may be considerable but is often over-rated. Mechanical soot blowers are preferable to hand blowers as a remedy; too little stack draft aggravates this trouble. There are several good blowers on the market. Scale and dirt inside the tubes cannot help the rate of heat absorption of the boiler and are very detrimental to the boiler. In short, the boilers must be kept clean to get good results.

Mention should be made of the value of a feed-water heater. A good heater is the most efficient heat engine available, and extracts more heat, in comparison, than the finest turbine ever built. The latent energy of the steam is given up to the incoming water, resulting in the condensation of the steam. This heat is lost in the condensing cooling water of a turbine plant. With a good heater one pound of exhaust steam will heat six pounds of water; about 11° rise in the temperature of the feed water represents a saving of 1% of the fuel used. As many collieries in this district have all their steam available as exhaust steam, some of the excess could be used to advantage in heating water for washing purposes.

The loss in steam transmission should be negligible if the lines are of proper size and well insulated. There is one case on record in Nova Scotia where fully 8% of the steam made was lost through uncovered pipes alone.

Dealing with air-compressors it is advisable that the air ends at least should be tested every month, as with dirty valve seats, etc., it is often found that during from 25% to 30% of the stroke (or the time the engine is running) no useful work is accomplished and the capacity of the machine is very much reduced with a consequent lowering of the mechanical efficiency.

The transmission loss for compressed air need not be very great although insufficient sizes of air lines will give an unwarranted drop in pressure, as much as a 35-lb. difference between surface and coal face being not uncommon in cases of this kind. The velocity in mains should not exceed 25 feet per second, and in the branches 50 feet per second. This will give good pressure at the face and the transmission losses will be small. By testing systematically the air consumption of each coal-cutting machine by meters, not only are wasteful machines detected but the pressure available at the face is noted on the gauge. On comparing the pressure with that shown by the recording gauge in the compressor room at the time taken, an idea will be obtained of how much pressure is being lost in transmission. Possibly an obstruction in the line, such as water pockets, etc., may be the cause, and pit props have been found in air lines before now. The saving of a few feet of air at the delivery point may not seem worth while investigating, but it should be remembered

that when the air reaches this stage it is very costly. Operators cannot tell, (and probably care less) if their machines are using the proper amount of air, and machines using double their rated amounts are not uncommon and will run without any outside indications. For the purposes of illustration, if the coal-cutting machines in use in Nova Scotia were taking 20% in excess of their amount (and this is very common to un-metered equipment), a compressor plant capable of supplying 16,000 cubic feet of air per minute, together with a boiler plant of some 2,500 boiler horse-power, involving a monthly expenditure of about $10,000 (exclusive of interest and depreciation) would be required to compensate for this condition alone.

In conclusion, while this paper was not intended to be unduly pessimistic, one would be pardoned for feeling a trifle disheartened when it is realized that less than one-hundredth of the fuel value may be, and is, utilized in a poor system. When firemen are paid to think more and work less and when up-to-date methods are introduced, then a greater return from that 500,000 tons of coal consumed for power purposes will be obtained.

THE CROWSNEST PASS COALFIELD

By Robert Strachan

Rocky Mountain Branch, Fernie, B.C., May, 1920

The first record of the occurrence of coal in the Crowsnest Pass is by a Mr. Phillips who, accompanied by a Mr. Collins, reached Michel creek in 1873 while in search of furs, and noted the outcrops of coal at this point. The following year Phillips, accompanied by three others, returned and spent some time on what is now called Morrissey creek (which was named after one of the party) and Coal creek, travelling over to Michel creek, and, as he describes it, finding "coal everywhere, nothing but coal." In 1875 the Government voted money to construct a trail through the pass, and in the Reports of the Geological Survey of 1880 and 1882 the existence of coal is mentioned, while the coal areas were approximately defined and examined in a preliminary way by Dr. Dawson in 1883.

In 1887 Wm. Fernie and Lieut.-Col. James Baker began prospecting for coal in this district and continued to do so for eight or nine years, when a syndicate was formed in Victoria to acquire their interests. Ten years later, in 1897, the Crowsnest Pass Coal Company was organized to secure control of the coal areas, and systematic development was commenced at Coal creek. In the following year the Canadian Pacific Railway commenced the extension of the railway from Macleod and two years later it reached Fernie, when shipments of coal commenced. Since then the production has increased steadily, and today the annual capacity of the operating mines is well over a million tons.

The extension of the railway to the Boundary district afforded an opportunity for the sale of coke to smelting works there, and in consequence coke ovens of the bee-hive type were built at the collieries, and, by the careful selection of coals, a very good metallurgical coke was produced. The Canadian Pacific Railway enables the coal and coke from the district to be supplied to the Canadian market; in 1904 direct access to United States' markets was afforded by the construction of a

branch of the Great Northern Railway from Rexford to connect with the Morrissey, Fernie and Michel Railway.

In 1897, the first colliery to be opened was that of Coal Creek. This is situated about five miles from Fernie on the stream of that name, a tributary of the Elk river from the east, occupying a comparatively deep valley cutting through the Cretaceous rocks at a point where the valley floor, rising, meets the easterly dipping coal measures.

The seams opened were:

No. 1	Seam averaging	about	20	ft.	in thickness.
No. 2	"	"	6	"	"
No. 5	"	"	12	"	"
No. "B"	"	"	5	"	"
No. 4	"	"	40	"	"

The strike of the coal is approximately at right angles to the valley, thus enabling tunnels to be driven in the seams on each side of the creek; and, as this point is approaching the centre of the basin, the seams dip at much lower angles (from 12 to 18 degrees) than at their outcrop along the escarpment of the Elk river.

A few years later, in 1900, the Michel and Erickson colliery was opened on Michel creek, which flows into the Elk river about 24 miles north of Fernie; at this point the creek cuts through the northern part of the coalfield, eroding the upper portion of the coal measures in a narrow valley. Here seven seams of coal have been exposed, four on the south side of the creek, and three on the north. On the south side there are:—

No. 3 Seam,	averaging	5	ft.	in thickness
Upper No. 2 Seam	"	11	"	"
No. 4 Seam	"	7	"	"
No. 5 Seam	"	7	"	"

On the north side there are:—

No. 7 Seam,	averaging	11	ft.	in thickness with a 2.5-ft. parting
No. 8 Seam	"	12	"	"
No. 9 Seam	"	10	"	"

At present only the upper No. 3 and No. 8 are being operated.

In 1901 Morrissey colliery was opened. It is situated on the creek of the same name about five miles from where it empties into the Elk river. Here nine seams varying from five to forty feet in thickness, have been worked at different times, but owing to serious outbursts, or blow-outs, of gas, operations were discontinued in 1910.

In 1908 the Hosmer Mines, Limited, opened the Hosmer colliery (eight miles north of Fernie) by driving two cross-cut tunnels through the coal measures at a point 600 feet above the level of the Canadian Pacific Railway, and intersecting 10 seams of coal as follows:—

No. 1 Seam—18 ft. thick
No. 2 Seam—12 ft. thick
No. 3 Seam—22 ft. thick
No. 4 Seam— 4 ft. thick
No. 5 Seam—18 ft. thick (with a 10-in. shale parting)
No. 6 Seam— 8 ft. thick
No. 7 Seam— 4 ft. thick
No. 8 Seam— 5 ft. thick
No. 9 Seam— 8 ft. thick
No. 10 Seam—large seam (thickness not determined).

The Nos. 9 and 10 seams at Hosmer are supposed to correspond with Nos. 2 and 1, respectively, at Coal Creek. The lowest seams first cut in the tunnel have easterly dips of about 65 degrees, but this flattens out from there on to the minimum of about 25 degrees. (In 1914 for some reason work at Hosmer was discontinued, and so far, no attempt has been made to re-open the colliery.)

In the same year, namely, 1908, the Corbin Coal and Coke Company commenced operations on an outlying section of the coalfield, situated about 17 miles due east of Fernie. From this area a short railway, fourteen miles long, termed the British Columbia Eastern, connects with the Canadian Pacific

at McGillivary. Operations have been conducted here at two distinct points: in the valley where the seams of coal are reached by tunnels along the strike of the same, and on the top of the hill, about one and a half miles south, where an open cut reveals a synclinal basin about 370 ft. in width, the thickness of the coal near the centre of the syncline having been proved to be over 100 ft. In the tunnel workings, the seams are nearly vertical, and vary greatly in thickness, from the minimum of 10 ft. to the maximum of 250 ft., and it is generally conceded that the difference may be due to monoclinal folding. The geological relationship between these two points has not yet been worked out, and it is possible that they are both on the same bed of coal. That the coalfield is a very valuable one can be conceded when we consider that since the commencement of mining operations 15½ million long tons, or almost 17½ million short tons of coal has been extracted,, while to supply the demand of the smelters, 3,812,629 tons of coke has been made and shipped. The following are typical analyses of the coal from some of the seams:—

Colliery	Seam or Mine	Moisture	Vol. Comb. Matter	Fixed Carbon	Ash
Coal Creek	No. 1	0.84	23.59	68.40	7.17
" "	No. 2	1.0	21.85	71.30	5.85
" "	No. 5	0.5	24.00	65.2	10.8
" "	No. "B"				
Michel	No. 3	1.01	20.95	71.00	7.04
"	Upper No. 3				
"	No. 8	1.50	22.10	68.50	7.90
Morrissey	No. 1	0.9	22.19	70.99	5.60
"	No. 4	0.82	11.73	71.50	15.75
Hosmer	No. 2	0.9	21.3	63.4	15.3

Mr. D. B. Dowling, in the "Coalfields of British Columbia," refers to the Crowsnest coalfield, situated immediately west of the summit of the Rocky Mountains on the Crowsnest Pass, as being the most important deposit of coal mined in the province of British Columbia. The field is described as a long triangle with its base to the south, its greatest length being about

35 miles and its greatest width 13 miles. The coal measures were originally deposited over an area of nearly 500 square miles, but, due to the erosion around the edges where the rocks are crumpled and folded and the penetration of the deeper valleys into the region, their area has been reduced to about 230 square miles. As a result of this erosion, the Kootenay and overlying strata form an elevated plateau-like area bordered by depressions occupied by older beds. The great dynamic forces pressing from either side (east and west) in the uplifting of the Rocky Mountains have buckled the field into a north and south synclinal trough, of which the uplifting western edges of Cretaceous rocks are fairly uniform at about 3,500 to 4,000 ft. above, and parallel to, the Elk river, distant about three or four miles. About half way up, the coal measures are found outcropping, with dips of from 20 to 40 degrees to the east, in the Kootenay formation of the Lower Cretaceous series. The Fernie shales which underlie the Kootenay formation, average about 2,600 ft. in thickness, and there appears to be some doubt whether these shales belong to the Lower Cretaceous or Jurassic. The discovery of several specimens of the fossil *productus* in the limestone underlying the Cretaceous series is fairly good evidence that in this part the upper members of the limestone series are definitely Carboniferous.

Sections taken near Morrissey.	Ft. Thick	Sections taken on Coal Creek.	Ft. Thick
Coal	10	Coal	10
Intervening beds	140	Intervening beds	60
Coal	36	Coal	30
Intervening beds	197	Intervening beds	42
Coal	6	Coal	6

Another section taken at South Fork, Michel Creek, is compared with one taken near Morrissey:

South Fork, Michel Creek.	Ft. Thick	Near Morrissey.	Ft. Thick
Coal with parting	7.3	Coal	7.0
Intervening beds	47.0	Intervening beds	35.0
Coal	13.0	Coal with parting	14.0
Intervening beds	115.1	Intervening beds	154.0
Coal	2.1	Coal	3.0
Intervening beds	112.7	Intervening beds	120.0
Coal	7.9	Coal	10.0
Intervening beds	11.7	Intervening beds	140.0
Coal with partings	25.2	Coal, upper part impure	36.0

The majority of the coal seams are found in the lower part of the coal measures, and measurements of natural sections taken at various points show: (1) at Morrissey 23 seams with a thickness of 216 ft. of coal in 3,676 ft. of strata; (2) at Fernie also 23 seams with 172 ft. of coal in 2,250 ft. of strata; and (3) at Sparwood a greater natural section shows in the lower section 23 seams with 173 ft. of coal in 2,050 ft. of strata, and in the upper section 43 ft. of coal in 24 seams in a thickness of 2,015 ft. of strata. The majority of these seams occur in the lower 2,000 ft. of measures and the sections show the seams to be fairly regular but with great difference in the thickness of the intervening strata or beds.

A great many faults traverse the field; in the area lying between the south branch of Michel creek and the Elk river north of Coal creek the measures do not long continue to hold the same regular form they exhibit at Coal creek itself. A few miles north of the creek, the transition from the steep dips at the front to the nearly horizontal position further back is more abrupt, and a short distance farther north becomes a steep break with more or less faulting. The result of this is that the coal measures are brought nearer to the surface and they are found outcropping on the side of a deep gash in the hills made by a small stream emptying into Michel creek. The bottom of the syncline is probably a short distance east of Michel station and the measures appear to rise gradually both to the north and south, with the lowest point of the basin to the south of the stream. Estimates of the amount of coal contained in the field have been made of about 22,590,000,000 tons actual reserves, and of about 20,160,000,000 tons probable reserves, or in other words, about 40,500,000,000 tons.

It was early apparent that the working of the coal seams would be attended with some trouble on account of gas, for in 1900 Mr. Archibald Dick, then Inspector of Mines, remarked in his annual report, "After the mine was idle for only one day the gas seemed to disappear, but when at work the gas is liberated so that the atmosphere of the mine is heavily charged." Three explosions in Coal creek mines and two in Michel have shown the dangers to be apprehended from this source. At Morrissey, outbursts or blow-outs of gas of a magnitude unsurpassed by few in the annals of mining, have occurred, and from 1917 similar outbursts have become rather frequent at Coal creek in one of the mines.

'Bumps' seem to be due principally to the immense weight resting on the coal seams, aggravated by probable weakness in the strata and the unstable condition of the mountains, which are not only unsupported on the side next to Coal creek, but are cut by small streams in the mountain side, creating a condition almost analogous to that of a lever supported at one end. Reports on these occurrences have already been made by Mr. W. Fleet Robertson, the Provincial Mineralogist, and Mr. G. S. Rice, Chief Mining Engineer for the Bureau of Mines, of the United States. The first indication (in 1907) of trouble with 'bumps' was in No. 2 mine, where a 6-ft. seam was being worked; it had a shale and coal bottom, part of which was taken up to allow room for haulage. In 1916, the workings of No. 1 East mine seams had reached, or just crossed, the portion of No. 2 mine where the 'bumps' had been experienced in 1907-8, when a series of such 'bumps' occurred. These lasted for about three days, destroying the greater portion of the mine and killing one of the workmen. The No. 1 East Mine seam averages 20 ft. in thickness but only the upper 10 ft. is extracted, about three feet of coal and two feet of soft shale being left next the rock roof, the shale or rations forming the roof except where it is taken down for permanent roadways. The No. 2 mine, one of the oldest in the district, has been worked with very little regard to regularity and quite a number of the pillars had been extracted; but No. 1 East had been laid out very regularly with

large pillars, and probably not more than 25% of the coal was being extracted in this first operation. These 'bumps' have so far only occurred in such mines or portions of the mine where the overhead cover is at its thickest, or probably over 1,800 ft.; and, to be exact, where 'bumps' have occurred in both No. 2 and No. 1 East mines, the strata measure about 2,500 ft. It is usual in determining the amount of weight on pillars to allow 13 cu. ft. to the ton, so we have a pressure on the pillars at the points where the 'bumps' took place of about 192 tons under normal conditions.

The outbursts or blow-outs of gas at Morrissey and Coal creek were described by Mr. G. S. Rice in his report covering the occurrence of 'bumps'; and Mr. James Ashworth has also written various papers relating to these troubles. The following description is therefore very brief.

The No. 1 seam at Morrissey varies from 14 to 40 ft. in thickness, with a pitch of about 65 degrees, and only the upper 10 ft. of coal was being extracted. In 1903 the slope at this mine had only been driven in about 2,000 ft. when an outburst of gas occurred, displacing about 1,500 tons of coal, mostly in the form of very fine dust, and giving off enormous quantities of methane. When the debris was cleared away, a large cavity, 10 ft. x 8 ft. and 110 ft. deep was found ahead of the working face. The following year witnessed two more outbursts in the same mine, the first of which dislodged about 800 tons, and the second was estimated to have dislodged about 3,500 tons and caused an outflow of from two to five million cubic feet of gas. These occurrences were responsible for a loss of eighteen lives and caused the management grave anxiety. In view of this the colliery was closed.

Three years later an attempt was made to open up one of the lower seams, No. 7, and work was carried on for about two years. This seam, No. 7, averaged about 35 ft. in thickness and dipped at an angle of 56 degrees. About 16 ft. of the coal next to the floor was in course of extraction. The upper portion of this coal was fairly soft. In 1907, four blow-outs, or outbursts, occurred, displacing in each case between 15 and 75 tons of coal, accompanied by a large outflow of methane. This probably influenced the management in again closing the colliery and no attempt has since been made to re-open it.

Outbursts of this kind at Coal creek have so far been confined to No. 1 East Mine, which is situated on the south side of the creek, and extending towards the Morrissey portion of the field. The place where these outbursts have occurred is not in the portion nearest Morrissey, but rather at a part where the seam passes through a deep gash cut by a stream in the mountain side. In this mine during the last two years several of these outbursts have occurred, displacing from 40 to 140 tons of coal and giving off large quantities of methane. The following is an analysis of a sample of this gas taken seven hours after the blow-out occurred:

Chemical Analysis

Carbon dioxide	2.01
Oxygen	0.5
Methane	90.5
Nitrogen	6.9
Air	2.4
Fire damp	90.5
Black damp	7.1

The subject of the outflow of methane from, or due to the working of, the coal has been very comprehensively dealt with by Mr. Thomas Graham, late Chief Inspector of Mines for the province, in a paper read before the United States Inspectors of Mines Institute. In this paper Mr. Graham presents some remarkable figures respecting the volume of gas given off per ton of coal mined; and the following table gives a comparison between Mr. Graham's figures, which were recorded in 1916, and the average of these returns for 1918 and 1919.

COLLIERY	MINE	Cubic feet of methane produced per ton of coal mined		
		1916	1918	1919
Coal Creek	No. 1 East	2.789	5.389	4.925
"	No. 3	3.740	3.010	2.544
"	No. 1 South	1.598	1.917	1.796
"	No. 1 North	451	585	550
"	No. 2	1.100	1.716	1.491
"	No. "B" North	1.090	2.326	2.720
Michel	No. 8	87	145	130
"	No. 3 East	3.035	3.142	4.800
"	No. 3 Old	5.500	5.187	2.253
Corbin	No. 4	287	360	245
	Average	1.967	2.377	2.145

NOTE.—During 1916 and part of 1918. the No. 3 seam was worked at No. 3 Colliery; but in 1919 the Upper No. 3 seam was being mined, or the same seam as No. 3 East mine.

By way of comparison it may be stated that the gaseous mines of Wilkes-Barre, Penn., show from 2.000 to 3.000 cu. ft. of gas per ton; those of Southern Illinois from .50 to 260 cu. ft. of gas per ton; the fiery Austrian mines from 883 to 1,377 cu. ft. of gas per ton; and those of Saarbrucken basin from 1.65 to 2.160 cu. ft. of gas per ton.

The occurrences of these gases are in the coal or adjacent strata, and in the case of the No. 1 East mine, a large amount of gas is given off when the shale strata, which are used as a roof, give way.

The method of dealing with this flow of gas is to carry it away by ventilation, and some time ago the operating company agreed to keep the percentage of methane in the general body of the mine air below 2.5%, or failing that to withdraw the workmen. Further, it was agreed that if the Wolf safety lamp should show a quarter-inch gas cap, the same course would be followed. This agreement has been faithfully carried out. The Burrell gas detector is commonly used, the fireboss making a test, which is noted in the daily report, at least once during each shift, and by this means with the use of sample bottles in which samples of the various returns are taken and sent to the Department of Mines at Ottawa for analysis, fairly good conditions have been maintained during the past two years, as the following table, giving the average percentage of gas in the air, will show:

COLLIERY	MINE	DISTRICT	Percent. of Methane in return Airways	
			1918	1919
Coal Creek	No. 1 East...	Main return...	1.3	1.07
"	No. 1 East....	South split....	2.0	1.63
"	No. 1 East....	North split....	1.38	1.08
"	No. 1 South...	Main return...	1.71	1.34
"	No. 1 North..	Main return...	0.5	0.44
"	No. "B".....	Main return...	1.2	0.91
"	No. 2........	Main return...	0.7	0.7
"	No. 3........	Main return...	0.9	1.23
Michel....	No. 3 East....	Main return...	1.10	1.2
"	No. 3 East....	No. 6, E. split..	0.7	1.04
"	No. 3 East....	West split.....	1.66	1.61
"	No. 3 Old....	Main return...	0.69	0.71
"	No. 8........	Main return...	0.10	0.11
Corbin....	No. 4........	Main return...	0.33	0.28

In this connection it is worthy of remark that since the introduction of the Burrell gas detector and the maintenance of the stricter watch on the air-currents, very much less trouble has been experienced with accumulations of gas at the working face. An idea of the conditions commonly existing when explosive gas is reported in a working face, is given by the following

analyses of three samples of explosive gas taken for analysis. The samples were all taken in the same room or stall at the working face. The first sample was obtained at the point where explosive gas was indicated by the Wolf safety lamp, the second sample at a point one foot vertically above, and the third from a point two feet vertically above the first.

Chemical Analysis				Technical Analysis		
CO_2	Oxy.	CH_4	Nitr.	Air.	Fire Damp.	Black Damp.
0.7	12.5	25.1	61.7	59.7	25.1	15.2
0.9	7.7	56.6	34.8	36.8	56.6	6.6
0.9	8.1	57.3	33.7	38.7	57.3	4.0

In 1919 4,495,680 cu. ft. of methane was discharged into the atmosphere every 24 hours from the Coal creek colliery; 1,877,761 cu. ft. from Michel; and 73,440 cu. ft. from Corbin; or a total of 6,446,881 cu. ft. of methane; and if we assume the specific gravity of methane to be 0.55, barometer 25.5 inches, thermometer 50 degrees Fah., the result obtained is 117.5 short tons.

The outbursts of gas in many cases seem to come, if not from the roof coal, at least from the shales in the coal seam, the causes so far being unknown. Of course, it is known that all the gases are the result of the coal formation, most of them being occluded in the coal or adjacent strata. In Belgium, where outbursts on a similar scale have occurred, it is explained that the coal measures have been subjected to very heavy horizontal pressure, which has resulted in the formations slipping over and past one another, the irregular lines of fracture producing pockets, in which the gas accumulates, and in which were large quantities of dust resulting from the grinding of the coal by the movements of the rock. There is no doubt but that the Crowsnest Pass measures have also been subjected to heavy horizontal pressures and that they have also been subjected to movements, judging by the position of the slips that occur in the coal seams; there are no regular lines of cleavage, these running in all directions. Mr. James Ashworth, who has devoted considerable time to the study of these problems in this field, inclines to the theory that these gases exist in liquid form and that the outbursts were attributable to the volatilization of light oils or spirits that had been absorbed in patches of the soft coal, and on being released by the removal or thinning of the surrounding coal, volatilized with accompanying violence. As layer after layer of the saturated coal was blown off, the dust was carried away by the gas, and the outburst continued until the oil-saturated mass was blown away and the oil or spirits volatilized.

The occurrences of these outbursts in the Coal creek No. 1 East mine have been carefully watched during the past two years, and lately a method of drilling suspected places has been followed. In any place where the coal suddenly becomes tough, or where shale begins to show in the top part of the seam, or in a roll, it has become the custom to drill either one or two holes ahead. These holes are usually about 20 feet long and two inches in diameter, and in some cases the gas had been given off so freely as to extinguish a safety lamp four feet away. Such drill holes have, so far, given good results in the places where they have been drilled; but lately one of these outbursts has occurred in a cross-cut, displacing about 150 tons of coal and forming large quantities of gas, thereby seeming to indicate the necessity of drilling every working place.

The occurrence of these outbursts is indicated by a long roll-like thunder that seems to travel across the roof, giving to all within a certain distance the impression that it is exactly above their heads. Where it will blow out is problematical; it would almost appear as if it was seeking the weakest point; and the writer believes it generally finds that point.

The sound seems to travel like a wave, and suggests almost that the strata above the coal were giving way or yielding, leaving between the beds a space in which the gas can travel. This creates in the writer's mind the belief that the gas exists in pockets, which may be connected by small passages, and when the gas in the first or any of these pockets is liberated, the first wave causes the roll; and the writer firmly believes that the

strata immediately above the coal that is being mined are depressed when this first rush of high-pressure gas occurs, the scour of which causes the discharge of coal; and that, after this high pressure is reduced, the feeders blow with a gradually reducing pressure for some time until the pressure in the pockets becomes equal to that of the atmosphere or ventilating current. The method of drilling should reduce this danger to the minimum; and, of course, it may be necessary in the future to use longer holes of greater diameter and even to drill every working place.

The pressures experienced in drilling the holes have been rather disappointing, as the following results, obtained during the investigations of Mr. Rice, show:—

	Boreholes depths			Temperature Bottom 24 ft. hole	Mine Air
No. 1 East Coal Creek.	8 ft.	16 ft.	24 ft.	Deg.F.	Deg.F.
18 Room 10 East entry on solid coal on lower rib....	3	0.5	13.5	46	40
22 Room 10 East entry, in solid coal on lower rib....	0.5	7.0	18.0	50	48
No. 3 Mine, counter to main level in solid coal on lower rib............	1.5	2.0	2.0	65	60

The coal was commonly very soft, with a lack of cleavage.

To summarize, as the writer has already mentioned, no 'bumps' have been experienced where the overhead cover is less than 1,500 feet, representing a weight of about 115 tons per square foot under normal conditions, so that this would indicate the necessity of leaving very large pillars where the pillar-and-stall method is being used. Previous to the 'bump' in No. 1 East in 1916, only about 25% of the coal was extracted. When this district is re-opened (and it will give an idea of the destruction wrought when the writer states that it is only being re-opened now), only about 15% of the coal will be extracted. Narrow openings with large pillars seem to be the only solution of this problem, during such time as the workings are under this heavy load.

In coping with the gas outflow, the highest average daily outflow during the year at Coal creek was 1.63% and at Michel 1.61%, which is well below the standard as set.

The treatment of dust in mines such as these is also an important matter, and is carried out by watering in all dusty roadways once in every twenty-four hours. The water is conveyed by means of water pipes laid along all the important roadways and as near as is convenient to the working faces, and in watering the roadway, sides, roof and the working face an ordinary garden hose is used. On many of the haulage roads there is also installed a system of sprays, throwing a fine column of water into the air-current, which carries it along. On many of the haulage roadways a combination of watering and non-inflammable dust is used, second-burnt boiler flue dust and ashes being utilized for this purpose.

Edison electric lamps are in general use by the workmen, only the officials using Wolf safety lamps for the purpose of detecting inflammable gases; and while taking from the miner his only means of detecting dangerous gases may not meet with the approval of every mining man, the substitution of electric lamps has undoubtedly proved beneficial in these mines.

Special rules cover the method of timbering, the method for each mine or district being posted at the entrance to the mine; but this does not in any way prevent the miner putting in more timber should he consider it necessary for safety.

COAL TRANSPORTATION

BY M. A. MCINNIS

Annual General Meeting, Toronto, March, 1920.

The transportation of coal from the United States into Canada is such a large factor in its cost to the consumer that our facilities for handling coal and the future adjustments of freight rates will largely govern the distribution and cost of this commodity.

We, in Canada, are entirely dependent on the United States for anthracite (our consumption of which last year reached 4,782,000 tons). In 1919, we also imported from that country, bituminous coal to the extent of 17,275,000 tons, or over one-half of our requirements. At our present rate of importation of anthracite and bituminous combined, totalling 22,000,000 tons yearly, or 60,000 tons daily, it is to be expected that we shall, from time to time, be confronted with abnormal conditions as regards supply, and for these we should be prepared. The total coal consumption of Canada last year was 35,000,000 tons, or about four and a half tons per capita. When our population reaches 10,000,000—a number we may expect by 1925—the increase in coal consumption will be 9,000,000 tons, making a total of 44,000,000 tons yearly Unless, in the meantime, our home production can be materially increased, we will require by 1925 to import about 27,000,000 tons, or 74,000 tons daily Of our total coal imports, 42% is brought in by water, 46% by rail, and 12% by ferries. For handling the traffic by water, we have at present available:

	Storage capacity	Discharging rate per hour
On River St. Lawrence, 44 plants	1,750,000	7,800
On Lake Ontario, 9 plants	29,000	240
On Lake Erie, 3 plants	5,500	300
On Lake Huron and Georgian Bay, 52 plants	815,000	4,800
On Lake Superior, 11 plants	2,800,000	3,600
At Sault Ste. Marie, 4 plants	715,000	1,400

The 12% of our imports handled by ferries to two points on Lake Ontario and four on Lake Erie amounts to about 2,600,000 tons yearly.

Saskatchewan, Alberta, and British Columbia do not require to import fuel, their total imports from the United States being less than 25,000 tons yearly. The outputs of these three provinces reach nearly 8,000,000 tons. British Columbia exports about 1,000,000 tons of its production each year. The enormous coal resources of these three provinces—estimated at 1,200 billion tons—are just being realized, and will contribute for many generations to the progress of Western Canada. The matter of the transportation of fuel in these provinces need not concern us.

The territory at the head of the lakes, including Manitoba, imports 3,000,000 tons annually, of which 2,000,000 tons are taken by the railways. The greater part of this coal is carried by rail from the Pennsylvanian mines to Buffalo and other lake ports, then by water to Fort William and Fort Arthur, where the Canadian Pacific and Canadian Northern railways have large discharging and storage plants. It is here loaded into cars and shipped as far west as Winnipeg and other points in Manitoba. Although this territory is about 1,500 miles distant from the mines, the water haul of 800 miles, with its comparatively low freight rate, keeps the price of coal in Winnipeg within $1.50 a ton of the price in Quebec and Ontario.

Our greatest difficulty in importing coal by rail is occasioned at Niagara, or what is known as the Niagara gateway. The Canadian railways at this point are unable to accept coal at the same freight rate, and in the quantities delivered by the American roads. For the greater part of the year our railways can handle only a hundred cars a day. Owing to the comparatively short haul from the Pennsylvania mines to the cities and towns of southwestern Ontario, or the territory enclosed by a line drawn from Niagara to Toronto, and west to the shores of Lake Huron, these points have, in the past, depended on coal brought in by rail rather than on that brought by water transportation via Lakes Erie and Ontario. Toronto, Hamilton, and other points in this area should eventually take more coal brought in over the waterways, and for this adequate unloading and storage facilities will have to be provided. Last year, central Cntario imported 10,000,000 tons, of which the railways consumed 6,000,000 tons.

The bringing of coal into eastern Ontario and Quebec has but seasonable difficulties. During the summer season the larger coal importers and water transportation companies have fleets of coal carriers that ply between American lake ports and Canadian ports, the latter extending from Toronto to the city of Quebec. In this way about one-half of the requirements of this section is supplied; it includes a certain quantity of coal that is re-shipped to inland points by rail. In pre-war times, Nova Scotia shipped by water to the St. Lawrence market about 2,000,000 tons a year. Last year the shipments amounted to but 300,000 tons. With the increased demand in Eastern Canada for fuel for local industries, for the bunkering of ocean going ships, and with the export trade which is now being developed, this territory will have to depend largely on importations from the United States. With our splendid water facilities and the convenient connections of Canadian railways with the American railway systems in normal times, this section of the country is well enabled to provide ample transportation for the fuel imported to meet its requirements.

The coal requirements of the Maritime provinces present no serious problems. Last year, Nova Scotia, New Brunswick, and Prince Edward Island imported from the United States 175,000 tons of anthracite, nearly all of it being brought in bottoms. These provinces produce 6,500,000 tons of bituminous coal yearly. We often hear the wish expressed that the great coal deposits of Nova Scotia could have a rail outlet to Ontario and Quebec. As mentioned at the outset of this paper, transportation costs are generally the governing factor in the selling price of coal, and they are often the real controlling factor in coal production. During the writer's boyhood days in Nova Scotia, when common labour was paid at the rate of a dollar a day of ten hours, it was conclusively proved that the cost of hauling coal on the Intercolonial Railway was three-tenths of a cent per ton-mile. It is a thousand miles from Sydney to Montreal, so that at the time referred to the cost of hauling coal this distance would be three dollars a ton. As the best price obtainable for coal in Montreal at that time was $2.75, clearly its carriage to market by rail was not profitable. The coal trade of Nova Scotia advanced in the early 'nineties' in exact proportion to the improvement of the transportation facilities of the St. Lawrence. With steamers of 30-feet draft and 10,000 tons capacity, coal could be landed in Montreal at a freight charge of 50 cents a ton. As we improved our transportation and handling facilities at St. Lawrence and Maritime ports, our coal trade increased. The market was there in every case, and the coal was there, but its transportation was the weak link in the chain. Some day, the Government of the country may find a solution for this problem, and devise means of assisting the coal mines of the East in placing their product as far west as Toronto. For the present, with our many difficulties in other directions, this one must be held in abeyance.

The foregoing remarks emphasize the importance of our transportation systems as related to the insurance of a regular supply of fuel for our commercial and domestic needs. That we are not making the fullest use of our different means of transportation, and of the railways in particular, was brought forcibly to our attention during the strike of bituminous coal miners in the United States in November last. One of the most important results following that situation has been a movement to devise a more uniform distribution of bituminous coal during the year. It has been found too expensive to stock coal at the mines, as the cost of re-handling and degradation would add so much to the mine price that it would offset any advantage to be derived therefrom. This means that unless mines are supplied with orders for regular shipments throughout the year they have to close from time to time.

The alternative to this is for the buyer to store coal at the place where it is to be consumed. This is just what buyers

of bituminous coal, in the past, have not been doing, and to a great extent they cannot be blamed for this. Mine prices and freight rates have been practically the same throughout the year; and the buyer would naturally say, "Why should I store and pay for coal in May when I can do as well by buying in September?" What this means to the railways is clear. Tonnage that might have been hauled during the summer months when traffic is not so heavy, and operating costs and general conditions are more favourable, is withheld to be thrown on the railways during the fall and winter months when our large supplies of grain and other commodities have to be moved, and when snow and frost bring the efficiency of our rolling stock and terminals to its lowest point. The remedy lies in educating the buyer to the fact that it is to his advantage to take delivery of his coal as early as possible during spring and summer, and it is to this end that the movement referred to is working.

Similar conditions prevailed in the anthracite industry up to a few years ago, and were remedied by a sliding scale, the price being advanced 10 cents a ton from the base or April price until September, thus giving a very considerable inducement to the buyer to take delivery when shipping conditions at the mines and on the railways are at their best. That the buyer, always alive to his interests, took advantage of this sliding scale and placed his orders and took delivery early, can be seen by the records of mining and transportation companies. April, May, and June, which formerly showed the smallest tonnage during the year, now show the largest; and coal that was formerly moved from September to March is now safely stored away by the customer in early summer. The railways have also been benefited by transporting this tonnage when it can best be handled. Let the bituminous coal operator follow the example of the anthracite operators by giving some concession to the early buyer and let the railways quote lower freight rates in summer, and the consumer will quickly take advantage thereof, and will provide storage room where the coal is used, and where the cost of rehandling will be small.

By storing coal early in different parts of the country, we could diminish to a great extent the danger of a coal famine and its effects on our industries, avoid high prices caused by shortage, and without laying a mile of extra track greatly increase the usefulness of our present systems of transportation by letting them carry the bulk of our coal when it can be most easily handled. We would also get the maximum service from every railway car as unloading is done more quickly in summer than in winter; the tying up of junction points and yards by the coal cars that accumulate during the winter months would also be lessened. On account of our industries and our climate, we know that we require millions of tons of coal, and if we begin obtaining our supply on the first fine day in spring and continue throughout the summer, our transportation facilities will then be used to the best advantage, and will be left free later on for other necessary work which cannot be done during the summer season.

CANADA'S COAL SUPPLY

By F. W. Gray

Annual General Meeting, Toronto, March, 1920.

The political division of North America, as it finally evolved from the conflict of races and the divergent search for an identical ideal by two branches of the English-speaking peoples, bore no considered relation to the balancing of the mineral resources of Canada and the United States; and, in so far as coal supply is concerned, the boundary line was fixed before the national importance of coal in peace and in war was realized, and in ignorance of the coal resources of what used to be known as the Far West, and is now known as the Canadian provinces of Alberta and British Columbia.

If no national issues had arisen, and North America had developed its resources as one nation, then in the East the coal-fields of Nova Scotia would have supplied the Atlantic seaboard with bituminous coal; British Columbia and Alberta would have supplied the Pacific seaboard and the north-western United States, and the central territories would be supplied entirely from the great central coalfield of Pennsylvania and the adjoining coal-yielding states. This is the natural scheme of distribution. Under such circumstances, however, it is certain that the territory which is now included within our own borders would not have reached so advanced a development as is the case, for the independent impulse of our own nationality would have been absent in the north, and industry would have concentrated itself farther south and nearer the great central coalfield. Also, it may be surmised, the coal production of Nova Scotia would have been upon a much larger scale than it is; while Sydney would have been of greater importance and Montreal of lesser importance than is the case to-day.

But the national issue did arise. Canada is a nation, so acclaimed and recognized in the councils of the world powers, and although the boundary line between ourselves and our good friends in the United States has certain disadvantages to ourselves, we must even make the best of accomplished facts.

Our unevenly distributed and deficient coal resources, and to a large extent also their backward state of development, are a consequence of this country's determination to be a nation within the British Empire. We have desired national independence, and have achieved it, and as our coal problem is an outcome and a concomitant of this desire and achievement, it becomes a principal duty of Canadians to work for the solution of our most pressing internal problem—the country's coal supply.

It is necessary to state these dominating considerations in order to emphasize that our coal problem is not altogether economic or geographical, but is primarily associated with Canada's national independence and defence. It is with this idea, taking precedence of purely commercial considerations, that this presentation of the question is submitted.

North America is favoured above the nations of Europe in having a supply of anthracite—a most desirable fuel, more especially for congested centres of population, because of its smokeless character and great heat value.

Unfortunately, Canada has no anthracite, so far as is known, with the exception of some anthracite (metamorphosed coals of relatively small tonnage) in the West. Therefore, if we use anthracite it must be imported.

Large parts of Canada use bituminous coal and have never found it necessary to import anthracite. In many parts of Canada the burning of anthracite is not understood, and all grates and furnaces are adapted for the burning of bituminous coal. This being the case, and seeing that Europe gets along with bituminous coal, it can hardly be argued that anthracite is indispensable in those districts of Canada that can be supplied with bituminous coal from Canadian mines, and it follows that anthracite, under such circumstances, no matter how desirable, is a luxury.

There is also the further consideration that men can always do without that which they cannot get, and anthracite will shortly be a luxury for the rich only, as it will steadily increase in cost as it decreases in quantity. Old anthracite mines are today being worked over for what was left by a more opulent generation, and anthracite seams of under two feet in thickness are being mined, facts which tell more eloquently than figures the impending scarcity of anthracite.

Assuming, therefore, that bituminous coal can entirely replace anthracite in Canada, we have only to consider over what extent the bituminous coals we have can be distributed, or, how we can extend the zones of distribution of Nova Scotian and Western coal so that they may approach and, if possible, meet.

So far as Canada, west of Fort William is concerned, it surely can be equally well supplied with bituminous coal from the western mines in Canada as with bituminous coal brought from Pennsylvania. Transportation distances do not enter into the question in the same grave manner as they affect Nova Scotian coal.

West of the longitude of Lake Superior, there is as much bituminous coal in the province of Alberta alone as in the remainder of the western half of North America.

Canada has not yet apprehended all the implications of the vast concentration of coal, and probably oil also, that exists in Alberta, and there is no compelling reason why the zone of distribution and use of Alberta bituminous coal should not be as extended as that of Pennsylvania and West Virginia. West of Fort William, Canada is more than capable of providing itself with all possible requirements of fuel.

There remains to consider the possible radius of distribution of the coal of Nova Scotia; but, first, something should be said as to the extent of the maritime coal deposits and the costs of mining them.

The coalfields of Nova Scotia, while they are not relatively large, forming as they do only one per cent. of Canada's coal resources, have never been worked to full advantage because of divided interest and scattered operation.

The consolidation of operation that followed the formation of the Dominion Coal Company was the salvation of the Sydney field, but, unfortunately, consolidation did not go far enough to ensure the maximum cheapness of production that it only can make possible. Sporadic, unco-ordinated, haphazard, and, in some instances, unwise operation of the coal deposits of Nova Scotia, have conspired to prevent a healthy growth in the annual production of this province. One who (in, say, 1907) had anticipated that the coal production of Nova Scotia would be at the rate of ten million tons per annum by 1920, could not have been regarded as unduly optimistic. Indeed, the objective of the Dominion Coal Company alone was at that time seven million tons annually, as those who refer to the late Mr. James Ross' remarks on this matter may confirm for themselves. The disappointingly small production of Nova Scotia during the past six years is chiefly a result of the war, and in that respect is a passing incident, but underlying, and altogether apart from the temporary effects of war, coal production in Nova Scotia has shown a recessive rather than an advancing tendency. What is the reason for this lack of vigor in the maritime coal industry?

Without attempting to excuse the faults of operation that have hindered coal production in Nova Scotia, it may be answered that the non-progressive character of the industry is due to a general lack of encouragement on the part of the railways and large purchasing interests in Canada, and also to the failure of governments in successive administrations to understand the paramount influence of coal supply on financial, military, and naval security.

Coal must always cost relatively more to mine in Nova Scotia than it does in the uniquely favored deposits of the United States; but a considerable part of the abnormally high mining costs in Nova Scotia at the present time is a result of too small

a production of coal in relation to the capital invested in mine properties and transportation equipment. Nothing can so effectively lower the unit costs of production in Nova Scotia as an increase in the output of coal. The coal companies there possess equipment sufficient to handle from two to three million tons annually of additional coal so far as transportation and marketing facilities are concerned. Given a sufficient expenditure and the necessary lapse of time to open new and extend existing collieries there is no reason why Nova Scotia should not produce twice its present output of coal. Such a programme is, however, only possible through the thorough-going consolidation of the operating coal companies, unification and concentration of direction, and very large capital expenditures on new mines and transportation equipment. Before investors can be induced to undertake the heavy commitments indicated, there must be a change in the attitude of the public and the railways towards the coal-trade. Mr. C. A. Magrath, the Fuel Controller. in his Final Report, suggested that the railway companies should give contracts for their coal supply for a term of years, at cost, plus a fair percentage of profit, provided the coal companies made the necessary expenditure to equip and maintain properties with all appliances to enable production to be carried on at a minimum of cost. There is much to be said for this suggestion. It should be obvious that if in times of plenty our Canadian railways choose to starve our domestic coal mines by buying coal in the United States or by demanding that the domestic producers meet United States competition even though that involve a profitless transaction or an actual loss to the Canadian producer, our coal trade must live a precarious life and will always be unready to meet the national emergency which may at any moment arise through political, social, or diplomatic occurrences, or by reason of physical hinderances.

Canada cannot be administered as a successful economic whole if we ignore the obligations of nationality and insist on buying goods in the cheapest market merely because they are cheap. That way leads to loss of independence and national disintegration.

The apathy of public opinion, if not actual hostility, towards the struggling coal trade of Nova Scotia is not less effective because it is based on ignorance and is against the best interests of Canada, for not only has it discouraged the expansion of the known coalfields, but it has deterred the search for the hidden coalfields, the existence of which is much more than a presumption. It would be entirely incorrect if we were to assume that the known coal deposits of Nova Scotia comprise the whole of the coal resources of that province, and here, again, there is reason to complain of lack of interest on the part of our governments, for no part of Canada has been so neglected during the past thirty years in the matter of geological exploration and mapping as Nova Scotia. The Springhill coalfield, which has an unknown, but extremely probable southward extension, is a case in point. The port of Parrsboro, which now serves the Springhill coalfield, as at present defined, is distant by water only some 90 miles from St. John, N.B., which latter by the direct line of the Canadian Pacific Railway is about 380 miles from Montreal. There is nothing insuperable in sending coal from this field to Montreal even by rail. Much longer hauls are made from the mines to the great cities in the United States. There is, however, no necessity to send coal by rail. It has in the past gone from Nova Scotia to Montreal by water at the rate of two million tons in the season of navigation, and could be sent in very much greater quantity by providing additional transportation equipment.

The feasibility of sending coal by water from Nova Scotia to Montreal has already been demonstrated. Of very great present interest and importance is the question of what can be done to cover the gap between Montreal and Fort William that is now entirely dependent upon United States coal. The cheapness of transportation from the United States central coalfield to the Great Lakes and the adjoining territories arises from a combination of water transport and a preferred inland freight rate from the mines to the Great Lakes ports. The carriage of coal to Canada gives an outward load for the cars carrying iron ore from the Lake Superior ranges to Pittsburgh, which otherwise would make the outward journey in an empty condition. From such points as Ashtabula and Cleveland, the transportation of coal to Canadian ports is cheaply effected by the water routes.

Apparently the only way by which the radius of distribution of Nova Scotia coal can be greatly extended east of (say) the eastern extremity of Lake Ontario, is by deepening the St. Lawrence channel so as to give access to ocean-going vessels to the Great Lakes. In such event, Nova Scotia coal could compete on fairly even terms, so far as cost of transportation is concerned, with United States coal, as the all-water route from Nova Scotian ports to the point of unloading in a Great Lakes port would offset the preferred rail rate from the United States mines to the point of transhipment on the Great Lakes. This project is under investigation. So far, all the protests that have been made against the project are such that, if conversely applied, would constitute arguments for its carrying out, having regard to Canada's interests. It may be submitted that if the project is pronounced feasible, it offers to Canada the opportunity to become thoroughly self-supplying and self-contained in bituminous coal supply. By affording to Nova Scotia a cheap water-route for coal shipments, the coal miners there would be able to so enlarge outputs as effectively to reduce costs of production, and soft coal from Nova Scotia could be shipped far enough west to span the country and meet Canadian soft coal shipped from the western mines.

The deepening of the St. Lawrence waterway is, however, not an immediate possibility, while the necessity to make Canada more independent in bituminous coal supply is, indeed, a most immediate urgency. What is feasible in the enlargement of distribution of Nova Scotian coal to-day? We can at least get back to the pre-war shipments to St. Lawrence ports of some two million tons annually. Further, the same factors of increase in the cost of coal production have been at work in the United States also. Moreover, there is encouragement in knowing that the Canadian people have awakened to some extent to the serious handicap we suffer from such entire dependence on the United States for coal, the danger of dislocation of our business; the threat of discomfort and physical danger that are always impending whenever interruptions to our coal supply occur. These new conditions suggest that an extension of the pre-war radius of distribution for Nova Scotian coal may be possible at the present time if energetic effort is made by the operators to recover and extend the St. Lawrence markets.

The present moment offers an opportunity to the coal interests of Nova Scotia, and the transportation interests of Eastern Canada to work together to secure the future permanency of the coal trade of Nova Scotia, which, whether they appreciate it or not, is something on which the railways, the public and the Government of Canada are equally interested with the coal operators and the mining population.

The equipment of the Nova Scotian collieries is modern, and, apart from the duplication inseparable from divided interests, no grave criticism can be made of the technical or business management of the operating properties, but some changes will be necessary before the most efficient production is possible. In particular, the present system of single shifts will have to be replaced by multiple shifts. The present practice of working the collieries for only eight hours in each twenty-four hours, often for only five days a week, does not permit of full returns from the capital invested, or the extent of under-ground territory developed.

To sum up these opinions—which it may again be emphasized are all dependent upon the assumption that Canada can only be politically independent in so far as she controls and supplies her own bituminous coal—it would appear that to effect the necessary increase in the coal output of Nova Scotia, two things are chiefly required, namely, a unified control of the operations of mining, transportation, and sales, and the recognition by purchasing interests, for their own future welfare and protection, of the necessity to buy Canadian-mined coal, and to pay a just price for it.

The writer will not attempt to discuss what relief we can obtain from utilization of our peat-bogs and hydro-electric reserves, for these have been ably dealt with by specialists, and even their completest utilization can only be in the nature of a palliation of the fuel problem, and can never constitute a remedy.

With regard to bituminous coal supply, we may conclude that the problem is not so much one of a source of supply in Canada as it is one of deficient and difficult transportation. Canada has sufficient bituminous coal for its own needs, but the country has never undertaken to become thoroughly self-supporting from a conviction that this was not only desirable, but actually essential to national independence. It cannot, therefore, be said that our capacity to be self-supporting in bituminous coal supply has even been tested.

With regard to anthracite, the situation is different. It has been contended that where Canadian bituminous coal was available the use of anthracite is a luxury, and this contention is still maintained. There is, however, a portion of Canada which, until our transportation systems are perfected, must have anthracite, although, as also already intimated, the anthracite supply must year by year decrease in quantity and increase in price by reason of exhaustion of the United States reserves. It is suggested that the whole matter of anthracite supply is in need of oversight by some department of the Government, similar to the recent organization of the Fuel Control Department. It is unnecessary to enter into details of why it is desirable that anthracite should be purchased, transported, and stored in Canada in the spring and summer seasons. The ideal condition would be to have the cellars of the ultimate consumer filled before the close of summer with sufficient coal to last until the next spring, but this is an ideal difficult of attainment. The possibility of storing anthracite in Canada during the summer to an extent sufficient to eliminate entirely any movement of anthracite from the United States to Canada, after, say, October first in each year, is a proper matter for government enquiry, and if found feasible, for government management. Such an arrangement would save much money, much anxiety in Canada, and would be welcomed by the United States railways and mine operators. The necessity to import anthracite being one of our national handicaps, it is for that reason alone a matter in which the government should take the initiative. Anthracite, unlike bituminous coal, suffers no deterioration in storage, and is not subject to spontaneous combustion or heating when piled.

In conclusion, one may be permitted to quote a remark of George Stephenson's, made long before coal had become so indispensable as it is to us to-day, "The strength of Great Britain lies now in her iron and coal beds; the Lord Chancellor now sits upon a bag of wool, but wool has long ceased to be emblematical of the staple commodity of England. He ought to sit upon a bag of *Coals*." In Canada, we have long thought in terms of wheat and lumber, but these are in process of ceasing to be our staple products. Iron and coal are the two commodities upon which our future chiefly rests, and we cannot long have our frontiers march with the opulent and enterprising nation of the United States unless we develop our coal resources in a more thorough-going fashion than has hitherto been attempted. Far from expanding our coal output, we are not even holding our own, and every year's record of Canadian coal outputs is more disappointing than the one preceding. How is it that the worst examples of dishonored bond issues in Canada are connected with coal-mining enterprises, and that, in at least two well-known instances, the capital invested by Canadian and British interests has been lost, and re-organization has been effected by United States capital? While a good many reasons could no doubt be advanced in explanation, the lack of any well-defined policy to foster coal production in Canada, because of its national importance, will explain the ill fate of many well-intentioned and promising coal mining flotations on this side the line.

It may be necessary to explain that this presentation of the Canadian side of the coal problem is not made in any spirit of hostility towards the United States. On the contrary, the generous and whole-hearted manner in which the U.S. Fuel Administration co-operated with the Fuel Controller of Canada

in the desperate conditions of fuel shortage in 1917-1918 is gratefully remembered here. In this instance the United States shared its inadequate supplies of fuel with Canada in a manner worthy of all praise.

The people of the United States, however, are the last people in the world to excuse a lack of enterprise in another people, and if they should criticize the backwardness of our fuel policy, it would be criticism well-deserved.

LIGNITE IN SASKATCHEWAN*

By A. MacLean

Annual General Meeting, Toronto, March, 1920.

The possibility of the occurrence of coal (or lignite) in the Northwest Territories conduced to explorations of Saskatchewan areas at a relatively early date by the Geological Survey. Thus, in 1857, Dr. Hector, of the Palliser exploring party, made an excursion from the north to the neighbourhood of Roche Percée. Dr. Dawson, when attached to the International Boundary Commission, examined the region and reported thereon in "The Geology and Natural Resources of the 49th Parallel," and again in the "Report of Progress of the Geological Survey of Canada," 1879-80. In 1880, Dr. Selwyn spent some time in the supervision of boring operations in the field between Estevan-Roche Percée and Turtle mountain, to determine the eastern limit of the field or to connect it with the lignite fields of Turtle mountain. In the summer of 1903, Mr. D. B. Dowling reported on the geology of the Estevan-Roche Percée-Taylorton part of the region. Mr. N. B. Davis, of the Mines Branch, includes this area in his report on the clay resources of southern Saskatchewan, and in 1916 Dr. Bruce Rose submitted a report to the Geological Survey of Canada on the Wood mountain-Willowbunch part of the Saskatchewan field.

The beds in which the lignite occurs in the Estevan district have been determined by Dowling to be of Fort Union age,[1] and inthe report on the Wood mountain-Willowbunch area, Rose assigns the coal-bearing beds of that district to the same formation. In this area Rose records the presence of the Lance formation underlying the Fort Union (50 feet ?), and under that the Fox Hill sandstone (75 feet ?), and the Pierre shales of the Cretaceous. Since a full description of this part of the area is contained in the report by Dr. Rose (Memoir 89, of the Geological Survey of Canada), a detailed account here is needless.

In the Estevan-Taylorton field the exposures are very few;

*By permission of the Directing Geologist, Geological Survey of Canada.

[1]Report on Coal Field of Souris River, No. 786, G.S.C., p. 15 F.

and the data obtained from give a total section of only about 175 feet. This contains only the upper quarter of the lignite-bearing formation, but includes the seams of greatest thickness and the only beds at present being worked. Below the level of the Souris river in this district there are, of course, no exposures, and the only information concerning the lower beds is to be obtained from boreholes, drilled either for prospecting purposes or by the farmers for water. The proprietors of such prospects, as well as the drillers and the farmers, have kindly supplied such information as they had or were free to offer, and this information has been of the greatest value in determining the location of the deeper seams and the wider structural features of the district.

Combining the data obtained from natural exposures of the various sections and the information obtained from drill records, we get a composite section for the Estevan-Taylorton part of the district as follows:—

Condensed Section Estevan-Taylorton District

No.	Material	Thickness	Elevation (Taylorton)
1.	Sands and clays	70 feet	
	Lignite	5 feet	
	Clays and sands	55 feet	
2.	Lignite	58 feet	
	Sand and silt	20 feet	
3.	Lignite	7-10 feet	1815 feet
	Stiff clay and sand	25 feet	
4.	Lignite, 9′—15′	12 feet	1789 feet
	Dark grey clays and sands	130 feet	
5.	Lignite	2 feet	1657 feet
	Sands and clays	207 feet	
6.	Lignite	4 feet	1446 feet
	Clays and sands	209 feet	
7.	Lignite	4 feet	1264 feet
8.	Clays and sands		
8.	Lignite	4 feet	1233 feet
	Blue clays and sands	186 feet	1047 feet
	Shale (Pierre)	500 feet (plus)	547 feet

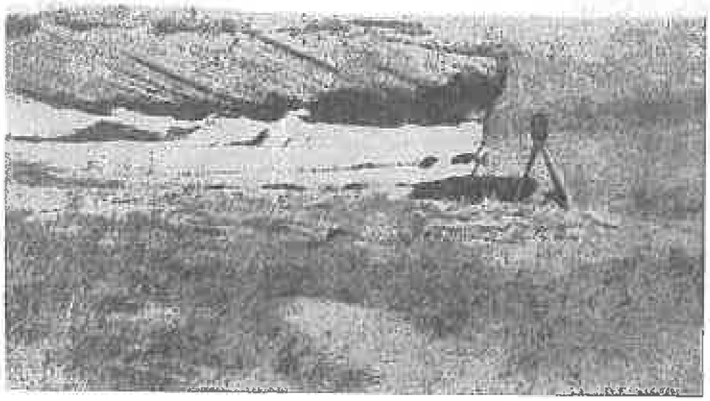


Fig. 1.—Cross-bedding in sandstone near Taylorton.

The section above the lowest lignite is considered to be of Fort Union age, and everything below the beginning of the shales is considered to be Pierre. Whether the 187 feet described by the drillers as blue clays and sands belong to the Fort Union formation, to the Lance formation, or to the Fox Hill sands, or whether it should be divided between them, the writer does not know. There is no exposure of this part of the section anywhere in the field unless it be in a cutting made by the Grand Trunk Pacific Railway, where it crosses the Souris river, near the International boundary. Here is a sandstone unlike any found elsewhere in the Estevan field, but which resembles descriptions of the Fox Hill sands.

THE LIGNITE SEAMS

1.—The upper seam (No. 1) occurs as shown on the map (Fig. 2), and occupies only the highest ground to the south and

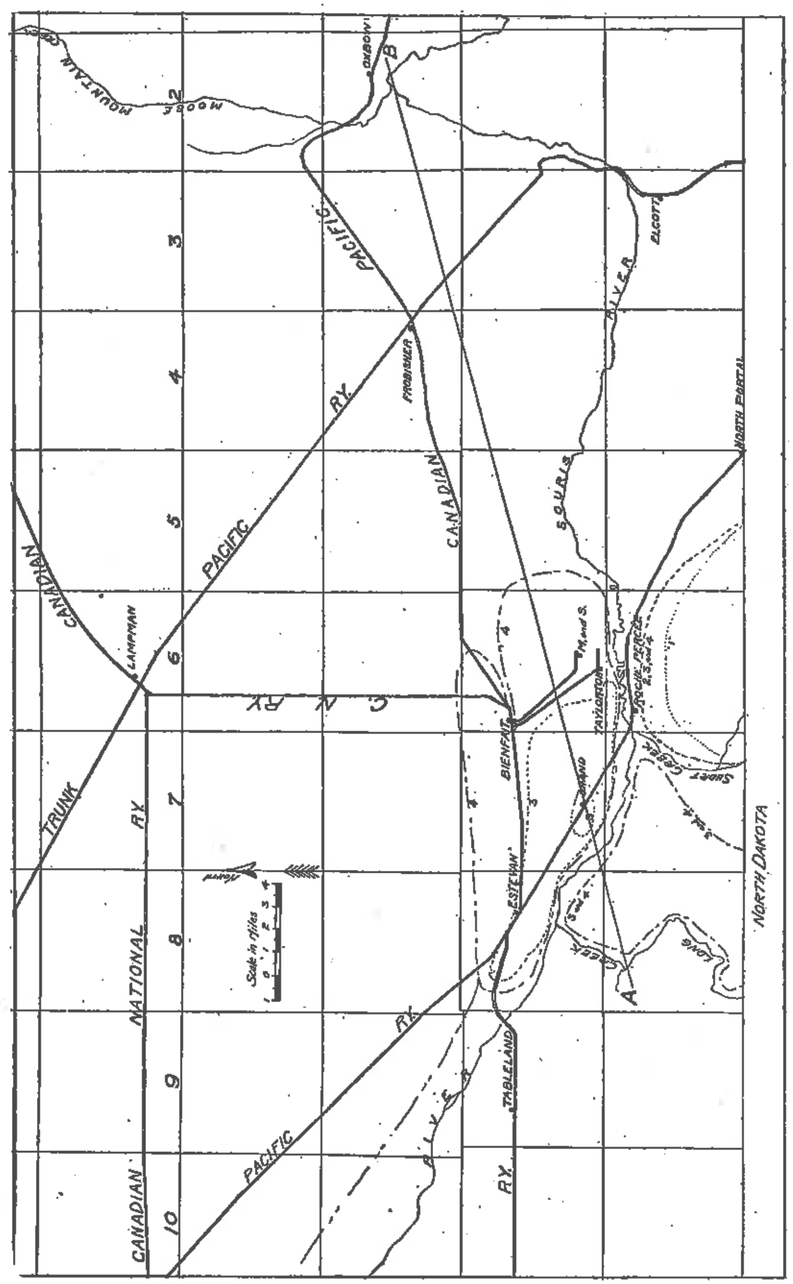


Fig. 2.

east of the Souris river and Short creek. It has a probable area of about 20 square miles, which, with a thickness of five feet, would give an estimated tonnage of 99,565,000. It has been mined by three operators in Canada and by the Makee Bros., whose mine is in North Dakota, a few miles south of Roche Percée and just across the boundary line.

2.—The next seam (No. 2) occupies about the same area as No. 1, but extends a little farther northward, outcropping on the slopes of the river valley. It has an area of about 30 square miles which, with a thickness of about five feet, would give a tonnage of 149,347,500. This is the seam that is being worked in most of the mines south of the river at Roche Percée, and the product is sold locally or teamed to the railway and shipped to different points in Saskatchewan.

3.—Seam No. 3 is more extensive. It occurs at a fairly low level at Roche Percée, being in most places below the Short Creek valley. It is worked there in two mines—the Siddal and the Interprovincial just across the river from Roche Percée. From here it extends northward to near Bienfait, thence westward through Estevan to Long creek, and thence south to the boundary. It pinches out toward the east and consequently is not found along the Bienfait-Taylorton line. Again near Shand it has disappeared or is represented only by thin streaks of lignite. It seems to be best developed at Estevan, where the seam is worked in all the mines with the exception of the Wooloomooloo. Here it is 7 feet thick in the main seam, with a further thickness of 3½ feet, separated from the main bed by clay partings of 12 and 24 inches. The area covered by this seam is estimated to be 90 square miles; and, with an average thickness of 6 feet, the coal content would be 537,651,000 tons.

4.—Seam No. 4 is considered to be the most important in the district, being the one from which all the coal is now mined in the Bienfait-Taylorton district, at Shand, at the Wooloomooloo, Estevan, and at Anderson's mine on Long creek. It is for the most part more consistent than the seam lying above it, but at Estevan it is badly parted by clay, being split into four

seams (totalling 7½ feet) by three beds of clay having a total thickness of 5 feet. Nevertheless, it is worked here on one of the seams 3½ feet thick (the smallest seam now being mined in the district). Its area to the east of Long creek at Estevan is probably 150 square miles. As this seam undoubtedly attains a thickness of 15 feet (to the east of Bienfait, it is said to have a thickness of 22 feet), it may safely be accorded an average thickness of 10 feet. On this basis its coal content would be 1,493,475,000 tons.

5, 6, 7 and 8.—The last mentioned is the lowest seam worked in this district, but below it are four others, with a wide distribution over the field. They are nowhere exposed, but are commonly encountered in drilling. At Taylorton they have elevations of 1657, 1446, 1264 and 1233 feet, being known as the 200, 400, and 600-ft. seams respectively. These seams have probably the same extent as the Taylorton-Bienfait seam except that they probably extend farther eastward and northward. The lowest two are probably the two seams occasionally encountered in well drilling at Oxbow; and probably also are the lowest seams met with at Turtle mountain, as in both these localities they seem to lie about the same distance above the Pierre shales. The fact that many wells penetrate to the shale at Oxbow without encountering these seams is due to the irregular pre-glacial surface, which had been carved by erosion in such a manner that, in some places, the seams have been removed and in others are left in old pre-glacial hills, which remained as buttes before they were covered with glacial drift. (See Fig 3.)

These seams have a total thickness of 14 feet, the top seam being only two feet thick and the others about four feet thick each. (At Oxbow the lower two are five and six feet thick respectively.) The area embraced by these seams is at least 200 square miles, hence their coal content (exclusive of that of the top seam) is 2,389,560,000 tons. This, including the coal content of the seams above, gives a total of 4,669,598,000 tons for the area to the east of Long creek, where it joins the Souris.

In outlining the areas for the above seams, it is considered that they are cut off to the eastward by the erosion depression

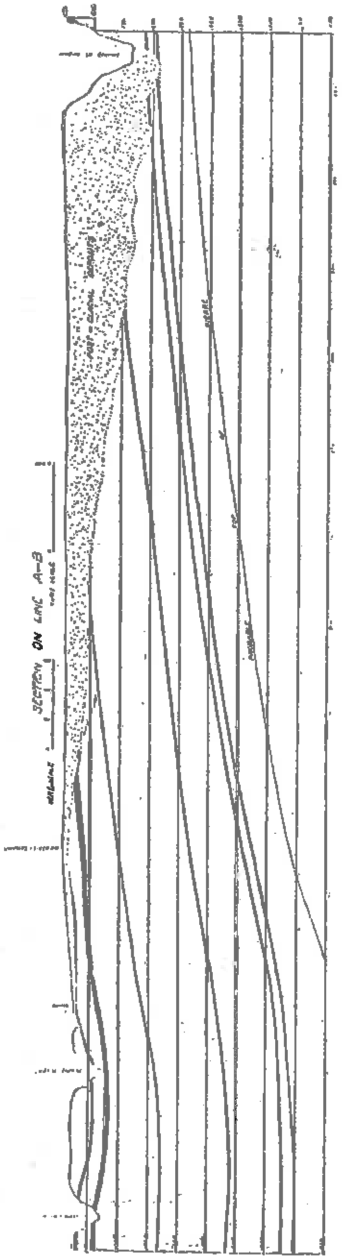


Figure 3.

of the pre-glacial surface, combined with the easterly component of the dip of the beds. To the northward, the dip brings the upper seams to the surface a short distance north of the Estevan-Bienfait line, or about the north side of Tp. 2, ranges 7 and 8. The lower seams should continue much farther north, but as about 4 miles north of Estevan a well was sunk to a depth of 400 feet without encountering coal, it would appear that the boundary of the lower seams is the same as that of the upper seams. At Lampman, about 15 miles north of Bienfait, lignite has been reported, and, several years ago, a shaft was sunk toward the seam. The work was discontinued, however, before coal was struck and no further work was done. Logs of several wells drilled for water in the neighbourhood show no record of coal. It is probable, therefore, that these wells are located in pre-glacial erosion valleys or that the lignite (as at Oxbow) occurs only in a few pre-glacial erosion remnants.

In reference to possible coal to the southwest of the Souris above Estevan, it may be stated that coal seams have been pro-

spected at Halbrite and at Goodwater, while at Tableland the existence of several seams has been proved by drill holes. Six miles to the south of Tableland there is a 14-ft. seam of lignite at a depth of 114 feet; at Torquay, it is found at 400 feet, at Maxim at 85 feet, and to the south of Neptune it outcrops at the surface at the foot of the Missouri coteau. The region from that neighbourhood to the Big Muddy country and from the latter area to the Wood mountain, Willowbunch district, is undoubtedly coal bearing, but to date, the writer has been unable to correlate these seams with those above described. This correlation is difficult on account of the structural features that probably intervene between the area to the west of a line through Halbrite and Tableland, and the area to the east of it. This may be discussed under the general head of structure.

In general, the beds of this area appear to lie horizontally, but after obtaining the elevations of the superficial beds, of exposures, and also of the deeper beds wherever definite horizons could be obtained, it was evident that there is, over both wide and narrow areas, a distinct variation from the horizontal. In the neighbourhood of Roche Percée and Taylorton, there is a pronounced depression, from which the beds rise towards the northwest, north, northeast, and east, tending apparently toward a summit in the neighbourhood of Moose mountain. This would make it appear that Moose mountain is a structural feature as well as, in part, an erosion remnant. In places the rise from this basin is as much as 45 feet to the mile (being greatest between Taylorton and Oxbow, in a northeasterly direction). This dip, of course, rapidly brings the beds to the surface in all directions except to the west, where the lignite-bearing formation is more or less continuous.

In addition to this larger structural development, the coal seams themselves show very decided rolls, summits, and basins, as suggested in the section (Fig. 3). For instance, at the Estevan town well the elevation of the 7-ft. seam is 1843 feet, and at the Parkinson mine two and a half miles to the southeast it is 1787 feet (a fall of 56 feet), while in the Nicholson mine, a quarter of a mile south of the Parkinson, the elevation is 1767 feet, or a drop of 20 feet in a quarter of a mile. These rolls and pitches are, of course, encountered in all the mines, and, in the case of those near the river valleys, may be due in part to differential subsidence consequent on the yielding and flowing of the underlying clays. This explanation will not, however, account for the more pronounced deformations of the beds, both near and distant from the river. One of the best marked instances of the disturbance of the district is to be seen in the stream valleys to the west of Halbrite, where the beds are folded, faulted, and tipped on edge (at all angles up to 90°), thereby exposing seams which are not shown elsewhere in the immediate district (Fig. 4). This folding is displayed again near Goodwater, but from that locality to Estevan there are no exposures and the attitude of the strata is unknown. To the west of Estevan, where the Neptune branch of the C.P.Ry. crosses the Souris, the same disturbance of the beds is shown both in the exposures in the cutting in the bank above and by the records of the test wells at the bottom of the valley.

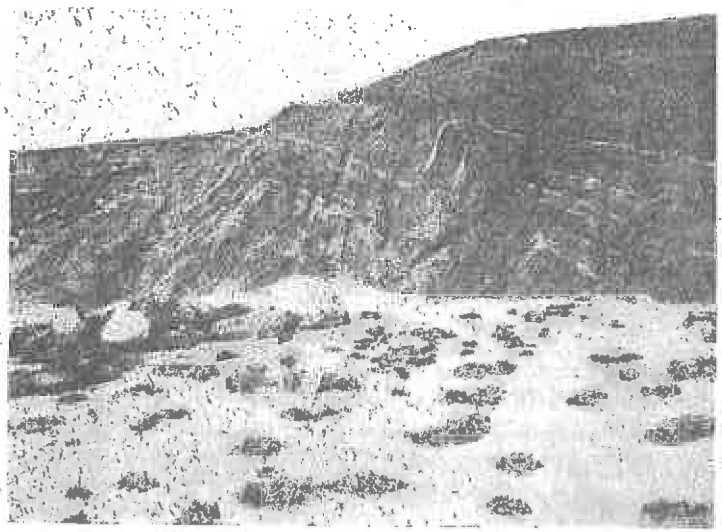


Fig. 4.—Folding near Halbrite.

Mr. D. B Dowling has suggested that the cause of this disturbance has been the sinking of the basin or the uplift of the rim about it causing a compression of the upper beds. It has not been possible, as yet, to find the lines or area along which this disturbance found expression, or to ascerain if they follow any line in particular. It is probable that there is a more or less continuous line from Halbrite toward Tableland and perhaps from there toward the boundary. Hence it has been deemed inadvisable to try to trace the Taylorton-Estevan seams across this line, and, for the present, estimates have been made only in respect of the seams definitely known to the east of Long creek where it enters the Souris at Estevan.

DESCRIPTIONS OF MINES

[1]*Western Dominion Collieries, Taylorton, Sask.*—This Company's mine is one of the oldest, and one of the largest, in the Souris coalfield. Its history dates back over 20 years. The mineral rights in the Taylorton area, where the mine is operated, consist of 2,280 acres. The shaft and works are situated in section 3, township 1, range 6, west of 2nd meridian. At present the seam is worked by one shaft, and the coal is found at a depth of 85 feet.

The hoisting shaft is 9 ft. by 18 ft. in the clear, and is timbered with 3-in. barring and heavy sets placed every five feet. The heavy sets are 8 in. by 10 in., and are suspended from one another by hanging bolts. The shaft is divided into three compartments, two for hoisting and one for a manway, which is completely boarded, and covered on top to protect the men from falling pieces of coal during hoisting operations.

The seam being worked is a low-grade lignite, 7½ feet thick. It lies horizontally and as there are no particular cleavage lines in the coal it can be mined as easily in one direction as in another. No bands of dirt are found in the seam, and it is remarkably free from 'bone.' It is overlain by a clay roof, and is superimposed on a firm clay floor. In an open fireplace or

[1]Description furnished by A. A. Millar, Manager.

stove the coal burns freely to a brownish-white ash, leaving no clinkers. It gives good results in stationary boilers, and is an excellent producer-gas coal.

As indicated by the Government reports, the market for this coal is gradually increasing, and the output for the Western Dominion Collieries, Ltd., for the year ending December, 1919, exceeded 115,000 tons. On an average, 90 men were employed above and below ground during the year. The analysis of the coal from the seam now being worked is:—

Moisture	31.66%
Vol. matter	25.25%
Fixed carbon	37.05%
Ash	6.04%
	100.00%

At Taylorton the 'room and pillar' method is followed. Two main entries are driven due north from the shaft, and at every 500 feet room-entries are turned off in pairs to the east and west. The entries are made 7 ft. wide for the full height of the seam, and the pillars between the entries are 35 ft., the same thickness being retained in the room entries. The rooms are 18 ft. wide, and are turned north and south off each entry. They are carried a distance of 250 ft., where they are cut off by the rooms from the other entry. All the coal is cut by electric coal-cutting machines, the depth of the undercut being 6 ft. The Morgan Gardner breast machine, equipped with a self-propelled truck, is in use. Four are in operation and give excellent results. No inflammable gas is found, and open lights are used throughout the mine. All blasting is done with black powder by the miner himself.

The main haulage entries are laid with 30-lb. steel rails, and the rooms with 16-lb. rails, the gauge being 30 in. Mine cars (150 in number) of 1.75 tons capacity, equipped with Whitney roller-bearing wheels are in use. For underground haulage, two 5-ton Morgan Gardner locomotives (of the trolley type) are employed for transporting the coal from sidings on

the room entries to the shaft bottom. The haulage from the room face to these sidings is performed by horses, and the sidings are advanced as the workings extend, thus keeping a regular haulage distance for the horses.

The tipple is of wood, 16 ft. inside by 75 ft. long, and the height of the headframe is 65 ft. to the pulley centres. Steel self-dumping cages, of the Olsen type, provided with standard safety devices, are used for hoisting. Two automatic car releasers are placed in the shaft bottom, thus permitting the cars to be rapidly caged. The shaft bottom is timbered with

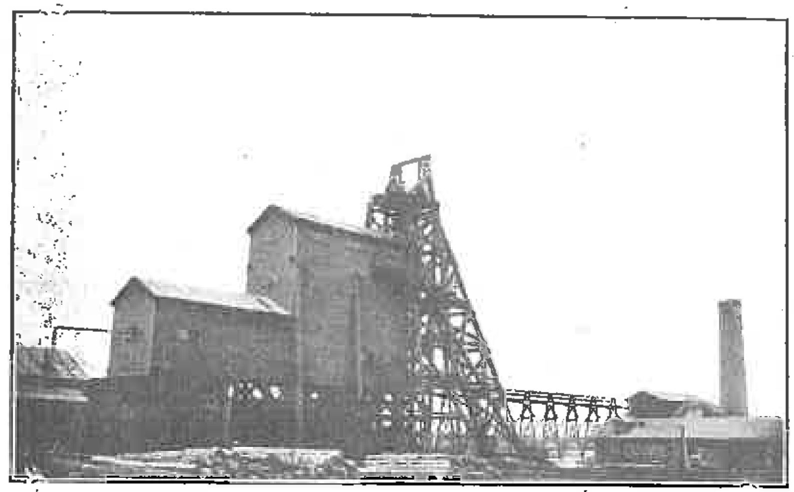


Fig. 5.—Tipple of Western Dominion Collieries at Taylorton.

heavy timber, and is double-tracked for 300 feet, providing ample storage for cars in the case of delays in hoisting. The hoisting ropes are of plough steel, 1⅛ in. in diameter.

The coal is delivered from the car into a dumping-chute, thence over a 1¼-in. bar screen, 12 ft. long, to the weigh basket. After the coal is weighed, the basket is allowed to drop (by releasing a brake) and the coal placed on the screens. The basket is then brought back into place by means of counter-weights. The screens are of the ordinary shaker type, 50 feet long, and are operated by a 15 h.p. Nagle horizontal steam-engine. The screening plates are perforated, and the coal is separated into four different grades in addition to 'run of mine.' The sizes usually made are:

Slack	0 to 1 in.
Pea	1 to 1½ in.
Cobble	1½ to 2½ in.
Lump	2½ in. and up

The percentages of the different sizes are:

Slack	10%
Pea	9%
Cobble	8%
Lump	73%

A mechanical loader is used for loading the lump size into box cars. This loader is of the 24-in. drag-extension type and handles the coal with the minimum amount of breakage. It is driven by a 20 h.p. enclosed motor. A 16-in. single-acting belt-loader is used for loading the cobble coal into box cars, and is driven by a 6-h.p. shunt-wound motor.

Two 100-ton track scales are installed at the tipple; all empty cars are weighed before loading and the amount of coal put in each is ascertained before the cars leave the loading chutes.

The power-house is of frame construction 22 ft. by 60 ft., in which are installed the hoisting engine and generator. The hoist is a Lidgerwood second-motion engine of 75 h.p., with a 5-ft. drum. The generator is a 100-kilowatt, 250-volt, direct current type, belt-driven by a Skinner engine. The steam-plant comprises three 100 h.p. boilers (of Jenckes type) and one 60 h.p. boiler. Slack coal is used for firing, being conveyed direct from the tipple to a bunker at the boiler-house by a 6-in. button conveyor, driven by a 10 h.p. horizontal steam engine.

Housing is provided for two locomotives which haul the coal over the company's spur track to the main lines at Bienfait.

The locomotive shed is of frame construction and is provided with ash pits, etc. The size of the building is 33 ft. by 66 ft.

A new ventilating fan is about to be installed, and with that in place the whole will form a very complete and up-to-date mining plant. The capacity is 125 tons per hour.

The mine is connected by a spur track to both the Canadian National and the Canadian Pacific Railways at Bienfait.

Manitoba & Saskatchewan Coal Co. Ltd.[1]—This company's plant is situated five miles southeast of the village of Bienfait, Sask., and has direct railway communication with the C.P.Ry. and the C.N.Rys., the coal being transferred to the railway yards by the company's own locomotive. The plant is equipped to handle 1,000 tons per day.

A two-compartment vertical shaft has been sunk from the surface to the coal seam 80 feet below. It is equipped with two self-dumping cages, the coal being raised on a steel head-frame to the dumping plates 70 feet above the rail level. The mine cars are of a net capacity of 5,000 lb., the tare being 1,700 lb.

The coal, after being dumped, passes over a bar screen into the weigh basket, the miners being paid on a screened coal basis. After leaving the weigh basket the coal passes to shaking screens 60 ft. in length, where it is graded to the following sizes: screenings, pea, cobble, screened coal and run of mine. The screenings are conveyed by a scraper conveyor to the power-house and utilized for raising steam. Each size, as it passes over the screens, drops into bunkers and is loaded into box cars on separate railway tracks. The mine run and screened coal pass over a picking table where any impurities are removed before being loaded into the box cars by an Ottumwa box-car cradle-loader.

The hoisting engine is a 19-in. by 36-in. coupled horizontal engine, geared 6 to 1. The diameter of the drum is 6 ft., and of the steel cables, 1¼ in. There are two return tubular boilers,

[1]Description furnished by Mr. Addie, Manager.

125 h.p. each, with a working steam pressure of 100 lb. Water is obtained from the Souris river, two miles south of the mine, where a pumping station is installed. A fan, 18 ft. in diameter, supplies fresh air for the mine workings.

The seam being mined is a high-grade lignite, at a depth from the surface of 80 ft. The average thickness is 12 ft; it lies practically level, and is worked by the 'pillar and stall' system. The stalls are 15 feet wide and the entries 9 ft. wide; pillars 18 yards square being left after the first working. These pillars are extracted in the second working.

The coal seam is non-gaseous and gives off little water. Until the present, horse haulage and hand mining have been in force, but it is the intention of the company this summer to provide electric haulage and install electric coal cutting and loading machines.

The company owns about 70 cottages and a large boarding house. The cottages are rented for a nominal sum and coal is supplied to each cottage at a charge of about $20 per annum.

Attached to the cottages are large gardens which are ploughed and fenced by the company.

The company also provides a large hall suitable for dancing and entertainments, and also a pool and billiard room for the use of its employees.

The Shand Coal & Brick Company.[1]—The coal seam averages 8 ft. in thickness and has an overburden of about 75 ft. It has a slight dip to the east.

The mine is worked on the 'room and pillar' system. Entries are driven 7 ft. wide with 30-ft. pillars. Rooms are set out at 30-ft. centres and worked 16 ft. wide. Eighteen inches of coal is left to support the roof but most of this is recovered in drawing the pillars. Hand mining methods are employed and haulage is done by horses. Air for the ventilation of the mine is supplied by a reversible fan of 10-ft. diameter. The coal is hoisted to the surface through a one-compartment shaft.

[1]Description furnished by Mr. J. Peterson, Manager.

The surface plant consists of tipple, arranged with gravity screens to grade the coal into lump, nut, run of mine, and slack. The lump and run of mine coal is loaded into the cars with an Ottumwa box-car loader.

The power plant is equipped with two 125-h.p. boilers, which supply the various needs of the mine, and in addition furnish power and heat for a brick-plant that is operated in connection with the mine and gives employment to forty-five men during the summer.

With the present equipment about 65 men can be employed as the mine.

PRODUCTION IN SASKATCHEWAN IN 1918

Name of Operator		Total Production
*Anderson, Niels	Estevan	1,041 tons
*Bienfait Commercial	Bienfait	19,261 "
*Bienfait Mine (The)	Bienfait	65,922 "
Eidsness Bros	Gladmar	1,285 "
*Estevan Coal and Brick	Estevan	10,954 "
Heuval, Henry V	Hart	1,521 "
*Interprovincial Coal Co	Roche Percée	4,487 "
Lilja & Dempsey	Shaunavon	2,261 "
*McNeil and Rooks	Estevan	1,249 "
*Manitoba & Sask. Coal Co	Bienfait	75,369 "
*Nicholson, H	Estevan	2,282 "
*Parkinson, Geo	Estevan	3,660 "
*Saskatchewan Colliery Co	Roche Percée	2,600 "
*Sask. Coal, Brick & Power	Shaud	32,609 "
*Western Dominion Coll	Taylorton	104,834 "
*Wilson, Alex	Taylorton	1,389 "
All other operators (51)		16,123 "
Produced by total of 67 operators		346,847 tons

*Operating in the Estevan-Bienfait-Taylorton area.

PRINCIPLES AND PRACTICE OF FUEL BRIQUETTING

By Edgar Stansfield

Annual General Meeting, Toronto, March, 1920.

In the two months following November 11, 1918, the writer visited, on behalf of the Lignite Utilization Board of Canada, almost every briquetting plant then in operation in the United States and Canada, and some that were not in operation, thirteen in all. He has since been in charge of briquetting experiments carried out by, and for, the Lignite Board in Ottawa at the Fuel Testing Station of the Mines Branch. The results of the Ottawa experiments are not yet ready for publication, but the following notes on the briquetting industry, whilst based on the above visits, are written in the light of the experience since gained. The above credentials do not class the writer as an expert on briquetting; nevertheless, it is hoped that this paper will interest many, as its subject is now attracting wide attention.

No definition of briquetting or explanation of the reasons for briquetting is necessary. It is also well known that the industry is further advanced in Europe than in America. The path of progress here has been marked by many failures, not a few of these due, the writer is satisfied, to attempts to transplant an industry from one country to another without regard to vital economic differences. An American industry, however, is now well established along American lines; we cannot hope that there will be no more failures, but we can reasonably hope that progress will be sure and rapid.

As just mentioned, an American industry has been established; nevertheless, the outstanding impression gained from the above trip was that of an infant industry wherein scarcely one stage of the process had been standardized. Moreover, all subsequent experience has taught that, even in the laboratory, it is extremely difficult to determine the best procedure for any stage in the process because of the many variables involved. There is no royal road to briquetting; the perfect commercial briquette as an aspiration, not a realization.

Coal is almost invariably briquetted with the addition of a binder. In some plants this is added as a pulverized solid, in others the binder is liquid or is melted before it is added. Figure 1 shows a typical flow-sheet through either type of plant.

FLOW SHEET, TYPICAL BRIQUETTING PLANTS, WITH HARD OR SOFT BINDER.

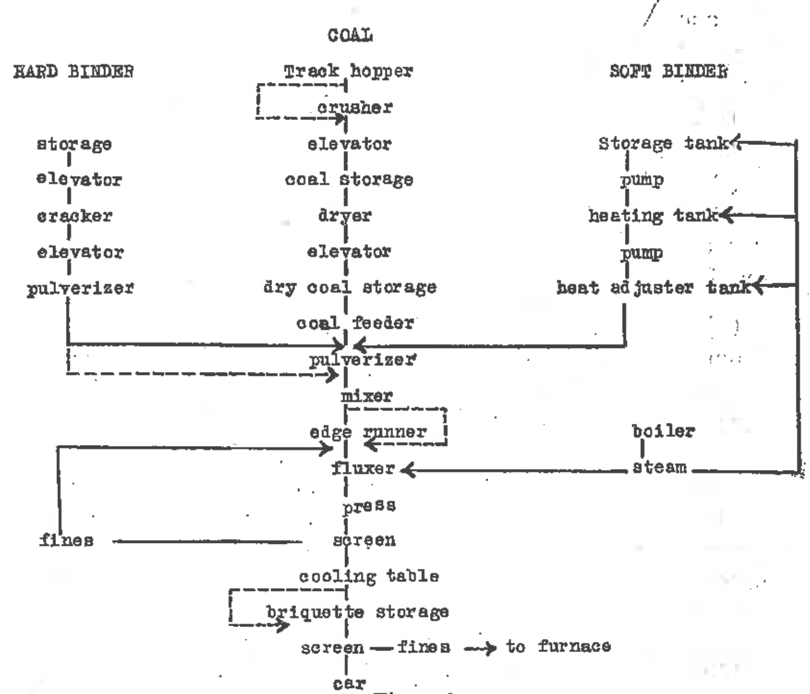


Figure 1

Many of the steps shown call for no comment. The coal or other raw material is crushed, if necessary, and then dried. Crushed or pulverized hard binder is added to the dried coal, which is then pulverized, or it is pulverized and then has a fluid binder added. The coal and binder are raised to a temperature sufficient to make the binder flow readily, and thoroughly mixed and kneaded together in mixers and in fluxers or kneaders of various types. These mixers are sometimes direct-heated, but are usually steam-heated externally; in most cases high-pressure steam is also passed into the charge in the mixers. The mixed mass is finally brought to a suitable temperature and sent to the press to be formed into briquettes. From the press the hot and somewhat fragile briquettes are sent over an inclined screen to remove the fins, broken bits, and other fines, which are sent back to the mixing train and re-pressed. The briquettes are then cooled and stored. They are usually re-screened before shipping, and the fines thus removed are burned in furnaces around the plant.

It will be well at this point to consider the properties that the finished briquettes should possess:—

(a) They should be of suitable size and shape for convenient handling, without sharp corners or edges that are easily knocked off, and of such form as to allow free passage of air through the furnace during combustion.

(b) They should be strong, clean to handle, not too brittle, and neither soft, friable nor easily abraded.

(c) They should be weather proof, not affected by rain or frost, or soften unduly when exposed to the summer sun.

(d) Finally, they should form a good fuel—long flamed, short flamed, or smokeless, according to the purpose for which they are required, holding their shape without softening or crumbling in the fire, not clinkering easily, and of high calorific value and low ash content.

These form a set of conditions that are extremely difficult, if not impossible, to satisfy in their entirety without making the cost prohibitive.

Many of the factors and variables involved in briquetting are enumerated below; some of the more important ones are more fully discussed.

Raw Material

The raw material is usually anthracite, bituminous coal, or lignite. These different materials may be briquetted raw or after carbonization; if carbonized, much depends on the method

and temperature of carbonization. Anthracite fines cannot easily be utilized without briquetting; plants are now briquetting the screenings as they are produced, and are also treating material recovered from the old culm piles, or dredged from the rivers of the Pennsylvanian anthracite district. Bituminous fines can be used in other ways, but are often briquetted. Lignite is briquetted raw in Germany, without the addition of a binder, but this process has failed with American lignites. Carbonization of lignite notably increases its calorific value, but produces a fine product that requires to be briquetted for satisfactory use in existing domestic furnaces. The porous and friable nature of carbonized lignite necessitates the use of nearly twice as much binder as is required for the same weight of anthracite. Coke breeze is difficult to briquette, and causes excessive wear in the machinery; it can be briquetted best with the addition of some bituminous coal. Semi-coke, or coke made by low-temperature carbonization of bituminous coal, is also briquetted.

Preparation of Raw Material

The coal is always dried if it is moist. This seems peculiar, as water or steam is almost always added later, but the binder does not adhere well to wet material. In some plants steam dryers are employed; in others, drying is accomplished by direct heat under the first mixers, the binder not being added until later. In most modern plants the coal is dried in a furnace-fired rotary-dryer.

The degree of pulverization of the material is most important. The optimum result is probably attained with material that packs with the minimum of voids, but no exact information appears to be available. A plunger press will more satisfactorily briquette coarser material than will a roll press.

Fine material makes a smooth, good-looking briquette, but requires more binder than a somewhat coarser material. The binder must completely coat every particle. Fine crushing increases the area to be coated, while coarse crushing will leave voids that can only be filled by the binder.

BINDER

As all known binders are comparatively costly, the success or failure of a briquetting plant will often depend upon the availability of a satisfactory binder at a reasonable cost, and upon the ability shown to reduce to the minimum the amount of binder employed. The quantity of binder required varies with its nature; varies inversely with the strength and density of the material bound, strong particles taking little, weak and porous particles more; varies with the pressure employed, a plunger press operating with less binder than a roll press; varies with the thoroughness of the mixing; and varies with the uniformity of feeding and the skill of the operator.

Organic binders increase the volatile matter, and, therefore, the smokiness in burning, although in some plants this defect is remedied by a subsequent heat treatment of the briquette. Inorganic binders have not, so far, been very successful, and they increase the ash content of the briquette.

Single binders may be employed, such as coal-tar pitch, natural asphalt, petroleum pitch or asphalt, lignite-tar pitch and wood-tar pitch; each of these of either high, medium, or low melting point; also sulphite pitch, either dry or in solution. Treated binders may be used, such as chlorinated or sulphonated tars, blown or oxidized asphalts, etc. Combination binders may also be used, almost any combination of the above being possible, or they may be used singly or in mixtures with the addition of any variety of tar, or soft asphalt, flour, starch, clay, water glass, magnesium chloride, cement, straw, etc., with or without the addition of water and with or without emulsification —a long list of possible binders, none perfect.

The two binders chiefly used are petroleum pitch and coal-tar pitch. Sulphite pitch alone or mixed with petroleum pitch is used in two plants. Coal-tar pitch is sometimes objected to on account of its smell; the briquettes, however, stand up well in the furnace. Petroleum pitch gives a slightly less objectionable smoke, but the briquettes become very weak when heated in the furnace. Sulphite pitch is distinctly less smoky than either of the above, but used alone, it does not make a waterproof briquette; this defect can be remedied by a low temperature baking or by replacing some of the sulphite pitch with asphalt or coal-tar pitch.

MIXING AND KNEADING

Three types of mixers are commonly employed, viz: horizontal helical mixers, vertical or horizontal fluxers, in which the mass of material is thoroughly agitated by means of revolving arms between stationary ones, and edge runners or masticators. In many plants the first two types are employed, in others, all three. The edge runner not only mixes but also pulverizes the coal. The binder may be added in powdered, molten, or atomized form. Only hard pitches can be added in the powdered form; this method reduces the amount of mixing required, but it gives trouble if the coal is warm when the binder is added. Storage of a reserve of solid pitch is apt to be troublesome, especially in warm weather. Pitches can be readily stored in tanks, and this is permissible if they are to be used molten; then a soft pitch is commonly employed, being steam-heated to the desired temperature and run through a regulating valve into the mixer. Atomized binder would have some advantages, but this method is not commonly used.

The mixture of coal and binder must be raised to a temperature high enough to cause the binder to flow freely. The mixers are often externally heated, but in addition, steam is commonly blown in. Blowing steam into the charge raises the temperature rapidly. It is also supposed to make the binder flow more freely, and to displace the enclosed air, thus making a dense briquette. A moist mixture is also less likely to adhere to the press. The Ottawa experiments have shown that an equally good, if not superior, briquette can be made without the use of steam; but the writer knows of only one plant where no water or steam is employed.

Mixing must be thorough if the amount of binder required is to be kept low; too prolonged a treatment, however, results in oxidation and an inferior product.

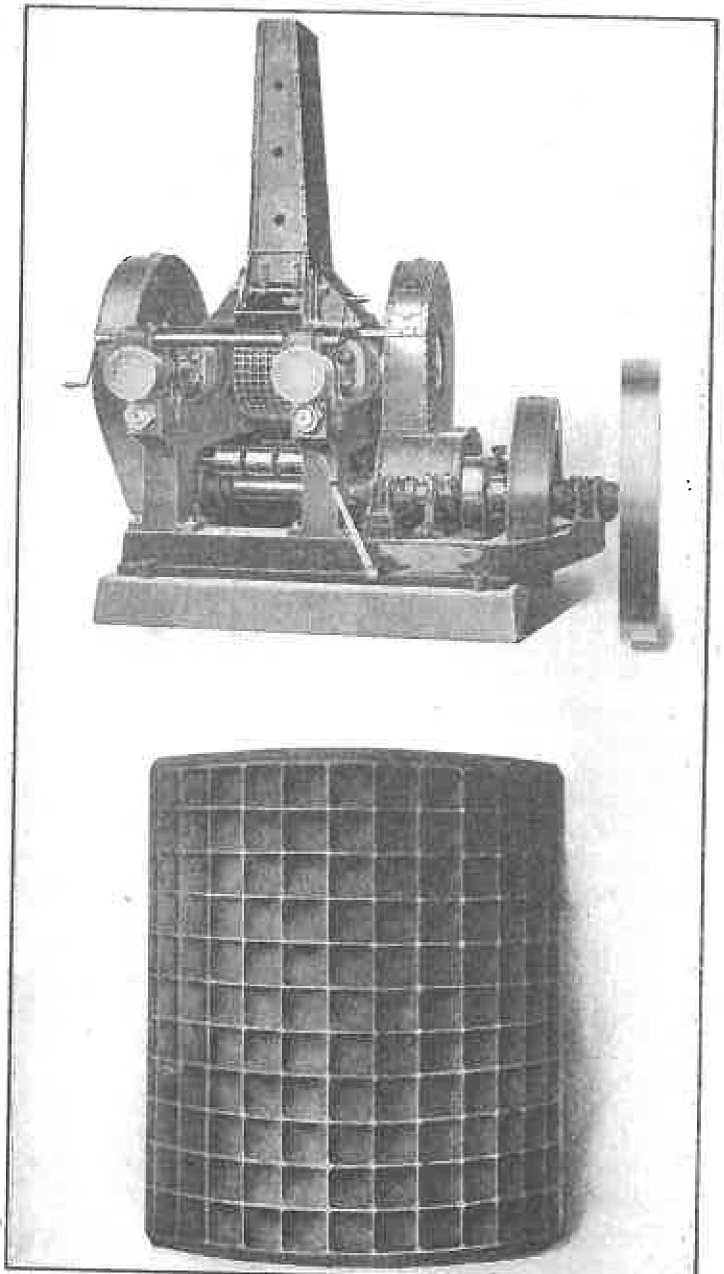


Figure 2

PRESS

American practice calls for small briquettes, that is, for briquettes of not more than one pound in weight. The simplest press for this purpose is the Belgian or roll press, made in a number of different patterns, one of which is shown in Figure 2. The material is fed to a pair of rolls, which rotate in opposite directions—one of the rolls is illustrated separately in the figure. The depressions in the rolls coming together, form pockets into which the material is compressed. As the pockets re-open, the briquettes drop out. The roll presses in this country make either egg or pillow-shaped briquettes of from one to seven ounces in weight. The material that is pressed on the flat surfaces of the roll between the briquette depressions forms what is known as a 'fin,' which joins the briquettes together. The fin is knocked off and sent back to the mixers for retreatment, leaving a rough surface on the briquette. Pillow-shaped briquettes can be made with fewer fins than can egg-shaped briquettes, but the latter are in other ways the better. As the rolls wear, the thickness of the fins increases. The rolls can be re-turned and adjusted a few times to remedy this, but have then to be replaced.

The Komarek press shown at the bottom of Figure 3 is an American modification of the roll press. It has a high output, making a barrel shaped briquette of about two ounces in weight.

The Rutledge press shown at the top of Figure 3 is also of American design. It is a plunger press of high capacity, making a cylindrical briquette of from 10 to 16 ounces in weight. The plungers and dies are made in blocks, commonly 14 to a block. After the dies in any block have been filled with material, plungers come up from below and down from above and the material is strongly compressed. The upper plungers are then withdrawn and later the other plungers driven further, eject the completed briquettes. The die blocks are so connected as to form an endless travelling belt; the upper and the lower plunger plates form similar but shorter belts.

Presses vary in capacity from the smallest roll presses, turning out two or three tons per hour, up to Rutledge presses

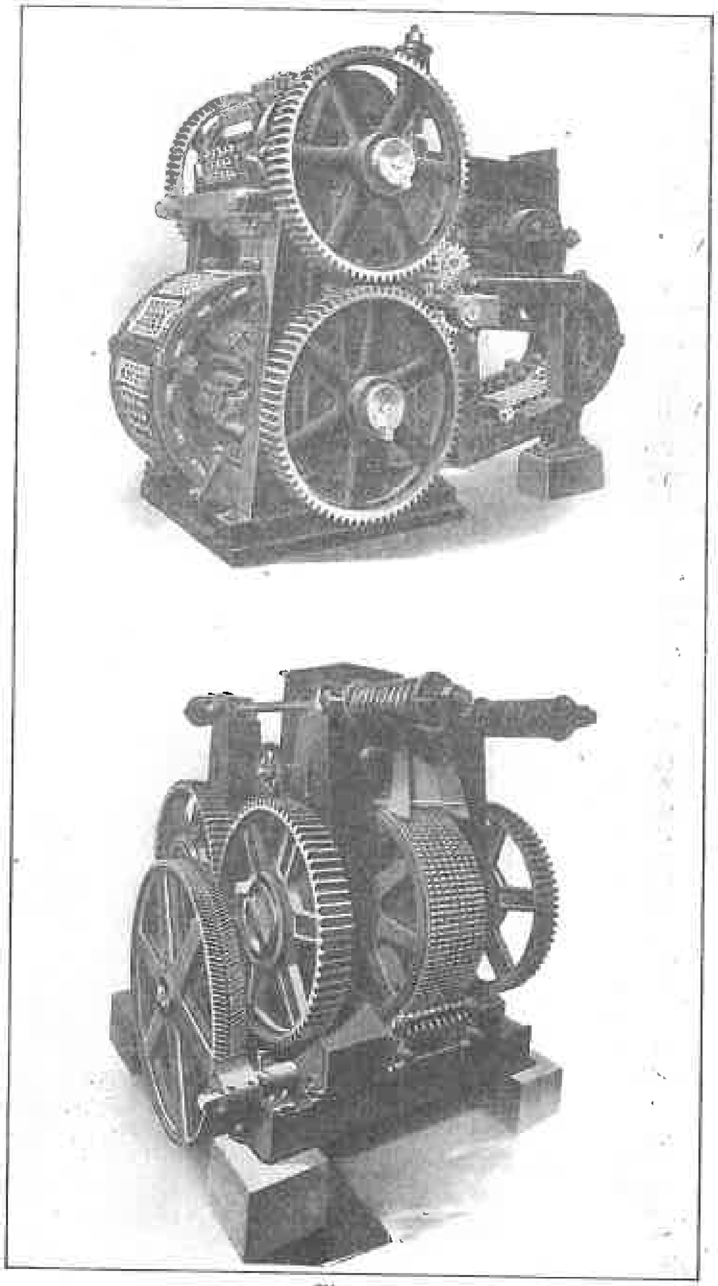


Figure 3

of forty, or even more, tons per hour. Roll presses vary not only in width of roll, but also in diameter. On this point, as on others, experts differ, some advocating small diameter rolls and others rolls of larger diameter. A denser briquette can be obtained by running the press slowly, but the output is thus cut down. It is important to bring the material to the right temperature before it enters the press. A high temperature tends to give a dense briquette, but if the temperature is too high, the briquette leaving the press is too fragile for economical handling.

Cooling Belts

Briquettes are cooled on long, slow-running, air-cooling belts or tables; or on shorter belts with water-spray cooling; or by total immersion in water.

Heat Treatment of Briquettes

Heat treatment of briquettes is the exception, not the rule. Briquettes made with sulphite pitch only require to be baked to render them water-proof. This also makes the briquettes smokeless, so that, although the briquettes are slightly weaker after treatment, the product is an almost ideal fuel. These briquettes are at present made only by the Fuel Briquette Co., Trenton, N.J. Another process, which has been developed at the experimental plant of the International Coal Products Corporations, at Irvington, N.J., is being tried on a large scale at their plant at Clinchfield, Va. This plant commenced operations in June, 1920, and a few weeks later was said to be running very satisfactorily. If the process proves the success that its promoters hope, it may play a notable part in future developments in Canada and elsewhere. So far only bituminous coal has been used. It is coked in special retorts at a low temperature and 'semi-coke' is thus produced. This coke is mixed with coal-tar pitch (produced in the tar refinery) and briquetted, and the briquettes coked at a high temperature in inclined retorts. The product is a dense, smokeless briquette of coke. A notable feature of the process is the high tar yield.

TYPES OF BRIQUETTING PLANTS

A study of the briquetting plants now operating gives an indication of the lines on which the industry will develop. These plants may be classified into four principal divisions:

(a) *Plants situated at, or near, a coal mine and operated by the coal mine operator.*—Plants of this class treating anthracite or semi-anthracite are those of the Bankhead Colleries, Ltd., at Bankhead, Alta., whose plant has been running since 1907; the Lehigh Coal and Navigation Co., at Lansford, Pa.; and the Delparen Briquette Co., Parrot, Va. A plant of this class treating bituminous coal is that of the Virginia Navigation Coal Co., Norfolk, Va. This plant is situated between the mine and its shipping port. Run-of-mine coal comes to the plant and is screened, the lumps being sold and the fines briquetted for the South American market. A plant of this class treating bituminous coal and lignite is that of the Pacific Coast Coal Co., Renton, Wash. This plant is situated similarly to the last one mentioned.

(b) *Plants situated at or near a coal mine, but operated by a separate operator.*—Plants of this class treating anthracite are those of the Scranton Anthracite Briquette Co., Dickson City, Pa. (the fines are water borne to this plant), and of the Gamble Fuel Briquette Co., Harrisburg, Pa., operated on fines dredged from the river. The latter plant is now closed.

(c) *Plants situated at the distribution point for the finished product.*—Plants of this class treating anthracite or semi-anthracite, are those of the Fuel Briquette Co., Trenton, N.J.; American Briquette Co., Philadelphia, Pa. (experimental plant now shut down); Standard Briquette Fuel Co., Kansas City, Mo.; Anthracite Briquette Co., Toronto, Ont.; and Nukol Fuel Co., Toronto, Ont. A plant of this class treating semi-coke is that of the International Coal Products Corporation, Irvington, N.J. (experimental plant). Plants of this type generally purchase screenings from the mines or from river dredgers, but they sometimes also purchase local screenings.

(d) *Plants situated at trans-shipping points of fuel where coal is screened.*—The plant of Stott Briquette Co., Superior, Wis., is of this type treating anthracite and bituminous coal; the plant of the Berwind Fuel Co., Superior, Wis., is of the same type treating bituminous coal.

DISCUSSION

MR. E. P. ROWE: It would be interesting to know the comparative fuel value that is contained in the lignite briquette in the West, as compared with the briquettes made from hard coal screenings.

MR. E. STANSFIELD: The calorific value of the briquette is dependent on the quality of the coal forming the briquette. Analyses show that some lignite briquettes have a heating value equivalent to 11,500 B.T.U. The anthracite briquette ranges from 11,500 to 11,600 B.T.U.

PEAT FOR FUEL

By A. A. Cole

Annual General Meeting, Toronto, March, 1920.

Canada has large coal resources both in the East and West, but for its coal supply central Canada (the province of Ontario in particular) is dependent mainly on the American coal-fields. Owing to labour troubles, transportation difficulties, and general war activities, the fuel situation in central Canada for the past two or three years has become increasingly difficult, and at times very acute. This led the Ontario Government in the session of 1918 to make an appropriation to ascertain to what extent peat and wood could be utilized for fuel, and so offset, in part, the shortage of coal. It had been almost decided to build an Anrep plant and test it on an Ontario bog, when it was found that the Federal Government was planning to build a peat plant from the design of Mr. E. V. Moore, of Montreal.

The Federal and Provincial Ministers of Mines, therefore, decided to build both machines for the experiments, and to divide the cost of the investigation equally between the two governments. The problem was to show whether peat fuel could be produced and marketed commercially in competition with coal.

The investigation was placed in the hands of a committee of four: Mr. R. A. Ross, of Montreal, and Mr. B. F. Haanel, of the Mines Branch, Ottawa, representing the Federal Government, and Messrs. R. C. Harris, of Toronto, and A. A. Cole, of Cobalt, representing the Ontario Legislature. Mr. E. V. Moore, of Montreal, was appointed engineer to the Committee.

Preparation of Peat for Market

The method adopted by the Peat Committee of preparing the peat fuel is known as the wet process, the product being termed 'air-dried machine peat.' This process is the only known economical one for the manufacture of peat fuel, and

is employed in Europe for the manufacture of millions of tons of peat annually.

The process consists essentially of excavation, maceration, and spreading, allowing the sun and air to do the drying and curing. The peat when taken from the bog contains about 88% moisture, and none of this water is removed mechanically; in fact, if the moisture content is below this amount it may be advantageous to add water to make the material work up to the best advantage.

The amount of maceration given to the peat determines to a marked degree the quality of the fuel produced; maceration increases the density of the finished product and hence also increases its value.

The peat is spread on the surface of the ground to dry, usually on a part of the bog which has been cleared and drained. To complete the drying and curing takes from two to four weeks. Towards the end of this time the peat is turned once, and a few days later is coned or stacked in hollow piles.

When the moisture has been reduced by about 95%, leaving a fuel with a moisture content of 25%, the peat is ready for marketing.

General Description of Peat Machines.—The Peat Committee has constructed two peat machines of different types, and with these will ascertain the commercial feasibility of manufacturing peat fuel in this country. Expensive manual labour makes it more imperative that mechanical appliances should be employed as extensively as possible.

Plant No. 1 is known as the Anrep Plant. In its design, care was exercised to adhere as closely as possible to the original drawings, which were made by the inventor, the late Aleph Anrep, Sr., for the equipment used in distributing the peat. The excavator, however, was re-designed in the light of experience gained with it in 1914. The machine was placed on caterpillers, and a new design of bucket-dipping element installed.

Excavation is made along the side of a ditch, the slope of which is about 45°, and the depth of the cut about nine feet.

The excavated material is passed through an Anrep mascerator, after which it is ready for spreading. From the mascerator it is discharged into open cars which are run out on to the spreading field by means of an endless cable haulage system. When the car reaches the spreader, which is operated as a separate unit, it is stopped and discharged, then is re-engaged to the cable system and continues around the field to its starting point.

The spreader smooths out the wet peat into a layer about five inches thick, and cuts it into long narrow rows by means of revolving discs attached to the rear of the spreader. The spreader lays down a row at right angles to the ditch that is being excavated.

The endless cable haulage system is laid out on the field in the form of a square and the peat is laid out alongside the cableway, each row being the length of one side of the square. As soon as a row is completed the tracks are moved over, the spreader reversed and a new row started.

Plant No. 2, the Moore plant, is identical in principle with the Anrep plant as regards excavation and masceration, but differs from it essentially in the method of spreading.

In this plant, the peat, on leaving the mascerator, is delivered to an endless conveyor belt carried by a fabricated steel arm, 160 feet in length; this extends from the plant platform proper on the opposite side from the excavator, and is at right angles to the line of the ditch. The wet peat is carried on this belt conveyor until it reaches a deflector, which turns it into the spreader. The plant itself is transported on two caterpillars and the belt conveyor arm on a third caterpillar placed 100 feet from the platform proper. The spreader is attached to the belt-conveyor arm and is operated by a chain belt drive which runs through the conveyor arm, operating also the third caterpillar. In setting out one row of peat, the plant travels the full length of the ditch, and as soon as the row is completed, the spreader is moved the distance of its own width along the conveyor arm and attached on the opposite side, so that another row parallel to the first and adjacent to it is laid out as the machine returns along its working face. These operations are repeated until the field is covered. The spreading field is therefore somewhat over 160 feet in width and is the same length as the ditch. To work to the best advantage, it is necessary that the ditch should be of sufficient length so that when the field is completely covered, the first row spread will be ready for harvesting. Much of this first row will already have been used in operating the plant so that the rest can easily be removed and the spreading of the first repeated. While this is being done the remainder of the belt conveyor can be used to convey the peat from row No. 2 to the permanent track at the end of the conveyor, where it is delivered into small open cars and taken in trains to the shipping platform. Mr. Moore's plan in this method of manufacturing peat is to make the handling of the material as far as possible mechanical, and in this he has succeeded to a marked degree. Thus the labour required to operate the No. 1 plant is from 13 to 15 men, while No. 2 plant requires from 5 to 7 men. The obvious disadvantage of the No. 2 plant is that it requires a very long ditch to keep it operating continuously, but this on the other hand will be overcome as the length of the arm is increased.

The capacity of each plant is approximately six tons per hour and the best results are obtained by working two shifts per day.

WORK ACCOMPLISHED AND PROJECTED

The Alfred bog was chosen as suitable for the installation of the two peat machines for the demonstration of the practicability of manufacturing peat fuel commercially in Canada, as it already has one working face prepared and a drying field which was partially drained. This bog is situated on the main line of the C.P.R. between Ottawa and Montreal.

The analysis of the peat content of the portion of the bog that is being worked is:

Volatile matter	69%
Fixed carbon	24%
Ash	7%

Heating value, 9300-9500 B.T.U. per lb. (absolutely dry peat).

The season at Alfred, which is similar to that of the greater part of southern Ontario, consists of from 100 to 120 days.

The price, in carload lots, at which the peat is being sold f.o.b. cars at Alfred station, is $3.50 per ton. This allows for a reasonable profit over and above the cost of production, and includes an allowance of 10% for depreciation and 10% for amortization.

Ottawa and Montreal afford a ready market for the product from many such machines as those built for the Peat Committee.

It is not expected that peat fuel will entirely replace coal even where peat is most plentiful; but for certain special uses it has advantages over coal and if used for these purposes will materially help to overcome the fuel shortage.

Peat can be used to advantage in open grates and in cooking ranges, but is not recommended for use in furnaces except in the autumn and spring when light fires are needed or in conjunction with coal when a coal fire has to be hurried.

The programme planned for the coming season is to work both plants to capacity and market the product, showing thereby what can be done on a commercial basis. It is also proposed to test a small 3-man machine which is now being built. It is hoped that such a machine will be suitable for a great many small bogs of comparatively shallow depth throughout Ontario and Quebec that are too small to be worked with the larger machines.

FUTURE PROSPECTS FOR OIL AND GAS PRODUCTION IN ONTARIO*

By M. Y. WILLIAMS

Annual General Meeting, Toronto, March, 1920.

To-day, Canada produces about 0.06% of the world's petroleum, and Ontario produces by far the greater part of the Canadian output. According to the report of the Mines Branch of the Department of Mines on the Mineral Production of Canada for 1918, the total production of petroleum for the Dominion was 304,741 barrels, the greatest yield since 1910. Of this, Ontario produced 288,692, Alberta 13,040, and New Brunswick 3,009 barrels. The value of the Canadian production for 1919 was $744,677, but the value of the imports of petroleum for the same year was $29,351,196. Under these circumstances, it is clear that any increase in production is of great national importance.

Five geological formations in Ontario have produced petroleum in commercial quantities; these are, in order from oldest to youngest, the Trenton, the Medina, the Guelph, the Salina, and the Onondaga (Corniferous). In general, productivity has been in inverse ratio to the relative ages of the formation; thus the Onondaga was the earliest developed and has been the steadiest producer and the most uniformly productive formation in Ontario. The oil of Petrolia, Oil Springs, Bothwell, Dutton, Mosa, and many smaller fields, is derived from the Onondaga limestone or the sandstone at its base. The long life of these fields has been the wonder of the petroleum geologist, some wells having produced commercial quantities of oil almost continuously for 50 years.

The lower beds of the Salina formation are dolomite, and are so similar to the upper beds of the Guelph formation upon which they lie, that it is difficult to determine where the boundary should be drawn. It is probable, however, that the lower part of the Salina contains the oil horizon of Pelee island and of

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Mersea township. The oil horizon in the Tilbury field is near the contact of the Salina and Guelph formations, but the gas of the Tilbury field has clearly been derived from the Salina formation.

The Guelph formation has probably furnished a part of the oil of the Tilbury field, and appears to have produced all the oil of the Wheatley field.

The Medina, although a good gas horizon, especially in the Niagara peninsula, produces petroleum in commercial quantities at one locality only, namely, in the Onondaga field near Brantford.

The Trenton formation, notwithstanding its production in Ohio, has, until the last three years, been very disappointing in Ontario. The tempting oil showings of Manitoulin island, which caused considerable activity in the sixties, in the nineties, and again in 1912 and 1913, were struck in the top of the Trenton formation. Some oil has also been found in this formation near Milton, and oil 'shows' have been reported from it at various other localities. The Dover field, however, is the only one in Ontario from which commercial quantities of oil have been obtained from the Trenton. In this field the top of the Trenton lies, on an average, about 2,900 feet from the surface, and the oil and gas which accompany it occur at varying depths between 120 and 290 feet in the formation. As already shown by the writer, as a result of the almost complete absence of water in the formation, the oil is found in a basin or syncline. In Ohio, the Trenton is, as in Ontario, a close-grained limestone, with beds of crystalline magnesian limestone and dolomite occurring through it. It is in these beds that the oil and gas occur in company with salt water. Without the concentrating influence of water, the accumulation of the fluids of lighter specific gravity is slow and imperfect.

FUTURE PROSPECTS OF SOUTH-WESTERN ONTARIO

The unexplored, or imperfectly explored, areas underlain by formations which are likely to contain oil or gas, will be considered in order, beginning with the youngest and best known.

The Onondaga limestone has sufficient cover to retain oil and gas in Kent, Lambton, and the western part of Middlesex and Elgin counties. Elsewhere the drift rests directly on the eroded surface of the Delaware or even the Onondaga limestone, and any oil contained in the formations has escaped. A little oil has been found in Essex county at Belle river, but this occurrence is the exception. Within the region of its probable productivity, the Onondaga limestone has been penetrated by thousands of wells, as not only those drilled into it to test its possibilities, but also wells drilled to test deeper formations, have passed through it. Under these circumstances, it is scarcely probable that any very large oil pools have been overlooked. The opening up of the Mosa field in 1917, after it had been bordered by drill holes, should, however, serve to check precipitancy in the drawing of general conclusions. The writer, during his work on the Ontario oil fields, has directed attention to some localities that would warrant further prospecting. Some of these have already been drilled and found to be of little value. Others are encouraging, and some production is expected from them.[1]

The Salina formation, as already stated, has produced much oil and gas. This formation has plenty of cover west and south of a line joining Southampton, Harriston, Stratford, Woodstock and Buffalo, but can best be prospected along with the underlying Guelph formation, from which it cannot be differentiated with certainty. On Pelee island, however, there is sufficient evidence of an accumulation of oil to encourage prospecting the Salina.

The Guelph dolomite, or the upper part of what is known to the drillers as the 'white lime,' is overlain by the Salina dolomites and shales and has sufficient cover to retain oil west and south of a line joining Southampton, Mount Forest, Kitchener, Brantford and Welland. Away from the vicinity of the oil- and gas-fields, the Guelph formation has been very imperfectly explored. Except where large accumulations of

[1]The account of these oilfields is to be found in the Summary Reports of the Geological Survey.

salt or gypsum occur, the Guelph is approximately parallel to the overlying formations, and hence suitable structure may be worked out either from surface exposures or from the information derived from previous drilling. Where salt is found in thick lenses the underlying formation is likely to be depressed, and so the salt regions have generally proven to be less favourable for oil and gas in the lower formations than the areas where the salt is either thin or absent altogether.

The Medina formation of the Niagara peninsula has, generally, been gas-bearing, but few occurrences of oil are reported from it. The one occurrence near Brantford appears as an exception, and judging from the knowledge obtained from the deep drilling in the western part of the province, the Medina is not to be considered as a promising oil producing horizon in Ontario.

The Trenton formation, as already stated, has been one of the best oil producing formations in Ohio, and is yielding oil at present in the west part of Kent county, besides containing small quantities of oil on Manitoulin island and elsewhere, as shown by previous drilling. In the cases of the small showings of oil, the accumulations were found a few feet below the black shale beds of the Utica, which rest upon the Trenton limestone. The areas where the Trenton may be prospected with hope of success are limited to those regions where the Utica is present in whole or in part, that is, from the outcrop of the Utica westward. A somewhat sinuous line connecting Port Hope with the foot of the escarpment west of Collingwood, forms the approximate eastern boundary of the Utica, and the area to the west may be considered as available for prospecting the Trenton. The eastern part of Ontario lying between Brockville and Ottawa is in part underlain by the Trenton formation, which, in the vicinity of Ottawa, is overlain by the Utica shale. This area, however, has not proved productive of either oil or gas in commercial quantities, one reason probably being that where covered by the Utica the formation is so faulted as to have lost what oil or gas it once contained. Gas in small quantities is commonly reported from this region, however, and considerable drilling has been done.

OIL PROSPECTS ON MANITOULIN ISLAND

Manitoulin island may well be considered by itself, as there the conditions are in some respects favourable for large accumulations of oil. The whole island is underlain by Trenton limestone, which outcrops at the surface at many places near its northern boundary. It is probably nowhere more than 900 or 1,000 feet below the surface even on the south side of the island.

Early writers called attention to the series of anticlines and synclines, suggested by the physiography of Manitoulin island, and although these are not as well ascertained as some maps have depicted them to be, decided structural features are known to occur. Considerable quantities of oil were struck near Wekwemikong, Manitowaning, Gore Bay, and Sheguiandah, and this oil appears in all cases to have been accompanied by water. The writer has not been able to learn whether the water was normal brine such as usually accompanies oil, or whether it was sulphur water. In either case it appears possible that the water entered the formation from the outcrop, dissolving salt and other substances from the beds through which it flowed. This seems more probable as the Trenton is nearly dry elsewhere in Ontario.

The drilling already done on the island has been limited almost entirely to the north side, notably near Wekwemikong, Manitowaning, Sheguiandah, and Gore Bay. These locations are all but a short distance from the outcrop of the formation beneath the water of the North channel. Under such conditions, the oil accumulations may scarcely be considered as normal. The south and middle portions of the island have never been drilled, although there is one unfinished well situated at Providence Bay. The southern part of the island is sufficiently far from the outcrop of the Trenton to provide against loss of oil or undue interference from water entering the outcrop of the beds. Until drilling is done in the untested portions of the island, the possibilities of the region cannot be definitely estimated.

THE JAMES BAY SLOPE

With the steady narrowing of the unprospected territory in the older parts of the province, attention has been directed to more distant fields, and especially to the large areas of Palaeozoic limestones, shales, and sandstones lying to the south and west of James and Hudson bays. The writer visited this basin in the summer of 1919 in order to study the oil possibilities. He followed the Mattagami and the Moose rivers down to Moose Factory and returned by the Moose and Abitibi rivers. The examination showed that of the formations which are likely to contain oil, the Onondaga limestone and the Salina shales, outcrop south of James bay. Previous geological investigation to the east suggests that the Guelph and Trenton formations probably underlie the outcropping formations. The Onondaga limestone is covered at some localities by Hamilton clay shale and the Ohio petroleum bearing shales, but the shale areas are small, and unless larger areas are found, it is hardly likely that the Onondaga will prove to have retained any oil accumulations of importance. The structure exposed in the Salina shale on the islands of the Moose river is favourable for oil, both in the Salina and in the underlying formations. The centre of the Onondaga outcrops is west of Moose river (between it and the Albany river), and it is probable that the sedimentary deposits are deepest near the centre of the basin. Somewhere in this central region, test holes would solve some of the problems relating to oil occurrence, even if oil were not found. The thickness of the deposits and the kind of formations present would be discovered by the first well sunk to the 'granite.' With each additional well, more territory would be prospected and by means of the necessary levelling and examination of samples, the structure of the area tested could be worked out, and new wells located on the strength of the knowledge obtained. Until wells have been drilled, the Palaeozoic basin of James and Hudson bays will remain interesting only by reason of its potentialities as regards petroleum resources.

OIL SHALES

In view of the increasing scarcity and increasing cost of petroleum, oil-bearing shales are now more than ever regarded as a future source of oil. The Ohio shales, which, in southern Ontario and elsewhere, contain oil, outcrop on the Mattagami river just north of Speight's base line and along the Long rapids of the Abitibi river. The outcrops on the Mattagami river are limited, but fossil evidence would suggest that the beds outcropping are near the top of the shales, and in that case the shales may be of considerable thickness, and of greater extent than might otherwise be supposed. The outcrops along the Abitibi river occur as a syncline about 1½ miles wide near the upper end of the Long rapids, and as an area of north dipping beds about half a mile wide, near the foot of the rapids. The total thickness of the beds represented is estimated as being not more than 100 feet. Small samples of reasonably fresh shale taken from the outcrop, show, upon analysis by the Fuel Testing Laboratory of the Mines Branch of the Department of Mines, a variation of from 3.9% to 5.5% of crude petroleum content. The samples were not large enough to make distillation tests, nor was the specific gravity of the oil ascertained, but assuming the specific gravity to be 0.86 the production would vary (in round numbers) from 9 to 13 gallons per ton of 2,000 pounds. Previous tests by the Fuel Testing Laboratory on the Ohio shales of Kettle point, Ontario, indicated a yield of 10 gallons of petroleum per long ton of shale; and 15 samples of New Brunswick oil shales gave an average of 48 gallons of oil per long ton of shale. Under these circumstances, having regard to existing transport and market conditions, it is clear that the oil shales of the James bay region cannot be worked profitably.

DISCUSSION

MR. LOUIS SIMPSON: The question of our oil resources should interest every citizen of Canada. I believe that Canada should produce, as far as possible, its own fuel, and I am satisfied that Canada cannot supply its own fuel if we are to depend upon coal. We cannot bring the coal that is required in the province of Ontario and in many parts of the province of Quebec from Nova Scotia, because the cost of transportation is too great. We cannot bring the coal that we require in Ontario or in Quebec from the far West, for the same reason: therefore, if we have to supply ourselves with our own fuel, we must look to sources other than coal.

It has seemed to me to be a very serious matter that we are sending out of this country one hundred million dollars per year to pay for coal. If you add to that the amount we are sending out to pay for the oil we are bringing into this country, the total reaches the astounding total of nearly one hundred and fifty million dollars a year. If we retained that money in Canada to spend amongst our own people, we would then have an opportunity to pay the debts imposed on us by the great war.

Unless we start to develop more economically and more properly the present undeveloped resources of Canada, I do not see how we are to pay our debts. I refer now in particular to our oil resources. It has been known for a long time that there are millions of tons of rich oil-bearing shales in the provinces of Nova Scotia and New Brunswick. There are also millions of tons of oil shales in other provinces, but not so rich possibly as those of Nova Scotia and New Brunswick. The people of New Brunswick have been moving to develop the oil shales. The only thing that stood in their way was that, for some reason unknown to myself or to any of my friends, the government of this country persisted in requiring that all the machinery required in this industry—there was none of it made here—should be subject to the heaviest import duty.

The same thing applies, though in a lesser degree, to those who have been trying to increase the production of oil in Ontario. They have had to pay very heavy duties on the pipes they use, and yet pipes for this purpose are not manufactured in Canada.

It is argued that were we to ask the government to remove the import duty on machinery not made in Canada, we would be likely to endanger the protective policy that has built up our

large industries of steel and coal. That is not a logical argument. It is just as easy for the Government to say that all machinery not made in Canada shall be admitted free of duty, as to do what they have done in the past. We know that at Sydney they have allowed the machinery that is required to build the new steel plate mill to come in free of duty. That did not kill protection. We know at Niagara they allowed the electric shovels which were required for the power plant there to come in without duty; that did not kill protection. We know at Montreal, in the harbour works, they allowed the bridging steel to come in without duty; that did not kill protection. Therefore, I cannot see why we should have any hesitation at all in asking the government to help the oil industry.

MR. E. P. ROWE: I have, in a small way, been engaged in the development of the deeper oil and gas wells, penetrating the Trenton formation in Ontario, for the past few years. Fifteen years ago I was instrumental in making a number of borings into the Trenton in the counties of Grey and Bruce, whose small production of gas was developed and traces of oil were obtained at a depth of 1,140 to 1,400 feet.

During the past two years I have been carrying on drilling operations in the new Trenton oil field of West Dover tp., of Kent co., Ontario, to which Dr. M. Y. Williams has just referred. All borings in this field, to date, indicate the occurrence of the oil in a synclinal fold at approximately 385 feet below the summit of the Trenton formation.

While the wells in this area have not delivered such quantities of oil as the Cretaceous rocks produce, as pointed out by Dr. Dowling, we must not lose sight of the fact, that the Trenton oil is of very superior quality, running high in gasoline, kerosene, and lubricating oil, and the present Dover wells are profitable producers.

The wells in Dover tp. enter the dolomite of the Trenton, where gas and oil are obtained at depths varying from 3,000 feet to 3,310 feet; and several horizons of porous dolomite occur between these levels. Some traces of oil have occurred as deep as 3,500 feet or about 600 feet from the summit of the Trenton.

Many of the early borings for oil in the Trenton in Ontario stopped after penetrating the first 100 feet of this formation, and finding nothing have probably condemned territory, that, in the light of these recent developments, may yet prove productive at the deeper horizons. I have listened with great pleasure to the papers by Dr. M. Y. Williams and Dr. Dowling and feel sure that greater attention will be given the Trenton in future prospecting for oil and gas in Ontario.

THE OIL POSSIBILITIES OF WESTERN CANADA[1]

BY D. B. DOWLING

Annual General Meeting, Toronto, March, 1920.

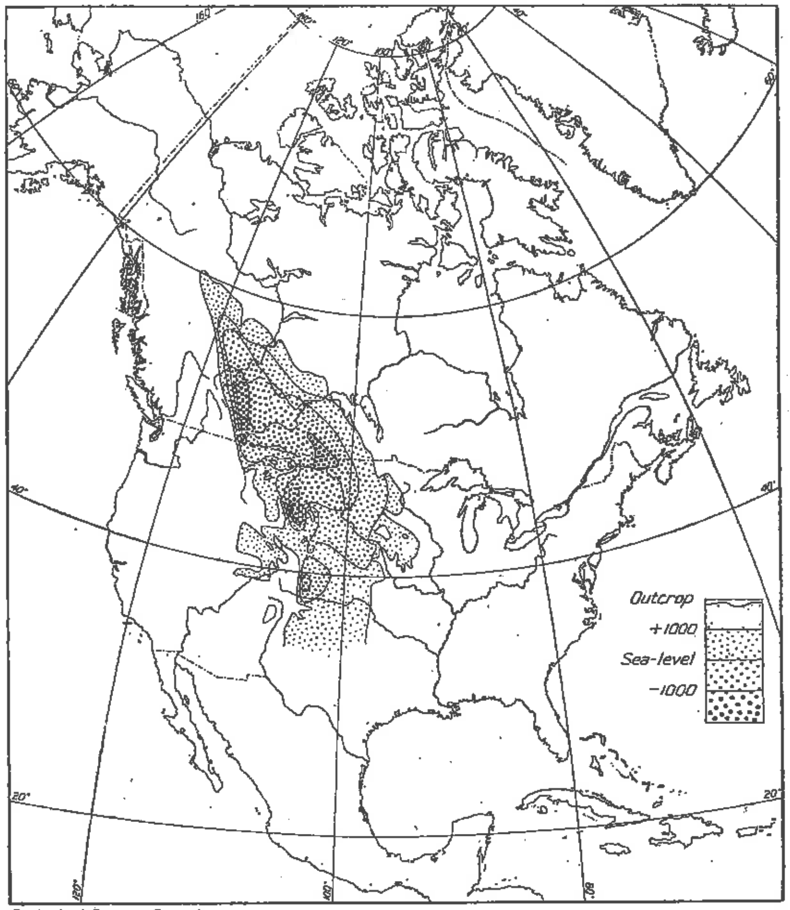
The wonderful increase in the various uses to which the products of petroleum have been applied within the last few years has called for an intensive search world-wide in its extent for new fields.

The potentialities of Western Canada in respect of oil resources have not been considered great hitherto, but the large production that has been realized from the Cretaceous deposits of the western American plains has stimulated prospecting in the northward continuation of these beds and also in the underlying Palaeozoic deposits which are found in the Dominion of Canada.

The structure of the oil-bearing area which lies in front of the mountains of Wyoming and Colorado has no exact counterpart in Canada inasmuch as the flexures in the beds, in the two States mentioned, are largely due to uplift as well as to the effects of tangential pressure, while to the north the deformation is largely due to tangential stresses. There is thus a more complicated structure in the American fields than in those of Canada, and to this is probably due a greater alteration of the petroleum forming substances and probably a greater migration through the measures to porous retaining beds. It is noted, also, that several horizons found to be oil-bearing in Wyoming have, in Canada, yielded but slight indications of oil. In the lower horizons, we have evidence of the presence of a heavy oil at several places, but, as the records are not numerous, a general statement as to the extent of this deposit is all that can be offered. Results of a study of the structure, together with facts obtained from the few drilling records, are embodied in the following observations:

[1]Published by permission of the Directing Geologist, Geological Survey of Canada.

Oil-saturated sands have been found in the lower part of the Cretaceous at the following localities:



Geological Survey, Canada.

Diagram showing basin occupied by Cretaceous sediments

Scale of miles
100 0 500 1000

1. At the southern edge of the province of Alberta, near the Sweetgrass hills, thick vaseline-like saturated sands.

2. On Milk river in the Beaver well, small flow of oil with artesian water.

3. In Etzikon coulee in the United well, sands saturated with very heavy oil.

4. Wells Nos. 2 and 4 at Viking, showings of oil in gas sands.

5. Oily shales at the bottom of the Morinville well.

6. The McMurray sands exposed on the Athabaska river, called generally tar sands.

7. Peace River, below the mouth of the Smoky river, thick oil in sands bored through.

All these occurrences appear to be in sands of approximately the same age and may be considered as indicating a rather extensive sheet impregnated by a heavy oil—in many places too thick for commercial extraction except where it occurs at the outcrop. The areas in which commercial exploitation might be suggested are those which surround the deep basin of the Alberta syncline, and include the outer foothills, which are on the western edge; the upraise at the south in the Bow Island anticline; and the northeastern margin of the basin as far south as the depression leading to the eastern basin which centres about Estevan. In this eastern part it is not known that the oil horizon of Alberta is present.

The occurrences in the foothills do not seem to prove the continuance of the oil deposits as far as the mountains. The oil, that is there found in association with wet gas, has the appearance of having been condensed by some distillation process; that is, it is not crude oil. Its formation might be hypothetically assumed to be due to the great pressure and moderate temperature to which the heavy oil of the sheet just mentioned would be subject, especially in the part deeply depressed in the Alberta syncline. The volatile constituents of the oil would follow the short limb of the syncline upward to zones of lower pressures and temperatures, where they would be condensed. The condensed portions reaching the overturned edge would there be trapped, as appears to be the case in the Turner valley anticline, and the gas would have vapours of the lighter oils still in it.

Renewed interest is being taken in the structural features of the outer foothills as being the edge of the Alberta syncline. Where this edge is marked by anticlines not too deeply buried, prospect bores are to be tried by the larger interests in the hope of striking light oils or of finding gasoline vapour in the natural gas.

The production of the foothill areas last year was about 13,000 barrels of a light kerosene oil containing probably 60% of gasoline. Small stills are installed at three of the wells, the oil being 'broken up' for domestic use; at the Calgary Petroleum Products Company's wells, absorption plants are installed and from the natural gas it is expected that up to 30 barrels of light gasoline per day will be obtained.

In connection with the foothills area, an area forming part of the mountains near Waterton lake merits consideration. In it the rocks are very much older than any known oil-bearing rocks, but in their bedding planes are distinct evidences of oil. As the oil does not seem to be indigenous to the rocks, it has been assumed that it comes from the Cretaceous beds that have been overridden by the overthrust of this mountain mass onto the plains. It has been assumed that the plane of the thrust fault is at a low angle and that the shortening of the crust reached large proportions, so that an extensive area may be underlain by rocks from which the oil might be derived. This assumption, which has elements of probability, would predicate an overlap of between 20 and 30 miles in width; but in the mountains to the north of this no single thrust reaches this proportion, and hence it seems necessary to modify the original theory.

The presence of the oils in rocks far above heights to which it could be borne by general water saturation, and its light specific gravity, suggest that the transference was gaseous,

probably along with vapours distilled from carbonaceous material in the over-ridden Cretaceous. The overlying beds are not badly folded and form a rather large covering mass that would retard the escape of the vapours.

The form of the cover is a synclinal trough, the lowest point being near the watershed which is here only a few miles from the eastern edge of the mountains. The trough is edged by anticlines and oil seeps are found near these, both east and west of the summit. If the ascending vapours penetrated the base of the syncline, where there would probably be open fractures, and followed the beds in both directions it might be expected that the lighter oils would be found at a greater distance from the centre of the syncline than the heavier oils, as they would have a greater penetration and longer life in the gaseous form. The facts seem to be in accord with this hypothesis for the seeps to the west, about 11 miles from the centre of the syncline, yield oil of very light gravity (40°-42° Baume), while those to the east, four miles from the same point, yield oil much heavier (30° Baume).

The distillation process that might be presumed to have been active at one time seems to have declined or become inoperative at the present time, there being little evidence of the escape of gas or of gas pressure in the wells now bored. These wells appear to be merely draining the beds above their level; and, if this is true, it follows that prospecting may have to proceed on the assumption that the saturation is not being supplemented by present emanations of moment; that is, oil derived from the draining of the beds by gravity from each side of the anticlines should be the object of drillings.

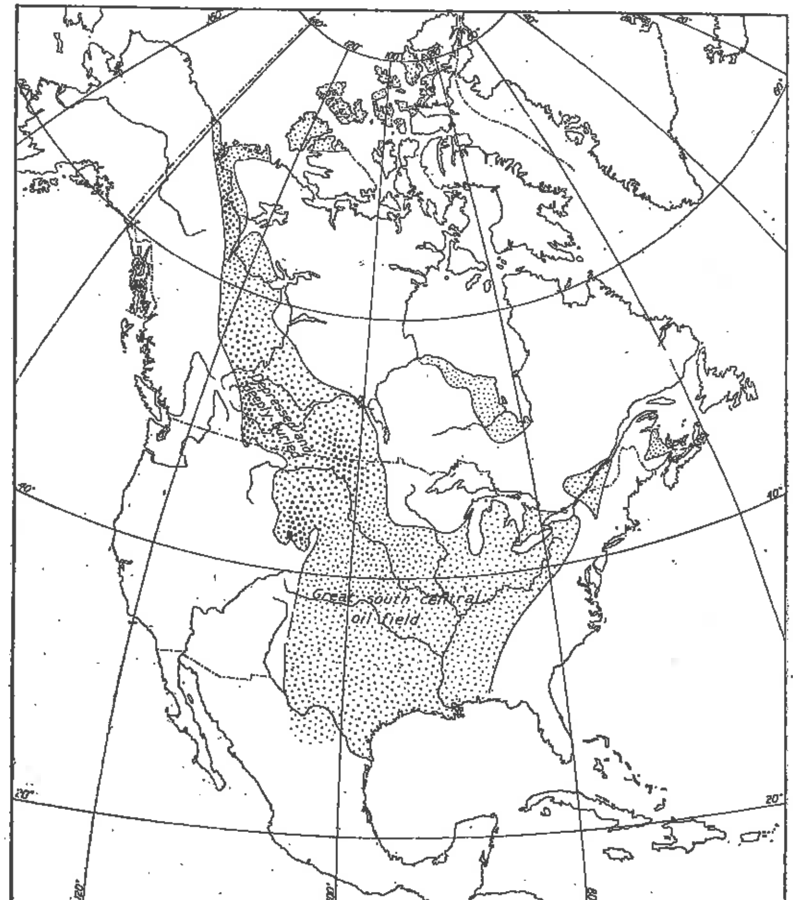
The supposition outlined above, of a gaseous origin for the oil, introduces another interesting possibility, namely, that the sand beds of the Blairmore formation, which no doubt underlie the Cretaceous of the plain in front, may have also acted as passage beds and that some of this oil may have been transferred through them to retaining structures under the plains in the folds which parallel the outer edge of the over-thrust mountains.

On the plains most of the prospecting has of late years been centred in the Peace River valley, where several wells averaging 1,100 feet in depth pass through sands impregnated with heavy oil. The flow is necessarily slow and, although the presence of oil seems to be proved, no production has been attempted, possibly on account of the trouble with water, which can generally be attributed to insufficient or defective casing and to lack of restraint on the part of operators in drilling through the oil sand into the water-bearing bed which lies below it.

The direction of the extension of the oil horizon depends on the attitude of the beds, and, as there are few exposures of the outcropping rocks and few bore holes by which to trace it, the general features only can be indicated. The gasfields at Viking and the Peace River oilfields are situated on a structural terrace which consists of a flattened strip on the eastern slope of the Alberta syncline. The flattening may in places even reverse the dip and form low anticlines. The western edge of the terrace is marked by an abrupt change of dip, and at the point where this line swings to the southwest, from near the border of Saskatchewan, surface crumpling is quite evident. The position of the oil sands in this part of the terrace is probably lower than desirable, but the flattened area or structural terrace widens considerably, and there is always the possibility of local flexures and rolls in the beds, providing local catchment areas for oil and gas. With this possibility in view, a bore is being put down near Czar, Alta., on Ribstone creek. Boring near Lesser Slave lake is being undertaken with the hope of extending the Peace River field. The information at present available is not sufficient for the determination of the exact direction of the line along which the sands will be found to carry oil between Viking and Peace river; its position can be determined only by future boring in the area.

The older deposits of the central part of the basin probably present the same succession of beds as in the southern and eastern fields; but the deposits of the eastern and northeastern margin show (for this central part of the continent) a geological history very different from that of the areas more nearly

connected with the outer seas. This has no doubt had an influence in the non-retention of oil in the beds of the outer margin of the basin. Boreholes in Manitoba and on the Athabaska and Peace rivers in this outer margin give negative results in the search for oil. There is evidence, in the absence



Geological Survey, Canada.

Diagram showing basins occupied by Palæozoic sediments

Scale of miles

100 0 500 1000

of early Palaeozoic sediments and also in the absence of middle Devonian formations along the northeastern outcrop, that the sea margin of this long period was a fluctuating one and that possibly the interval from Devonian to post-Carboniferous time saw a steady withdrawal of the sea or a slow elevation of the land. During this long lapse of time it is quite possible that large areas suffered denudation as the mantle did not consist of massive beds and would be easily fractured by differential elevation.

The retaining of fluid hydrocarbon in the elevated and probably thinner parts would therefore be extremely problematical. The central part of the continent was then depressed beneath the muddy Cretaceous sea, and possibly the cover of shale then deposited over the remaining Palaeozoic beds has helped to retain in them some of the original oil. The general rise of the continent in Tertiary times brought the edge of the basin—the part now exposed—far above the sea level and it would be natural to suppose that the drainage of the oil since that time would tend toward the basin.

In the Manitoba portion, where the Devonian section contains beds that are elsewhere oil-bearing, it should be noted that these beds are there about 800 feet above sea level. Their westward continuation at Moosejaw would be at least 1,000 feet below sea level so that, provided the wastage before Cretaceous times did not exhaust the beds, the cover of Cretaceous shales and the limestones of the Upper Devonian might retain the oil drained westward. In the absence of sharply undulating structure the oil might be found on this slope as low as the line of water saturation, which is not known for this basin but might be assumed to be as low as sea level.

The prospecting of this horizon would mean drilling through the Cretaceous shales and would be confined to a strip of country not more than about 60 miles in width, measured from the edge of the Cretaceous escarpment.

Along the northern border the problem is somewhat different in that it is not known with any degree of accuracy

where the supposed oil-bearing beds, which are exposed in Manitoba, extend. The outcrops of the Palaeozoic at the contact with the Archean show overlapping of the beds; that is, at different points Devonian, Silurian and Ordovician sediments respectively are found in contact with the pre-Cambrian. Fluctuations of the margin are also shown in the individual beds of the formations; for instance, the Middle Devonian is absent in the exposures on the Peace river below the Cascades, and the upper Devonian is resting on the gypsum beds of the Silurian.

To the southeast, the section is probably complete and the location of the oil-bearing Middle Devonian beneath the cover of shales and Upper Devonian limestones is the first problem to be solved, since on its position depends largely the occurrence of oil in the Devonian. Where it can be found near the margin and at a convenient elevation there would be some hope of a future production.

As it is not certain that the oil beds are present at any point in the northern area, drillings in the Peace and Athabaska valleys are in the nature of explorations; but without such adventurous prospecting little will be learned.

Northward of the granite spur which extends westward between Athabaska and Great Slave lakes, another embayment of the ancient sea is encountered. In this the deposits show a sedimentation somewhat similar to that in Manitoba. As these rocks are not deeply covered by Cretaceous sediments and as the basin did not become greatly deepened by the mountain building stresses, exposures of rocks not far above the oil-bearing beds are found much farther from the margin than in the basin beneath the plains. These exposures also include the oil-bearing beds which are not exposed in Alberta and Saskatchewan, and fortunately can be reached by the drill over a much larger area. The western and northern portion is folded into definite ridges known as the Franklin mountains and offers many points of attack where the possibilities for oil pools would seem to justify drilling.

DISCUSSION

MR. DOWLING: As an item of interest rather than as part of the programme, I might mention that in the southern corner of the Rocky Mountains, we have an area in which we are getting some oil in the Cambrian or pre-Cambrian rocks. The oil production so far has not been very great, but it is promising. The question is a puzzle to the geologists, and they are suggesting that the block in the pre-Cambrian rock has been shoved over the prairie and that it has over-ridden some Cretaceous rock that has provided this oil, and that the oil has gone upward into the pre-Cambrian rock. This has been studied last year by some of the geologists of the Imperial Oil connection, and they figure that possibly the suggestion put forward is quite a feasible one. However, the character of the oil bears out the suggestion. It is a very light oil that is found on the British Columbian side of the divide; it is not a crude oil, it is a light coloured kerosene, and the oil on the Albertan side is a dark coloured kerosene, showing probably a little of the heavier oils. The thrust might very well have been enough to depress the ends of the Cretaceous formations downwards to a heated zone, and the distillation of the carbonaceous material might have formed dry gas or wet gas, and so on, and brought up with it some of the distillate products which penetrated these rocks and are now found as oil springs. The wells do not run. They simply are draining the oil from the rocks on all sides. The oil is found altogether in the cracks and bedding planes of the rocks.

MR. L. SIMPSON: Are there any vapour springs in the vicinity?

MR. DOWLING: There is one at Kinetta Lake.

DR. M. Y. WILLIAMS: Mr. Chairman, it is very interesting to know that oil is found in the pre-Cambrian in Alberta, occurring in the cracks of the pre-Cambrian rather than in the bed themselves.

It is sometimes the duty of a Government Bureau to keep people from spending money foolishly, and while it is sometimes

difficult to get oil concerns to drill a suitable sized well, it is at other times difficult to get them to stop drilling when they have gone far enough. I have in mind a case in Wellington County where a well was put down into the Pre-Cambrian formations.

We have tried in vain to persuade people that they were wasting their money by throwing it into this well. I think it is better for us to explain to people who are prospecting that even if some oil is found in pre-Cambrian rocks, it very rarely occurs in commercial quantities. Geologists at the present time are almost unanimous in their opinion that oil occurs in commercial quantities only in stratified rocks; and I think the cases such as Mr. Dowling noticed, are the few exceptions to the rule and are explained most satisfactorily by the explanation already given: namely, that the oil has passed from the stratified rocks into the crevices and basins in the harder and older rocks.

SOME FEATURES OF THE APPLICATION OF GEOLOGY TO MINING AS PRACTISED BY THE GEOLOGICAL DEPARTMENT OF THE ANACONDA COPPER MINING COMPANY

By Paul Billingsley

Western General Meeting, Vancouver, November, 1919.

The geological department of the Anaconda Copper Mining Company, of Butte, Montana, has become a very effective and useful branch of the Company's organization. The character of the Butte orebodies affords great scope for the application of geology to the solution of mining problems, and the department has, therefore, been able to demonstrate its usefulness very convincingly.

The Butte ores occur in rather narrow veins, from two to twenty feet wide, in quartz-monzonite. These veins are cut by at least six successive series of faults, some of the earlier of which also contain ore-shoots. Thus, either in plan or in section, the veins are seen to be broken and displaced at frequent intervals, the direction and extent of movement varying with each fault, but the movement was usually sufficient to throw the ore outside of the drift or raise. When it is considered that the high-grade character of the ore makes the recovery of every fragment of vein important, and also that many of these faulted fragments are too small to be hit by haphazard cross-cutting, the necessity for an understanding of the structure becomes apparent. The mine operator requires to know promptly which way to turn a working; he is not much interested in microscopic details of rock alteration or in theories concerning the source of mineral solutions.

Organization.—In 1900, an early date in the annals of mining geology, H. V. Winchell and D. W. Brunton organized a geological department for the Anaconda Company, realizing that only the complete compilation of geologic data would solve the problem of fault-structure to the degree necessary to aid mining operation. The work thus begun has been greatly

extended and also improved in detail under the direction of Reno H. Sales, chief geologist since 1906. F. A. Linforth, who in recent years has been in special control of the work in the Butte mines, has contributed many of the refinements of method now in use. His paper presented at the Butte meeting of the A.I.M.E., in August, 1913, gives an excellent outline of the work as conducted at that time.

Requirements of Mining Geology.—The Anaconda Copper Mining Company's Geological Department has thus, from its formation, had for its prime objective the helping of the mine foremen and superintendents by affording them a knowledge of the structure of the ground. Its members have realized that the geologist should be the servant of the man who is responsible for the work of the mine. As a result of experiment and progress, covering a period of twenty years, it has come to regard the following as essential:

First, friendly co-operation and mutual respect between mine foremen and geologists. The geologist must be familiar with the foreman's plans in order to advise him intelligently, while the foreman must appreciate the value of geologic advice, and be prompt to seek it when in difficulty.

Second, prompt geologic examination of all new workings, in order that suggestions may be made without loss of time. Much useless work is thus avoided.

Third, accuracy in underground observation. Details of structure or of mineralization must be recorded exactly as they appear in the ground—big faults distinguished from minor slips, sequence of veins and faults at intersections properly interpreted, important minerals such as secondary chalcocite, or primary chalcocite, observed and noted. To take notes properly as a rule involves long study under very unfavourable conditions of air, water, ground, etc. Hazy or indefinite notes are only less useless than erroneous ones. If the geologist is not clear in his own mind as to which way the mineralization trends, or as to which of two faults is the intersected one, and which the intersector, he should settle the point by study in the mine, with the evidence before him, and not leave the facts indefinitely stated in his notes.

Fourth, immediate platting of notes, (and particularly of structural features,) on general geologic maps and sections, in order that the new evidence may be studied in its proper relation to that already accumulated. While it has been affirmed that decisions as to detailed structure must be made underground, it does not follow that general ideas can be correctly gained from that viewpoint. It is one of the most vital features of the work of the Geological Department that such general ideas are formulated only from a study of *all* available details, each detail being as accurate as it can be made by careful work with pick and compass. It is particularly important that one should not let his judgment underground be influenced by what he expects to find. Often a big fault does not come where it is expected; the reason will be discovered in due time.

Fifth, a smoothly-working system whereby conclusions reached by a study of geologic data are translated promptly into terms of mine operations.

Methods.—The means by which these fundamental ends are obtained have been described by Linforth, but may bear repetition here.

Co-operation with foremen is a matter of state-of-mind. The geologist must not be above such co-operation. Often the foreman knows more about the geology of his mine than he can express in the geological language of science. Generally, he can teach the geologist much about mining methods.

Promptness has been attained by employing a sufficient number of geologists so that each man can 'cover' the territory assigned to him in somewhat less than one month. Many foremen would like more frequent visits from the geologist. Important faces are visited after each round. It has been found that while a geologist, experienced in the Butte work can map correctly about 25,000 feet of workings per month, this gives too little time for the study of special problems in

the mines, and does not permit of a sufficient number of trips with the foremen and bosses. At present, the members of the department perhaps cover about 6,000 feet of workings each per month.

Underground notes, on loose-leaf sheets, are entered on a scale (usually either 20 or 50 feet to the inch), suitable to the degree of detail desired. Coloured pencils are used. Red represents vein mineralization, the ore being shown in a darker colour than the gangue. Blue is used to indicate fault gouge, a light blue signifying slightly crushed rock, and a heavy dark blue strong clay selvage. With these conventions, it is possible to make a drawing that absolutely reproduces the appearance of the face or back of a drift, showing to any one familiar with the scheme the intensity of mineralization, amount of faulting, direction of drag, etc. The necessity of accurately reproducing the appearance of the ground is a great stimulus to careful observation. In this respect, the Anaconda system is superior to others in which mere conventional symbols, such as red dots or crosses, mark the position of ore, or ruled lines of uniform width indicate faults. The fine distinctions which are perceptible underground should appear on the geological maps with equal clarity. Unimportant details such as joint planes, etc., should be ignored.

In Butte, geological maps are based upon the co-ordinate system used by the mine engineers. Sufficient sets of convenient size are made to cover the entire camp in checkerboard manner. The maps are on tracing cloth, with a separate sheet for each level. In addition to the regular working set on a scale of 50 feet to the inch, sets on scales of 100 and 200 feet to the inch are maintained. Mine foremen are provided with 100-scale sets covering their own mines, and these sets are kept up to date by the department. It is of prime importance that the foremen should have for daily use a set of maps on which the geologic structure is shown. The expense entailed by the preparation and maintenance of these foremen's sets is more than repaid by the saving effected from their use in underground work. Cross-sections, generally on a 100 scale, are made at sufficiently close intervals to cover all details of structure. In many parts of the district, a cross-section every 100 feet is essential. It has become axiomatic among members of the department that the preparation and study of these cross-sections is one of the most fruitful methods of using the geologic data. It may be stated without reservation that even those most accustomed to using maps and thinking in terms of three dimensions seldom fail to profit by the use of cross-sections. Many an unexpected orebody has been found through their assistance. They are too seldom employed in general.

In order to transmit to the proper officers the geologic conclusions reached a system of 'recommendation sheets' is employed. These sheets are filled in by each geologist as he reaches his conclusions. They carry information under the following headings:

Mine;
Level;
Description of recommended work;
Object of recommended work;
Approximate distance to be run;
Date of recommendation;
Result (to be filled in later).

The recommendations of the subordinate geologists are submitted to Mr. Sales or Mr. Linforth for approval, and if endorsed are forwarded to the mine foremen for execution. Duplicate copies are filed for the information or action of the general superintendent. It has been found desirable to keep each foreman supplied with a large number of recommendations for each working level, in order that he may carry out the suggested work at the time most convenient to his general operations. Recommendations requiring immediate action are appropriately marked.

Conclusion.—This outline covers the routine work of the Anaconda Geological Department. I have purposely emphasized those features in which I believe it to excel—for instance, the

close co-operation with the men in the mines, the accuracy and detail of the notes taken, and the concentration of attention upon structural geology. Its excellence in these lines may be attributed both to the special problems of Butte geology which had to be faced and to the good leadership and direction which has solved them.

The problems which mines in other districts may present to the geologist are not necessarily the same as those of Butte. In other districts mining operations may depend less upon structure, and more upon such matters as the genesis of the ore and its association with certain rocks. But the correct interpretation of these points cannot be made without the preliminary work of thorough and detailed mapping of the veins, rocks, and faults. From this structural work alone can the succession of geologic events which have created and localized an orebody be worked out in its proper sequence, and upon this knowledge only can underground prospecting and development be successfully conducted.

DISCUSSION

CHAIRMAN (Mr. E. E. CAMPBELL): In my opinion, there is no branch of mining engineering more important than the study of geological associations and structures surrounding an orebody. We have all heard a great deal of the useful part played by the geologist in Butte and the part he has taken in perpetuating the life of that most important copper mining camp. A great deal of that success is due to Mr. Billingsley. He has won a very enviable reputation for himself in the working out of the most intricate faulting problems, and in Butte he is recognized as the authority on that subject. On account of the elaborate system of faulting in the Butte district, a man qualified to work out such structures is invaluable. It is a work requiring natural aptitude. In school many can grasp geometry, but a much smaller number can manage descriptive geometry. The man who has the faculty of easily grasping the principles of descriptive geometry, is a man who should develop into a structural geologist. He must be able easily to visualize the three dimensions.

Some time ago I had the opportunity of visiting the Copper Queen mine at Bisbee, Arizona, where they have a very efficient geological staff. Only a few years ago the life of the Copper Queen mine was reported to be limited. During the last four or five years, the manager informed me, the ore reserves in that mine alone have been increased to some twenty million tons. With the high-grade ore occurring in that district, this tonnage is of great importance, and all credit for this is given to the members of the geological department who worked out the probable areas for development. The geological work in this camp is along the lines of rock association rather than structure.

DR. E. T. HODGE: Mr. Billingsley has a reputation for modesty. Therefore, he did not bring out certain very important facts that should be mentioned. The history of the Geological Department at Butte is worth relating. In the early days when the geologist first entered Butte, he was a pest. As time went by he became less and less a pest. And now the Geological Department at Butte is in entire charge of all the mine exploration work and exerts a dominating influence in the direction of mining operations.

I would like to ask Mr. Billingsley this question, "In what percentage of cases wherein the geological staff at Butte predict the finding of a vein do they fail in their prediction?" Also, "In what percentage when they predict the finding of a vein is that vein filled with ore in commercial quantities?"

MR. BILLINGSLEY: In the present state of knowledge of Butte structure, it is almost invariably true that a vein is found as predicted on geological grounds. The finding of ore in the vein used to be a "fifty-fifty bet." In recent years, however, as we have come to pay more attention to the position and pitch of ore shoots within the veins, commercial ore is found, I think, in about 85% of the work recommended.

DR. HODGE: That is a very conclusive answer to that type of mining man who works on the assumption that no one

can see into the ground; and who says that predictions as to the probabilities of the ore are quite useless.

A MEMBER: Will Mr. Billingsley kindly explain the condition in respect of faults at Butte?

MR. BILLINGSLEY: In the first place, we find that in Butte the ground does not move in definite blocks as depicted in the text books. The movement is irregular, showing the ground to be more or less elastic. In the earliest fault series in Butte, the northwest faults, the movement has taken place in a direction three to five degrees from the horizontal, the southwest side moving slightly downward toward the southeast. These northwest faults are reticulated, with intervening rhombohedra of solid ground, forming a great shear zone. This peters out to the southeast. There is evidence that the thrust movement went a little beyond the point of adjustment of the stresses, with the result that there was a reaction, and a series of transverse (northeast) tensional faults were formed. These have an essentially normal displacement. The Rarus fault, next in the series, has almost a straight normal movement. It has a northeast strike, and a flat dip to the north. The earlier faults have dips of from 75 to 90 degrees; the Rarus dips 45 degrees. The next, the middle fault series, has a complicated movement best illustrated by diagrams. In brief, the hanging wall side moves first diagonally downward, then horizontally, and then diagonally upward, these movements proceeding from west to east.

DR. R. C. WALLACE: Since Mr. Billingsley mentioned the subject of relationship between the mine foreman and the geologist, I would like to ask a question on the matter of mining procedure. To whom are the recommendations of the geological staff transmitted?

MR. BILLINGSLEY: That is another question that can be answered in two ways. Years ago, after the exploration work had been placed in charge of the Geological Department, a system of recommendation sheets was inaugurated. These sheets would contain the name of the mine, the level, the nature of the work to be done, the distance to be driven, and how soon it would be desirable to start work. The geologist, on returning to the office with his underground notes, would study his maps and sections and fill out these sheets. One set he retained, one went to the mine foreman, and one to the general superintendent of the mines. It was found desirable to have from ten to twelve sheets for each level, so that the foreman would always have some recommendation that could be taken up without conflicting with his mining routine. As the relationship with the foremen improved, the work of the geologist became simplified. By accompanying the foreman on his underground trips, the geologist can see the veins and faults when first encountered, and can tell the foreman what work is to be done and which way to go without any machinery of forms and sheets. If he can make it appear that the ideas originate with the foreman, so much the better.

MR. R. W. BROCK: That geologists have not been employed more in connection with actual mining has been more the fault of the geologist than of the mining engineer. Geology started as economic geology. There would never have been the science if it had not been for its value in mining. After a good start as an applied science, these philosophers got hold of it and made it a highbrow subject. It became *infra dig.* for a geologist to do anything that would be of value to anybody. He would no more think of applying geology to practical purposes than an artist would of contracting to paint an advertisement.

I remember when I was leaving the European university where I had been studying, the old professor under whom I studied said to me, "You are going to America, you will be in great temptation; I know they will try to make use of you in mining; whatever you do, have nothing to do with economic geology." So long as geologists took that attitude mining had to depend on such geology as the mining engineer could supply in addition to his other work.

Again, when the geologist did visit a mine and was asked for his opinion on it, he very foolishly gave it. In walking through the mine he had not seen enough geology for his opinion

to be worth anything, and it was appraised by the miner at its real worth. Unfortunately, the miner was apt to consider it an opinion based on geology, instead of what it really was, one *not based* on geology, and to conclude that geology was no use around a mine. As you can see from what Mr. Billingsley has stated, the information that is so valuable, that produces such notable results, is obtained only by very close detailed study of every inch of the workings, and the careful compilation and study of the information so gained. The geologist, like the mining engineer, has to be on the job all the time. This point is recognized in the motto of the Geological Society of America, "*Mente et Malleo*," "By thought and dint of hammering." You cannot express your opinion that is worth anything at a moment's notice. If you want your opinion to be more than a possibility a great deal of close work and study has to be done. If a sufficient amount of work has been done, the opinion expresses not a possibility but a probability, and if still more work is done, it becomes a practical certainty. Most of the opinions dragged out of geologists, unfortunately, represent instances where examinations have not been made in sufficient detail to furnish conclusive evidence.

Very often prospecting work will have to be done to furnish the data that is necessary. That, of course, involves the expenditure of money. The geologist ought to refuse to express an opinion where sufficient data are not available. What the engineer wants is certain information.

Often a limited amount of prospecting would definitely settle a geological problem and enable the geologist to furnish definite information. A very good instance of the amount of work that is sometimes necessary was furnished in the scheme for the water supply of New York City. Shortly after that was started, geologists were called in They were not able by mere examination of the surface to obtain the full information that was necessary. They got permission to spend money. They spent something like four million dollars on geological investigation. That geological work far more than paid for itself in the amount of money that was saved; and furthermore, it is highly probable that the whole scheme would have been an absolute failure in the end if the information supplied by the geologists had not been obtained.

Since the whole development of a mine depends upon the geology, the money which is expended in obtaining geological information is money well spent. The greater the expenditure on this account, the more accurate the information will be, and consequently the greater the saving in prospecting and development. In coal mining stratigraphical paleontology comes in. It is as important as any other branch of economic geology. In fact, the work that has been done along this line by the U. S. Geological Survey has added more in cash value to the mineral wealth of the States than all the rest of the work they have done. I want to emphasize that, because there is a tendency for all economic geologists to go in for the science of ore deposits only. This other branch has been neglected, but it is important and offers a fine field.

MR. F. W. GRAY: I am very glad to hear Mr. Brock speak of the value of paleontology and stratigraphy in their relation to coal mining. It is something that has been neglected in Canada. It is no fault of the Geological Survey. The Geological Survey is inadequately provided with funds, and the salaries paid to the officers are not too large. In Nova Scotia we have a coalfield that was mined for a century before anthracite was discovered in the United States. No more is known of that coalfield to-day than was written by Richard Brown between 1825 and 1860.

The paper we have just heard regarding Butte shows that applied geology is what was once called an infinite capacity for taking pains. Geology repays in exact proportion to the amount of time spent. The casual employment of a geologist is good; to employ him permanently is a good deal better; and a lifelong concentration of one good geologist on one good area or one coalfield is of much greater value.

In Great Britain, Professor Kendal and Professor Lapworth put a life study into the coalfields of Great Britain and they

have discovered by indication an extension of the central middle coalfields of Great Britain. That was undoubtedly found by indications and it has added hundreds of years' supplies of coal in Britain. There is quite possibly a similar coalfield in the southeast portion of England. The Kent coalfield was discovered in the same way and I am old enough to remember the manner in which people regarded the finding of coal in the garden of England. But they are mining good coal in Kent to-day and it is due to the silent lifelong work of Professor Lapworth.

Coal mining is neglected largely because of the attitude of coal operators towards geologists. They have not been able to see that a geologist is as necessary a part of the organization of a big mining company as the manager of the mine. We could not have a better exemplification of that fact than in the information Mr. Billingsley has afforded us.

POSSIBILITIES FOR PLATINUM IN WESTERN CANADA

By W. L. Uglow

Western General Meeting, Vancouver, November, 1919.

Platinum still continues to be a very scarce metal, and has advanced far in price from that fixed during the war to the present price of about $130 per ounce troy. The latest reports do not indicate any improvement in the market condition regarding the supply of new material, although the war ended more than a year ago. Doubtless some years must elapse before the Russian situation will be sufficiently clarified to permit of production from the Ural mountains at the pre-war rate of approximately 300,000 ounces annually. The output from the United States of Colombia, the second largest producer, was increased considerably under the stimulus of high war prices from 15,000 ounces in 1913 to 32,000 ounces in 1917; but it is hardly to be expected that the decrease in the Russian output can be counterbalanced to any appreciable extent by the increment from the platinum fields of Colombia. In studying the rate of increase from the latter source it is interesting to note that as early as 1824 the output from Colombia was about 16,000 ounces, or about half that of the present time.

During the last three decades, Canada has been the world's third largest producer of platinum. As most of the Canadian supply is obtained during the refining of blister copper from the Sudbury mines, this portion of the production has not been credited to Canada, but, mainly, to the United States, where the processes of refining have been largely conducted. Native platinum has been found in various parts of western Canada, though chiefly in the Tulameen district of the Similkameen mining division, from which, during the period, 1887-1891, an average annual output of 1,500 ounces was recorded.

It is submitted, therefore, that the present conditions of the platinum market, the Canadian record of past production, and the relatively wide distribution of platinum minerals in

the West, are reasons for giving increased emphasis to the possibilities of augmenting the Canadian output of this metal.

The aim in the present paper is to correlate the data already available respecting the potential resources of platinum in western Canada, to direct attention to the types of deposits, and to stimulate prospecting and investigation of those that seem to offer the greatest possibilities in the way of actual production.

Chief Types of Deposits

Platinum is derived from two sources—lode deposits and placers. In the latter, the metal occurs as native platinum in the form of grains, points, pellets, and nuggets associated with gold, iridosmine, chromite, garnet, etc. Lode deposits contain platinum usually in the form of sperrylite, the arsenide of platinum, although in several ores of this type the mineralogical form of the metal has not yet been determined.

Under the heading of lode deposits, are several well-marked sub-divisions, each characterized by fairly definite rock types or by specific mineral associations. These are classified below:

1. *Peridotite-Dunite-Serpentine Class.* — Olivine-bearing rocks of this character are known to contain native platinum disseminated through their masses in small amounts associated usually with chromite. Important occurrences are those of Mount Solovief in the Ural mountains[1]; the Ronda mountains in southern Spain[2]; Olivine and Grasshopper mountains[3], Tulameen district, B.C.; and Tasmania[4].

2. *Pyroxenite-Gabbro-Diorite Class.*—Dike-like and sill-like bodies of basic rock, usually olivine-free, and consisting of pyroxene, hornblende, basic plagioclase, enstatite, mica, etc., occur with a wide distribution; and in various localities are found to carry valuable amounts of platinum and the related metal, palladium. Chalcopyrite and pyrrhotite commonly

[1]U.S. Geol. Surv., Bull. 193, 1901.
[2]U.S. Geol. Surv., Mineral Resources of the U.S., 1915.
[3]Geol. Surv. Can., Mem. 26, 1913.
[4]Mineral Industry, 1914, p. 609.

occur in association with these. Examples are the norite of the Sudbury district, Ontario[1]; the Great Eastern and Key West dikes of Clarke county, Nevada[2]; the Rambler dike near Laramie, Wyoming[3]; a hornblende diorite at the Walhalla copper mine, Victoria, Australia[4]; and the Black Lead, Franklin camp, Grand Forks mining division, B.C.[5]

3. *Copper Gold Vein Class.*—Quartz veins with values in copper and gold, carrying such minerals as bournonite, bornite, tetrahedrite, arsenopyrite, chalcopyrite, chalcocite, chrysocolla, malachite, or plumbojarosite, are found, in places, to contain quite appreciable amounts of platinum and palladium. Examples of this type are found at the Boss mine, Yellow Pine mining district, Nevada[6] (where pockets of ore of 1,000 to 2,000 tons were found to contain up to 4 ounces of platinum and palladium per ton); at the Roll Call Mining Co.'s property near Villa Grove, Colorado[7] (where assays from a two-foot vein showed over 5 ounces of platinum per ton); and at various localities in Europe, New South Wales, and Canada.

Over 99% of the world's supply of platinum has been derived from placer deposits, and our best information still points to this type as containing the greatest possibilities for future production. In some placer districts, notably those of the Ural mountains and of the United States of Colombia,[8] the gravels are worked primarily for their platinum content, while in others, such as the beach and river gravels of California and Oregon,[9] the metal is derived as a by-product during the process of recovering the gold. In the Ural mountain districts[10], in the

[1]Mines Br., Dept. of Mines, Can., The Nickel Industry, 1913. Royal Ontario Nickel Commission, Rep. 1917.
[2]U.S. Geol. Surv., Bull. 430, pp. 192-199. Mineral Industry, 1909.
[3]U.S. Geol. Surv., Min. Res. U.S., 1902, pp. 244-250.
[4]Mineral Industry, 1917, p. 543.
[5]West. Can. Min., Eng. and Contr., Jan. 1919.
[6]U.S. Geol. Surv., Bull. 620, 1915. U.S. Geol. Surv., Min. Res. U.S., 1915.
[7]Mineral Industry, 1917, p. 541. Salt Lake Min. Rev. Mar. 15, 1917.
[8]U.S. Geol. Surv., Min. Res. U.S., 1911, 1915. Mineral Industry, 1917.
[9]U.S. Geol. Surv., Min. Res. U.S., 1911.
[10]U.S. Geol. Surv., Bull. 193, 1901.

Ronda district of Spain,[1] in the Fifield and Platina districts of New South Wales,[2] and in the Tulameen district, B.C.,[3] the platinum is characteristically associated with chromite sand, indicating that rocks of the peridotite-dunite-serpentine type were the original source of the precious metal.

Placer deposits originate from the disintegration or decomposition of lode deposits, and the transportation and concentration of the heavier minerals, including the precious metals. Where decomposition and erosion have been great, better opportunities have been afforded for this concentration. Consequently, districts in which the topography is mature, and where the erosion has been great enough to expose deep-seated phases of the mother lode rocks, are those which should give the greatest possibilities.

PLATINUM DEPOSITS AND OCCURRENCES IN WESTERN CANADA

1. *Lode Deposits*

(a) *Peridotite-Dunite-Serpentine Class.*—In the Tulameen district, B.C.,[4] large dike-like masses of dunite considerably altered to serpentine and containing lenticular and disseminated masses of chromite are found cutting older sediments and volcanics. Associated with the chromite are small amounts of platinum. The dunite of Olivine and Grasshopper mountains has been studied many times, but few serious attempts have been made to effect a commercial extraction of the platinum content. The metal has been found to occur in amounts up to two ounces per ton, associated with the chromic- and serpentine-rich phases of the dunite.

During the summer of 1918, small shipments of chromite were made to Tacoma, Washington, from Olivine mountain, and values up to $10 per ton were reported In this connection,

[1]U.S. Geol. Surv., Mineral Resources of the U.S., 1915.
[2]U.S. Geol. Surv., Min. Res. U.S., 1911. Mineral Industry, 1917.
[3]Mineral Industry, 1914, p. 609.
[4]Mineral Industry, 1914, p. 609.

it is interesting to note the experiments that were conducted by V. N. Chorzhevsky[1] with the dunite of the Nizhne-Tagilsk district in the Urals in an attempt to get the disseminated platinum particles into the form of concentrate. These experiments strongly indicated a commercial possibility in the application of the process, but the inception of the war put a stop to any further progress in this direction for the time being. The similarity of the geology in the two districts is marked, and it is quite possible that a thorough testing of portions of the Tulameen dunite would outline areas of the rock that might be sufficiently valuable to mine, especially under market conditions that are likely to continue for some years to come.

Other districts in British Columbia characterized by the presence of basic rocks of this type are worthy of examination from this point of view. On Scottie creek,[2] a tributary of Bonaparte river, about 20 miles north of Ashcroft, outcrops of serpentine are found which carry disseminations and lenses of chromite. Diamonds have been found in this chromite, and the occurrence is sufficiently similar to that at Olivine mountain to justify some investigation into its possibilities as a platinum carrier.

In the Cariboo and Quesnel districts, platinum has been found in many localities in the gravels, but the original source of the metal has not yet been determined. The occurrences suggest chromite-bearing dunites or serpentines as the parent rocks, and search for, and investigation of, these rocks might lead to the discovery of another source of primary platinum.

(b) *Pyroxenite-Diorite-Gabbro Class.*—During the investigation, in 1918, of various platinum localities by the Canadian Munition Resources Commission and the Geological Survey, the 'Black Lead' of the Franklin camp, Grand Forks mining division, was sampled, and the following results were obtained:[3]

[1]Mineral Industry, 1916, p. 596.
[2]Can. Min. Inst., Bull. Oct. 1919, p. 1044.
[3]West. Can. Min., Eng. and Contr., Jan., 1919.

Property	No. of Samples	Oz. of Platinum metals per ton
Maple Leaf	3	0.15, 0.17, 0.38
Lucky Jack	3	0.04, 0.06, 0.08
Mountain Lion	2	0.02, 0.09
Golden Age	1	0.06
Averill group	2	0.09, 0.09
Buffalo	2	0.08, 0.19
Ottawa	1	0.06
Columbia	1	0.04

The rock carrying these values is described as a shonkinite-pyroxenite and is a marginal phase of the augite-syenite intrusion. It is dark green to black in colour, and consists largely of augite with smaller amounts of potash feldspar, hornblende, and magnetite. Accessory minerals are titanite, apatite, pyrite, chalcopyrite, and bornite.

The mineralogical form in which the platinum is found in the Franklin camp is not known, but from an analogy with other localities it is presumed to occur as sperrylite.

Sperrylite has been reported from the Copper Mountain mine near Princeton, B.C.,[1] but no further details are available.

In many respects, the Franklin occurrence resembles that at the Key West,[2] Great Eastern[2] and Rambler[3] properties mentioned above, from which platinum has been obtained on a commercial scale.

Careful attention to other basic pyroxenic rocks carrying small amounts of copper minerals might result in the discovery of other similar occurrences of platinum.

(c) *Copper-Gold Vein Class.*—The presence of platinum has been reported in quartz veins from a small number of localities in British Columbia, but only in a few of these instances have the reports been verified. It has been found in a gold-bearing

[1]Geol. Surv. Can., Summ. Rep., 1918, pt. G., p. 8.
[2]U.S. Geol. Surv., Bull. 430, pp. 192 199. Mineral Industry, 1909.
[3]U.S. Geol. Surv. Min. Res. U.S., 1902, pp. 244-250.

quartz vein on the Mother Lode claim of the Contact Consolidated Mines, Limited, Burnt Basin, B.C.[1] Samples of this ore sent to Baker & Co., Newark, N.J., gave from a trace to 0.25 ounce of platinum per ton. The quartz carries free gold, chalcopyrite, pyrite, galena, sphalerite, and molybdenite.

This occurrence and that at the Roll Call and Boss mines in Colorado and Nevada lead to the suggestion that the metal may be discovered in other mines of this general type on closer investigation.

Platinum has been recovered for a number of years by the refineries treating blister copper in the United States, Canada, and South America; and it is evident that it has been largely derived from copper sulphide ores. Although the cost of the recovery may be high, and although no allowance may be made to the smelters for platinum content, the fact remains that in the United States the annual recovery from this source has been increasing at a rapid rate, as shown by the following table:[2]

Production of Platinum Metals reported by Refineries in U.S.A.

1914	3,430	troy ounces
1915	6,495	" "
1916	24,518	" "
1917	33,009	" "

These recoveries do not in the ordinary course of events benefit the individual shipper, as the presence of the metal in such small amounts in the ore is difficult of accurate quantitative determination, and the shipments are necessarily mixed. On the other hand, the stimulating of attention to these possibilities, and careful determinations of platinum in the raw material, may result in the discovery of lode deposits sufficiently rich for separate treatment.

In spite of the fact that by far the largest portion of the world's production is derived from placers, and of the fact also

[1]Geol. Surv. Can., Summ. Rep., 1901.
[2]U.S. Geol. Surv., Min. Res. U.S., 1916-1917.

that western Canada is noted for the occurrence of platiniferous gravels, it is worthy of note that the Canadian production of the metal at the present time is very largely derived from ores of the lode type.

2. *Placer Deposits*

(a) *Tulameen Drainage System, Similkameen Mining Division, B.C.*[1]—For upwards of thirty years, placer platinum has been found in the valleys of the Similkameen and Tulameen rivers, and along Granite, Slate, and Champion creeks, tributaries of the Tulameen river. These streams have received a large part, if not all, of their loads from the dunite, serpentine and pyroxenite of Olivine and Grasshopper mountains, which contain metallic platinum associated with chromite.

No extensive operations have been undertaken in this district for the recovery of platinum. Hydraulicking and ground-sluicing on a small scale have been carried on intermittently since 1885, and these operations have been responsible for most of the platinum that has been produced. For the period prior to 1887 no details of production are available, but the following table covering the period from 1887 to 1905 is taken from Government sources:[2]

PRODUCTION OF PLATINUM (IN OUNCES TROY) FROM TULAMEEN DISTRICT

1887	2,000	1892	500	1897	266	1902	10
1888	1,500	1893	257	1898	100	1903	
1889	1,000	1894	160	1899	137	1904	20
1890	1,100	1895	633	1900	...	1905	30
1891	2,000	1896	125	1901	22		
Total							9,860 ounces

These figures account merely for the output reported through Government sources. In the earlier days of Tulameen gold-mining, much platinum must have been saved as a by-product from the recovery of gold, but as the material was then of low

[1]Mineral Industry, 1914, p. 609.
[2]Mineral Industry, 1914, p. 609.

comparative value no particular attention was paid to it, and very little record of the quantities saved was kept. The amounts of the metal recovered by the Chinese in the course of their operations and not reported, must have been considerable. Mr. C. F. Law, of Vancouver,[1] who has an intimate knowledge of the district, estimated the total output at about 20,000 ounces.

The production from the Tulameen country in the past has been the result of sporadic and intermittent mining, which was neither preceded nor accompanied by systematic exploration of the gravels. In other mining districts it has frequently been the case that such systematic investigation of mineral-bearing ground following the mining on a small scale of the richer portions has demonstrated the presence of larger areas of lower grade material which, however, had a considerable commercial value. There seems to be no reason why this should not be the case in the Tulameen district, especially since the value of platinum has increased from $3 per ounce in 1887 to $130 or $135 at the present time.

Certain sections of the gravels are no doubt exhausted; but the valleys are characterized by a series of high and low benches which have not been more than scratched. In the autumn of 1918, an attempt was made by the Munition Resources Commission[2] to drill the gravels of the valley flats for a short distance below the mouth of Slate creek, but the use of drills which were too light for the ground, and the sudden termination of the war, prevented the Commission from carrying the work to a point anywhere near completion.

This district has been the most important source of placer platinum in North America. In many respects it resembles the Ural mountains districts, but differs from them in that it has not undergone conditions of deep secular decay with the development of mature topography. Small grains of platinum may be obtained from the panning of gravels in a great many places

[1]Mineral Industry, 1914, p. 609.
[2]Can. Min. Inst., Bull. April. 1919, and Final Report Can. Mun. Res. Comm., 1920, p. 147.

along the benches and in the stream bottoms where the gradient is steep. Ground-sluicing and other operations have been conducted intermittently on a small scale between the mouths of Bear and Champion creeks during the last three or four years, and in most cases the miners have earned a livelihood therefrom. One operator has succeeded in obtaining from ten to twelve ounces of platinum per season in this way. Several prospectors in the district have in their possession small glass phials containing from a few pennyweights to upwards of an ounce of fairly clean platinum with some osmiridium, while others have remarkable collections of nuggets, which they have saved from their sluicing operations throughout a period of years.

These various kinds of evidence point strongly to the possibility of a future supply of the metal from the Tulameen country. What is needed, however, is a campaign of systematic testing and exploration which might advantageously be undertaken or encouraged by the Dominion Government.

(b) *Quesnel Mining Division, Cariboo District, B.C.*—From time to time platinum has been reported from several points in the drainage systems of this division, but no serious attempt has yet been made to make the recovery of this metal an issue of prime importance. This is a possibility for the future, and the great development of placer gravels combined with the deep weathering and prolonged erosion present conditions favourable to the occurrence of a platinum-bearing district. Results of the recent work of the Geological Survey may possibly direct attention to areas of olivine-bearing rocks which may constitute the source of the platinum metals so far found; and in this way may suggest areas meriting more careful investigation.

Something has been accomplished in the matter of investigating the platinum content of the auriferous gravels of the Quesnel district by the Consolidated Cariboo Hydraulic Mining Co., near Quesnel Forks, and the Quesnel Hydraulic Gold Mining Co., on Twenty-Mile creek.

In the Annual Report of the Minister of Mines, B.C., for 1905, Mr. J. B. Hobson, the manager of the Consolidated

Cariboo Hydraulic mine, contributed the following notes on platinum:

"For several years past, qualitative tests have been made from time to time for the presence of gold, platinum and osmiridium in the heavy concentrates that remain in the sluices after claaning up; and while making one of these tests in May, 1903, the presence of palladium was indicated in addition to platinum and osmiridium. An analysis of a sample of concentrates made by J. O'Sullivan, F.C.S., chemist, of Vancouver, in May, 1903, gave large percentages of gold, platinum, palladium and osmiridium, which brought the value of the concentrates up to $3,872.76 per ton. A second sample taken from a pan of concentrates taken from the sluices after the clean-up in September, 1904, was sent to Mr. J. O'Sullivan, chemist, of Vancouver, and gave the following results:

	Ounces per ton.	Gross Value per ton of 2,000 lb.
Gold	95	$1,900.00
Silver	180	90.00
Platinum	64	832.00
Palladium	64.4	1,769.00
Osmiridium	42	1,386.00
Copper	10.5%	16.56
Total Value		$5,993.56

"The gold and silver values are, no doubt, included in particles of pyrite and argentiferous galena, and partly in small particles of gold covered by manganese and other metallic oxides, and cannot be recovered by the process of amalgamation. The platinuum, palladium and osmiridium are found in minute metallic grains and enclosed in small fragments and nuggets of magnetite and chromite, which appear to make up quite a large percentage of the sluice concentrates found after cleaning up.

"What quantity of these high-grade concentrates are included in the deposits, or can be recovered therefrom, cannot be determined until after the completion of the system of undercurrents which is to be placed at the end of the sluice outside the tunnel, where everything of value will be separated from the tailings, before going over into the dump, and concentrated on the undercurrent tables. These undercurrents will probably be completed before the close of the ensuing season."

In the Annual Report of the Minister of Mines, B.C., for 1918, Mr. J. D. Galloway, Government resident engineer, makes the following comments upon the work of the Quesnel Hydraulic Gold Mining Co. in the matter of investigation of the platinum content of their gravels:

"It has been long known that some of the black sand (a concentrate from the placer gravels) in the Quesnel river district carried more or less platinum contents. The amount of the platinum is quite variable; many samples of black sand on assay do not show a trace, while occasionally quite high assays have been obtained. From many places up and down the Quesnel river and its tributaries, platinum has been reported, generally in small amounts. No appreciable quantities of platinum were ever recovered from placering operations

in the Quesnel division, what platinum there is apparently occurring in almost microscopic grains with the black sand, which latter has seldom been saved. The presence of this platinum can rarely be detected by the eye, and it is only through assays of samples of black sand that its occurrence has been noted. The Annual Report for 1902, page 64, gives a list of such assays, with localities. These assays were made on a number of samples of black sand, the ratio of concentration of which was quite unknown, so that they only indicate the presence of platinum in the gravels without giving any idea of the value to the yard. It may be well to point out here that the usual small quantity (amounting often to about an ounce or less) of black sand obtained from panning a full pan of gravel represents a very high degree of concentration—say 1,000 to 1. It follows, therefore, that the platinum content in the black sand concentrate must be quite high before the value per cubic yard of gravel becomes of commercial importance.

"From a number of samples of black sand taken in 1917 from the hydraulic pit at Twenty-Mile, assayers in the United States returned varying amounts of platinum. Granting that the samples were representative and the assays accurate, the platinum content of the gravels was of importance; in addition, the assays showed a considerable gold content in the samples."

These reports indicate that by means of careful work certain amounts of platinum may eventually be obtained from the Quesnel district as a by-product in the recovery of placer gold.

(c) *Liard Mining Division, B.C.*—Mr. Hamfield, of the Berry Creek Mining Co., Limited, reported as follows regarding platiniferous black sand in the placers of Thibert creek:[1]

"Experiments were made to concentrate the black sands containing the minerals of the platinum group. For this purpose an undercurrent, and a series of tables covered with cocoa-matting, canvas and burlap, were installed at the end of No. 2 sluice.

"Although it was, this year, largely experimental, the mechanical concentration was quite satisfactory. During the twenty-one odd days that the concentrating plant was in operacion, it yielded 250 pounds of concentrates, and this amount could have been greatly increased by a man in attendance with some experience in concentration.

"Assays of these concentrates, made at the British Columbia Government Assay Office, gave 60 ounces of platinum to the ton of concentrates, and assays obtained in San Francisco gave up to 15 ounces of platinum and 7 ounces of gold per ton. These results were obtained almost entirely from top gravel, and as the bottom gravels will naturally contain more of the heavier minerals, the concentrates from the latter should be very much better than this year's output."

(d) *Cariboo Mining Division, Northeastern District, B.C.*—On Government creek, a tributary of Hixon creek, which latter joins the Fraser about 35 miles south of Fort George, a number of placer leases have been recently tested by Mr. Dougald Cameron,

[1]Ann. Rep., Minister of Mines, B.C., 1905.

and the presence of platinum with the gold has been demonstrated. The following notes of this occurrence are taken from the Annual Reprt of the Minister of Mines, B.C., for 1918:

"The placer ground held on Government creek by Dougald Cameron and partners, which was described in last year's report, was re-examined during the summer. It had been reported that the black sand on this creek carried an appreciable platinum content, and to determine this point some sampling of the ground was done.

"Samples were taken from different points in the gravels along two miles of the stream. At one place, gravel from bedrock was put through a rocker and the black sand concentrate saved. The other samples consisted of the black sand recovered from the panning of two, three or four pans of gravel.

"The black sand from each sample was weighed and assayed for gold and platinum. From these results and knowing the average weight of a pan of gravel, the value in gold and platinum in a cubic yard of the gravel has been calculated. The following table shows the results:

Yardage in Sample	Value in Gold per Yard	Value in Platinum per Yard	Total Value per Yard
12 cu. ft.	$ 2.94	$ 0.18	$ 3.12
1/30	.34	.06	.40
1/30	.10	Trace	.10
1/60	Trace	Trace	Trace
1/20	.52	.01	.53
1/30	.02	.005	.025
1/40	.29	.006	.296
1/30	.05	Trace	.05
1/50	Trace	Trace	Trace
1/20	.79	.03	.82
1/60	.005	Trace	.005

"The average of the eleven samples is: gold, 55 cents per yard; platinum, 3 cents a yard; or a total of 58 cents a yard. From this, it will be seen that the platinum values are unimportant, but that the gold content, while variable, averages up well. These concentrates from the panning of the gravels contained most of the gold that occurred in the quantity of gravel panned. Some of it was visible to the eye, but all was fine and probably a portion of it was flour gold."

(e) *Yukon.*—The following notes on the occurrence of platinum in the Yukon were furnished by W. E. Cockfield:[1]

"Platinum has been reported to occur in samples of gold dust from near Dawson, and from the Yukon generally, in amounts which have a commercial significance. In one case. 390 milligrams of osmiridium per ounce of gold was obtained. This platinum appears to be directly combined with the gold, and for that reason has doubtless been overlooked in the ordinary process of melting and refining. Taking these figures as the basis of calculation, the total output of gold dust from the Yukon would contain at least 2,500 ounces of platinum metals per annum.

"The platinum occurring in the auriferous gravels on Burwash creek is small in amount. Here, it is visible as a separate metal in the gold dust, amounting to about 150 milligrams per ounce of gold. This can be separated from the gold dust with care, but the production from this locality cannot exceed four or five ounces per season, under present conditions of mining.

[1]Geol. Surv. Can., Summ. Rep., 1918, part. G.

"Platinum has been reported also from a number of other sources, such as Ferguson creek, a tributary to O'Connor (Kaskawulsh) river, Teslin (Hootalinkwa) river, Scroggie creek, and Stewart river, but no figure as to the content of the gravels can be given.

"In conclusion, it appears that the platinum metals are widely distributed in the Yukon, but as yet there are no deposits known which could prove profitable to exploit for these metals alone. Where gold is present in sufficient quantities to pay for mining, platinum and the allied metals may prove to be valuable by-products at some localities, and it is worth while testing occasional samples of gold dust from the territory in order to see that none of these constituents is being overlooked."

(f) *Peace River, B.C.*[1]—Platinum has been found in small amount in the bars and benches of the upper Peace river, chiefly between Hudson's Hope and the Ne Parle Pas rapids. It occurs in the form of fine grains associated with fine gold. As yet, very little systematic work has been undertaken for the purpose of testing these gravels, but very small amounts of the metal have been obtained during the past two seasons as a result of the operations of two or three individual placer miners. It is not expected that any appreciable addition to the British Columbian output will be made from this source, as the platinum particles are small and they must have been transported a long distance from their parent source.

(g) *Ingenica Creek, Omineca Mining Division, B.C.*—Some attention has recently been given to the placer gravels on Ingenica creek, a tributary of the Finlay river. It was the intention of an Edmonton syndicate to drill bench leases during the summer of 1918, but, owing to low water, it was impracticable to get the drill in to the area desired.

Black sand from these leases carries small amounts of platinum, but nothing has yet been accomplished in the way of systematic testing or recovery of the metals.

(h) *North Saskatchewan River, Alberta.*—Platinum in small amounts has been recovered from the bars and benches of the North Saskatchewan river, in the vicinity of Edmonton. No attempt of any importance has yet been made to mine this

[1]Final Report Can. Mun. Res. Comm., 1920, p. 156.

platinum. Certain portions of the gravels were recently tested, however, by the Canadian Munition Resources Commission,[1] whose results indicated that though platinum was present in the form of very small particles associated with fine gold, its amount with respect to the gold was only about four or five per cent. It is possible that certain sections of these gravels might contain enough platinum in association with the gold to warrant mining on a small scale.

SUMMARY AND CONCLUSIONS

From the evidence here adduced it is clear that there are possibilities of a future supply of platinum from Western Canada. Three lines of attack are needed to develop these possibilities: First, attention must be directed to those localities the geology of which is similar to that in producing districts; second, the ground should be accurately tested in accordance with approved methods; and third, progress must be made in the matter of facilitating the recovery of platinum from lode ores and auriferous gravels. In this connection, attention is directed to methods of recovering platinum by amalgamation, described in the *Mining and Scientific Press*, October 12th and December 21st, 1918.

A platinum determination is a difficult one for an assayer to make with his regular equipment. Samples of material in which the presence of platinum is suspected should be sent to a reliable chemist who is accustomed to such work. Many examples have come to the writer's notice of ores which have been reported as platiniferous by local assayers, but which proved barren after the material was analysed by such companies as A. R. Ledoux & Co., New York, and Baker & Co., Newark, N.J.

Prospectors and mining companies should pay particular attention to the following types of rocks and ore-bodies as possible carriers of platinum:

1. Masses of dark green, greenish gray or black dunite, peridotite, or serpentine, especially where they carry chromite.

[1]Can. Min. Inst., Bull. April, 1919, and Final Report Can. Mun. Res. Comm., 1920, p. 185.

2. Masses of pyroxenic or hornblendic rocks of green to black colour, especially where they are accompanied by chalcopyrite, pyrrhotite, pyrite, etc.

3. Gold-copper-quartz veins carrying chalcopyrite, and complex antimonial and arsenical compounds of copper and other metals, such as bornite, tetrahedrite, bournonite, plumbojarosite, enargite, etc.

4. Placer deposits, which contain olivine or chromite associated with their black sand, or which are located in areas whose drainage systems cross masses of intrusive rocks, such as dunite, peridotite or serpentine.

DISCUSSION

MR. CHARLES CAMSELL: In this paper Mr. Uglow has given us a very clear exposition of the platinum possibilities of British Columbia. It seems to me that, with our platinum resources, particularly those of British Columbia, we ought to be able to produce sufficient for our own needs; but we do not. The production will be this year, I believe, about ten ounces only. I am more familiar with the Tulameen area than any other part of British Columbia. Mr. Uglow has explained how the platinum occurs there. Although it has been known for many years that platinum occurs in the rocks of this district, extraction has not, up to the present, been a success, and the reason is that the platinum was so sporadically distributed through the rock that it cannot be extracted profitably. There is no question about its presence in the peridotite, because I have a sample about an inch square, showing its occurrence in the form of a narrow vein.

Of all of the known sources of platinum in British Columbia, I am inclined to think that the Quesnel river country is the most promising for production on a large scale. I have arrived at this conclusion largely on account of the amount of gravel in the Quesnel river as compared with what we know exists in the Tulameen. Mr. Uglow has referred to the experiments by Mr. S. J. Marsh on the recovery of platinum from black sands, and I hope that those experiments will be successful.

MR. E. A. HAGGEN: It is a matter of great interest to know that within the past two years Mr. S. J. Marsh, who is a pioneer engineer in this country, has been giving considerable attention to the extraction of platinum from the placers. Mr. Marsh's work has been checked by the United States metallurgists, who express great satisfaction with his results, so much so that a large dredging company proposes to make a thorough exploration of platinum-bearing deposits in British Columbia during the coming season.

Recently in an investigation of the Leach river gravels I encountered some platinum. One sample of the gravels showed 85c. to the yard. Engineers associated with Mr. Marsh who examined those samples told me that there was more platinum there than we had obtained. Those gentlemen believe that the time is not far distant when British Columbia will become one of the world's important platinum producers, and we all hope that that may be so. The occurrences of platinum are widespread over this country.

Another metal that should be looked for in this province is palladium. Ten years ago my attention was drawn to the fact that certain ores from Prince of Wales Island contained a good deal of palladium.

MINERAL OCCURRENCES IN THE STEWART DISTRICT

BY E. E. CAMPBELL

Western General Meeting, Vancouver, November, 1919.

During the past year no part of British Columbia has occasioned more interest, from a prospective mineral producing standpoint, than that tributary to Stewart. About ten years ago Stewart was the centre of a noted mining boom, and, although at that time several properties of decided merit were found, the district was more or less discredited by the foolish exploitation of a few properties of questionable value.

The interest now being taken in mining development in the Stewart district is based on many valuable discoveries of high-grade ore, and the area in which these discoveries have been made is so extensive a one that the district as a whole cannot fail to become one of first importance. Only that portion of the Stewart mineral area situated on the Salmon river will be discussed in this paper, as it is in this part that many of the recent finds of rich ore have been effected.

It has always been the opinion of geologists and engineers that the successful future of the mining industry in northern British Columbia must depend upon the development of primary ores of comparatively low metallic content. Due to heavy glaciation most of the oxodized portions of the outcrops have been removed; the same can be said of the surface enriched zones, as few orebodies on the north coast show, on development, definite evidences of surface enrichment. The recent discovery of extensive orebodies carrying minerals that are unquestionably of secondary origin, establishes a reversal of the conditions previously thought to exist, as well as a change in the opinion of mining engineers respecting the types of ore to be encountered in the district.

Owing to the absence of rich surface ores, prospecting for minerals in northern latitudes has never been carried on as enthusiastically as in the districts farther south, such as California, Arizona and Mexico. In these southern areas no glacial erosion occurred at the time the northern portion of the hemisphere was enveloped in ice, and, consequently, these districts were left free to the processes of oxidation and subsequent enrichment. This produced many rich surface ores which were eagerly sought for by the prospector, and, which when found, could easily be turned to his credit. The northern prospector generally had low-grade ore to deal with, the development of which usually required much capital.

The discovery of rich surface ores in the Stewart district has so stimulated prospecting that the final results to the prospector, mine operator, and the province should be manifold, and such discoveries should be an incentive to the further examination of other districts and the consequent uncovering of new orebodies. A large part of the Coast range of British Columbia and the adjacent portion of south-eastern Alaska is composed of granitoid rocks. These vast areas of granite enclose, or are adjacent to, many scattered areas of sedimentary, and numerous types of igneous rocks, and it is these areas flanking the granite that afford the greatest promise of reward to systematic prospecting. The Stewart mineral deposits occur in one of these areas adjacent to the granite.

For the first five miles up the Salmon river from where it empties into the Portland canal, the country rock consists of granite. In contact with this is highly altered schist of apparently sedimentary origin. Beyond this is a vast area underlain by greenstone and tuffs, and it is in these rocks that the recent ore discoveries have been made. The greenstone, which is essentially a quartz porphyry, has, in places, been sheared and silicified and it is in such zones that the ore occurs. In places the greenstone is overlain by a volcanic breccia, varying in texture from fine to coarse grained. This rock is generally grey to green in colour, and shows the same alterations as those found in the quartz porphyry. Where it is fine grained and changed to the colour of the greenstone, its accurate classification in the field is almost impossible.

The greenstones, or altered quartz porphyry, containing the ore, are part of a series of rocks called by Mr. R. G. McConnell the Bear River formation. These greenstones vary widely in appearance, ranging from light green and almost normal quartz porphyry, to a dark green and highly altered product.

Mineralization is confined largely to the greenstones and is only present to a minor degree in the breccia. On the Province claim of the Big Missouri group, an enormous development of the primary ore replacing the greenstone is exposed. On the Yellowstone group the ore occurs entirely within the breccia.

The Province outcrop covers an area of approximately 1,500 feet by 1,000 feet. The Yellowstone outcrop does not show as large an area of mineralization because the breccia is not shattered or silicified to the same extent as the greenstone, and mineralization is confined to sheer zones and fissures, but with much higher values. It is possible that the thickness of the breccia is not great and that more extensive mineralization, probably with high-grade ore, is present in the underlying greenstone.

The primary ore in the district consists of the following sulphides disseminated in a schist gangue: sphalerite, galena, chalcopyrite, and pyrite, carrying gold and silver. The following is a typical analysis of the large outcrops of disseminated low-grade ore in the district; the figures are the average of a large number of samples:

Zn	Pb	Cu	Au	Au	Insol	SiO_2	Fe	CaO
4.0	1.2	0.4	0.04	1.7	66.1	59.1	9.8	1.4
			S	Al_2O_3	MgO			
			11.1	5.8	2.0			

A number of smaller outcrops of primary ore, such as on the Hercules, Indian, Yellowstone, and some outcrops on the Missouri group, are of much higher grade and show values as follows:

Zn	Pb	Au	Ag
6.3	5.6	0.1	7.65

The structural features in respect of these low-grade ore-bodies in the greenstone are, first, marked schistosity, the direction of which governs the strike of the ore zones; and second, intense shattering of the greenstone, with the development of an aggregation of reticulating quartz veins and stringers. The metallic sulphides are disseminated through the silicified greenstones, as well as through the quartz veins and stringers.

The high-grade orebodies, which are undoubtedly of secondary origin, on the Premier, Little Joker, E. Pluribus, and Forty-Nine, occur altogether within the greenstone.

The general structure of the mineral zones carrying the high-grade ore is similar to that of the zones carrying the low-grade. An important feature, and one which has a bearing on the high-grade ore-shoots, is shown in a series of fissures crossing the line of schistosity of the mineral-bearing schists. This cross-fissuring is most pronounced in the Premier mine, and it is along these fissure zones that some of the exceedingly rich ore is found. The extent of these rich shoots has not been determined, but sufficient development has been done to prove that the mine is one of great richness. A sample of one hundred sacks of ore taken from this mine showed the following values:

Zn	Pb	Cu	Au	Ag	Insol	SiO_2	Fe	CaO
3.3	1.7	0.56	6.18	148.6	75.1	64.9	7.6	1.3

S	Al_2O_3	MgO
8.6	6.5	1.7

Although later shipments gave higher values, this analysis affords some conception of the richness of the ore encountered in the mine. A depth of 200 feet below the surface has been attained in the zone of secondary enrichment, with no evidence of decreasing values.

The same general structure prevails on the other properties having rich secondary ores.

The minerals present in these enriched orebodies are native silver, argentite, ruby silver, sphalerite, galena, and pyrite

carrying high gold values, and occasional small amounts of chalcopyrite. The existence of the enriched surface ores has not, at present, been explained. These mineral zones were doubtless overlain by a series of volcanic breccias, and it may be that these overlying and protecting rocks were the means of preserving the secondary ores from destruction during the period of glaciation.

The nateral facilities to the conduct of mining in the district are distinctly favourable. The mineral-bearing area is from twelve to twenty miles from tidewater, and the intervening country presents no great difficulties in railway construction. Up to an elevation of 2,000 feet the hills are densely wooded with a good quality of spruce, hemlock, and cedar. Many thousand horse-power, continuous throughout the year, could be developed in Cascade creek, which flows across the mineral area.

This district should prove most attractive to the mining engineer. The ore is complex and will require skill in its treatment. The rich ore can be smelted direct, but the problem of the treatment of the low-grade material offers a broad field for the exercise of metallurgical skill and ingenuity.

DISCUSSION

THE CHAIRMAN (MR. CHARLES CAMSELL): During the past year no mining area in North America has, I believe, attracted so much attention as the Stewart district.

PROF. TURNBULL: In discussing a previous paper I may have given the impression that I considered the silver ores of the Portland Canal section to be primary in nature. This would be erroneous, as there appears to be little doubt now that the exceptional values found are due to secondary enrichment, so that I shall not take issue with Mr. Campbell on that point. On the other hand, the impression that all the workable values are due to secondary enrichment, would not be a fair one to the district. There are some facts which go to indicate that the mineralization as now found, is in part at least primary, and that there is reasonable hope of finding workable primary orebodies, which would of course add greatly to the life and importance of the district. To give all the arguments on both sides would take too long at present, but I think that we can await the results of further development work with some confidence.

THE CHAIRMAN: That was the impression I obtained regarding these ores, although I have never examined the district. The ores now being developed are exceedingly rich, but the future of this camp will probably depend on the low-grade primary ores. Even so it seems to me, the future of the district is practically assured.

MR. P. W. RACEY: I do not wish to enter into a discussion as to the primary or secondary nature of the ores in the Stewart and Alice Arm districts. There is, however, one point that I would like to mention in addition to the information which has been given by Mr. Campbell. The majority of the ore deposits in the Salmon River district are more or less lenticular bodies whose strike is north-easterly to easterly. Towards the northern portion of the camp and about a mile and a half north of the E. Pluribus claim, are ores which I believe are formed in a different way to those usually found in the camp. In this particular portion of the camp, a great mass of moderately fresh dikes have been intruded through the schists and older rocks, and now strike in an easterly direction. These dikes originate from the batholith of granodiorite which lies to the west. An older dike of much crushed porphyry, strikes nearly north and south, and was seen to have been considerably faulted by the fresher dikes. This older crushed dike had its fragments recemented with secondary quartz containing a considerable amount of pyrite and smaller amounts of galena and zinc blende. On the surface no sign of the sulphides could be seen, except that the rock was rusty. It had a light brown colour, and showed a considerable percentage of the cementing quatrz. Practically no values were observed on the immediate surface, but on breaking into the dike to a distance of six inches to two feet, excellent values in silver were found to be associated with the sulphides.

Not very much work has been done on the ore of this kind, but at least enough has been done to show that these older dikes may be a valuable source of ore. They have been passed over by many of the prospectors, who apparently were of the opinion that they were worthless because on the surface nothing could be seen to suggest the silver content, and for the reason that the dikes did not have a very attractive appearance outside of the actual portions where quartz could be seen. The particular dike to which I have reference, varied in width from fifteen to fifty feet, and if further development shows that ore of commercial value can be found for the full width, or even for a portion of the width, it would prove of importance. Samples taken by myself showed the fractured material of the dike to contain very good values, and I think that the difference in this ore to that found in other portions of the camp is worthy of mention.

THE CHAIRMAN: These particulars confirm the opinions expressed years ago by geologists and mining engineers regarding the association of what is known as the Coast Range batholith with ore deposits. If you look at the geological map of British Columbia, you will see that the Coast Range batholith is cut by a great number of very deep fiords; but Portland Canal and Observatory Inlet are the only two that cut right through the batholith to its eastern side, and consequently at the head of these two inlets the most favorable condition exist for the occurrence of ore deposits. Most of the other fiords to the south only go a certain distance into the batholith and do not go to its eastern border. It is this eastern border that prospectors have always been advised to prospect for metallic deposits. The contact is pretty well defined and is well exposed. The western contact is submerged beneath the sea so that it cannot be prospected. However, prospecting in the batholith is not as hopeless as it might appear to be. The Coast Range batholith as shown on the geological maps of Canada give a wrong impression of the extent of the batholith. It is marked as a huge mass of granite rock, 100 miles wide and one thousand miles long. As a result of investigations carried on in the last few years, it is being shown more and more that it is not a single huge mass of granodiorite, but that there are remnants of the old stratified roof of the batholith still lying in the batholith itself, which have not been eroded away, and all these bands of stratified rocks are more or less mineralized. So that while the batholith has been avoided by prospectors, possibly because it has been supposed to be nothing but granodiorite, yet, at the same time, it is well worth while prospecting on account of the remnants of the stratified rocks in it. Moreover, it is a country well worth prospecting, because the transportation difficulties are solved by the existence of these deep water fiords, so that ore deposits in it are comparatively easily developed.

MR. S. S. FOWLER: I would like to ask in respect of the western limitations of the batholith, whether the Surf Inlet mine is near the edge of the batholith or not.

THE CHAIRMAN: It is in the batholith.

MR. FOWLER: How near the edge of the batholith is it?

MR. E. E. CAMPBELL: It is quite close to it because there is limestone not very far from Surf Inlet, on Princess Royal Island.

THE CHAIRMAN: Some years ago it used to be considered when prospecting, that when one entered a granite area you could look for the sign that was hung at the entrance to Dantes' Inferno, "All hope abandon ye who enter here." Ore bodies, however, are being found in the granite, and there is no reason why they should not be found in the granite.

MR. FOWLER: The question was whether the batholith was known to extend to the west of the limestone that goes in a north and south direction in Surf Inlet and is a shear zone in the granite. West of that shear zone is limestone, and I want to know whether they have ever found granite west of that limestone.

MR. DOLMAGE: You will find similar rocks on the west coast of Graham Island.

MR. E. E. CAMPBELL: My experience is that granite is found intruding through rocks in many different parts of the West Coast, and is not necessarily connected with the main batholith on the surface. In the areas surrounding Anyox can be found many islands of granite intruding the sedimentary and other rocks. Granite occurs on Graham Island which is a long distance west of Surf Inlet. I have also run across it on Prince of Wales Island on the Alaska Coast, where it can be found extending into the ocean. This granite probably underlies all of the sedimentary rocks on the coast and breaks through to the surface at the places where it has been found.

Regarding the age of the granite, I would not care to go into a geological discussion correlating the different ages of granite occurrences. I think most of the ore that we get on the coast is contained in rocks older than the Coast Range batholith. It may be that the granite, or some of its offshoots, was the agent that mineralized these rocks. There is no doubt in my mind that all of the rocks at the head of the Naas River and the rivers flowing into Alice Arm, Hastings Arm and into Portland Canal, are all older than the granite.

DR. DOLMAGE: I have taken a great deal of interest in the problem of secondary enrichment. I may also say I have examined specimens of ore from a great many parts of British Columbia but I have found very little secondary ore. I might also say from the description of these deposits, no evidence has been adduced that would prove to me that the ore is secondary. The fact that some of the ore is very rich is not a proof that it is secondary. Some primary ores are very rich indeed. The fact that the ore is cut by veins that carry the richer minerals does not prove those richer minerals are secondary. It proves that they are later in time of deposition than the other minerals. I am sorry to say I have not had the opportunity of examining any of the ore from the Dolly Vardon mine, and I am not saying it is not secondary. But I do say that I have received no conclusive evidence from the description I have heard that it is secondary. The strongest point in favour of it being secondary ore is the presence of ruby silver. That was the only mineral mentioned that serves for argument. Ruby silver is nearly always secondary. The minerals that we once thought secondary are dwindling rapidly. It is not so long ago that the geologists of that day thought that bornite was a secondary mineral, and a good many mines were turned down on that account. In my study of the Marble Bay mine I found bornite at a depth of 1,700 feet. There is no doubt in the minds of geologists of to-day that bornite is primary.

MR. E. E. CAMPBELL: That statement of mine was just an opinion that such a possibility might exist. You will grant that it is possible that the quartz porphyry which is sheared and mineralized could later be oxidized and surface enriched before the period of vulcanism. Later on when the volcanic rocks were deposited on top of these zones, they protected them from surface erosion and from being scored away by the ice. I have seen several cases of such protection, one of which was very striking and lead to the development of one of the richest copper mines in the world. This property is the United Verde Extension mine at Jerome, Arizona. An adjoining property called the United Verde was developed chiefly on primary ore which extended to the present surface. Geological investigation showed that large masses of sedimentary rocks and igneous flows had been denuded from this surface. Between this property and the United Verde Extension a fault occurred which had a down throw of 2,000 feet. In the final levelling of the country, the United Verde orebody was exposed while the ore occurring in the faulted zone still carried an overburden of later rocks to a depth of about 800 feet. A shaft was sunk through these surface rocks exposing the greatest body of chalcocite ever found in Arizona.

MR. FOWLER: Mr. Campbell's remarks cause me to give you an illustration of the importance of these primary and secondary ores. As to how far down a secondary mineral may be found is an open question, but I would like to illustrate the fact that they proceed to depths which seem quite unexplainable by telling you that, at the Bluebell mine at Kootenay lake, we have found considerable amounts of native copper at a depth of about 250 feet below the surface of Kootenay lake. At about 400 feet below the level of the lake, we penetrated the orebody,

and in one cavity I found a magnificent display of the most beautiful crystals of selenite. However, I have never had the good fortune to find any primary orebodies. The bottom of Kootenay lake is 440 feet below its present surface, and these minerals were deposited before Kootenay lake existed. In other words, the water level was 500 feet lower than it was now. That is a point, I think, of great interest to us all.

DR. WALLACE: In the pre-Cambrian in Northern Canada we do not expect to find secondary enrichments. The feldspars half an inch below the surface are often absolutely unweathered. Recently I found chalcocite in some of the copper sulphide bodies that occur in the pre-Cambrian in northern Manitoba. I do not know just the state of our knowledge concerning chalcocite as a secondary mineral. It would take a great deal to convince me that chalcocite in the above case had anything to do with secondary enrichment. In fact I am convinced it had not.

MR. R. R. HEDLEY: I gather from Dr. Dolmage's remarks that he was skeptical about the existence of secondary ores in British Columbia. Perhaps that is because he has got gone far enough afield. I wish he could convince me that the ore found in Stump Lake district was not secondary. With regards to chalcocite in Aspen Grove, perhaps Mr. Fleet Robertson might express an opinion.

MR. FLEET ROBERTSON: I have got into all sorts of trouble by not being able to find copper of the proper percentage in certain areas. As far as I can see the chalcocite at Aspen Grove was in small particles in the rock, but I could not see any evidence of secondary enrichment. Fissures were present, but they were small and did not bear any relation to the general mineralisation of the district. So that I think the evidence as far as Aspen Grove is concerned is that the chalcocite is not secondary.

SILVER ISLET AND VICINITY, THUNDER BAY DISTRICT, ONTARIO

By T. L. Tanton

Annual General Meeting, Toronto, March, 1920.

Mining claims were first taken up on the north shore of Lake Superior in 1846. Twenty-seven blocks, measuring approximately 5 miles by 2 miles, were surveyed in that year. One of these blocks, No. 8, called Joseph Wood's location, enclosed a tract lying east of Thunder cape and at the southern end of the great peninsular which separates Black bay from Thunder bay. The trend of the longer axis of Wood's location parallels the general course of the shore, andthis is almost the same as the trend of a number of diabase dikes and most of the associated veins which occur in this locality. For twenty-two years this mining location was allowed to lie almost entirely neglected,[1] but in 1868, after silver had been found near Port Arthur and farther west, the Montreal Mining Company determined to make a systematic exploration of this and of their other claims. They were probably influenced in taking this step by the Ontario Government's action of imposing a tax of 2 cents per acre on Lake Superior mining claims. Mr. Thomas Macfarlane took charge of the company's exploration party, and on the 23rd of June reached Wood's location. He has written a very interesting account of his progress and observations leading up to the discovery of Silver Islet.[2] Macfarlane noted that Sir W. E. Logan had called attention to the faulted contact between the upper and lower copper-bearing series which occurs on Wood's location,[3] and intimated that it was to be expected the many dikes and sills near Thunder cape would present interesting relations to the sedimentary rocks. Macfarlane adds, "I therefore determined to make a complete geological map of Wood's location." An accurate survey of the shore line was undertaken, and while planting pickets on the numerous islands fronting the location, an assistant, Mr. Gerald C. Brown, landing on the rock, which

[1]Trans. A.I.M.E,, Vol. 8, p. 227.
[2]Idem, p. 228.
[3]Logan, W. E., Report to Commission of Crown Lands, 1846.

was named shortly afterwards by Macfarlane "Silver Islet," observed the vein and the galena in it. Macfarlane then visited the island to collect samples, taking with him three men. One of these, John Morgan, found the first piece of native silver in the vein, close to the water. One blast was sufficient to detach all the vein rock carrying ore above the surface of the water. Examination of the submerged vein a short distance out showed the presence of rich silver ore with galena.

The history of the development of Silver Islet mine is one of the most interesting in Canadian mining literature; and is recorded in a valuable document prepared by Mr. E. D. Ingall

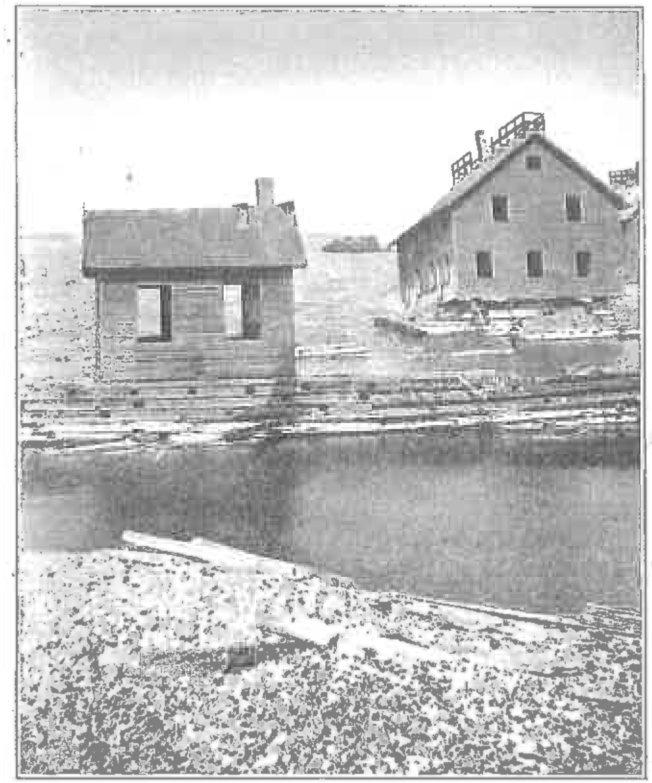


Fig. 1.—Silver Islet as it appeared in 1919.

in 1887, shortly after the mine closed.[1] The Islet originally measured about 80 feet each way and rose at its highest point 10 feet above Lake Superior, being in an exposed position three-quarters of a mile from the shore of the mainland. In order to keep the waters of the lake from washing over the Islet during storms, breakwaters and cribbing were built and, although these were very solidly constructed, so great was the stress to which they were subjected by storms and ice pressure that they required to be replaced in whole or in part on several occasions.

Working under considerable difficulties, and following the vein downward through varying widths and long lean stretches, a depth of 1,230 feet was attained in the mine. Because, however, of lack of fuel mining operations ceased early in the spring of 1884, the mine flooded, and since no indications of rich mineral deposits had been found in the vicinity on the mainland, the property was abandoned. The total value of the silver mined at Silver Islet was $3,250,000.[2]

Logan's report indicated to Macfarlane that the faulted contact between the Animikie and Keweenawan formations (to give them their present names) should be carefully examined. At the very inception of this work, while making their base map, the exploratory party found the mineralized vein along the extension of this fault offshore from the contact they set out to examine. This casual discovery, on account of its richness apparently, became the sole object of attention; the determination to examine the faulted contact on the mainland was lost sight of and all subsequent work on the mainland was directed with the sole aim of locating the extension of the particular vein which occurred on Silver Islet.

At the present time, only the partly ruined mine buildings and the dump can be seen on the Islet, which now measures 140 yards by 70 yards. (Fig. 1). Meanwhile the recent advance in the price of silver has aroused interest in the many abandoned silver properties of the district, and recently Silver Islet was acquired by American capitalists.

[1]Geol. Surv., Can., Ann. Report, 1887, pt. H.
[2]Geol. Surv., Can., Ann. Report, 1887, 37 H.

In 1919 a Canadian Geological Survey party examined the shore of Lake Superior from Port Arthur to Nipigon, and while in the vicinity of Silver Islet collected information that may be helpful in deciding whether other orebodies like that of Silver Islet may be expected to occur in that vicinity, and if so, where search for them may best be made.

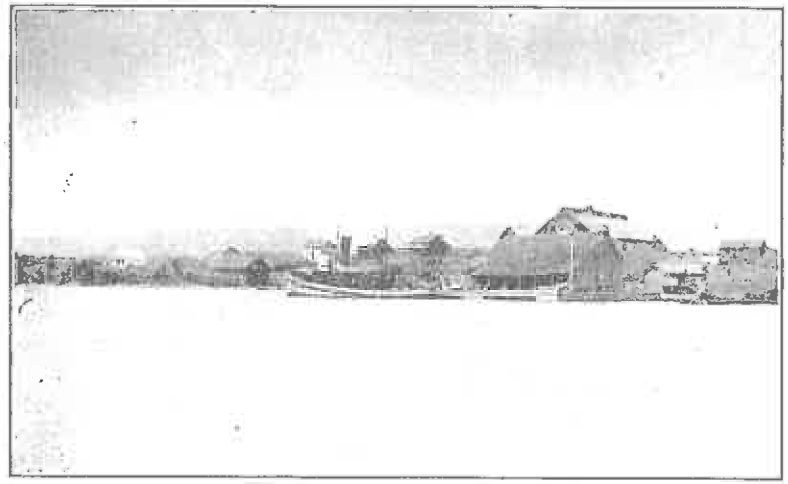


Fig. 2.—Silver Islet Landing as seen from Burnt island. The great diabase-capped mesa near Thunder Cape, three miles away, can be seen in the background.

The orebody at Silver Islet occurred in a divided vein 20 feet in width, trending north 35° west and dipping about 80° toward the northeast. The vein occupied a fault which cut across a diabase dike, its differential product (graphite-bearing aplite), and Animikie shales and greywackes. The dike was 300 feet thick and had an apparent trend of north 59° east and dipped 75° toward the southeast. The Animikie sediments are nearly horizontal with a dip of 5° toward the east-southeast. The fault showed a relative displacement on either side of the vein, the northeast side having moved 80 feet toward the northwest, and horizontal striations could be seen in the slickensided country rock. No other data on the dislocation are available. The original Islet was 80 feet in diameter and had a maximum elevation of 10 feet above Lake Superior. It represented a small projection on the edge of the diabase dike together with the soft graphitic differentiate and the vein which it enclosed.

During the progress of mining the vein proved to vary considerably in width, from mere stringers up to 20 feet. Silver was found in irregularly spaced pockets, the greatest concentrations occurring above the 400-foot level in two 'bonanzas,' which in shape resembled a thin section of an inverted pear.

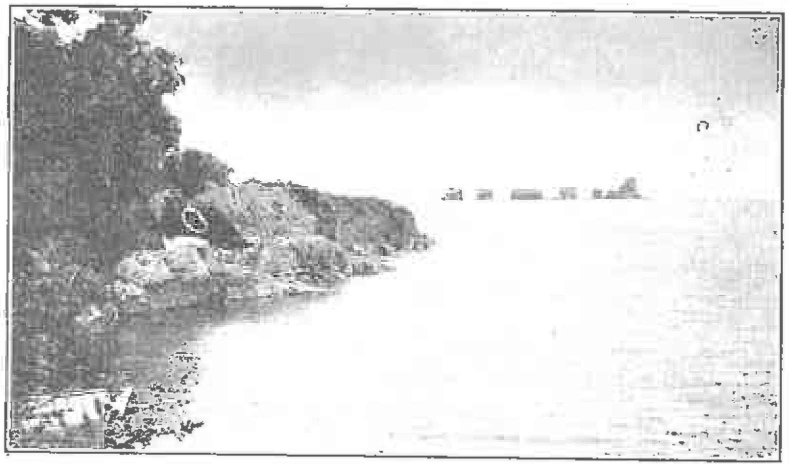


Fig. 3.—General view of Silver Islet from Burnt island, half a mile distant. The extension of the Silver Islet vein is exposed on the spur of Burnt island that shows in the photograph.

The vein was followed along its trend by tunnels at the 560-foot level (i.e., below the main silver bodies as found on the Islet). These tunnels extended nearly 700 feet each way from the Islet, and traversed Animikie sediments within which the vein had not well defined walls; it also cut four diabase dikes which were parallel to the main dike on the northwest and also to one small inclined dike toward the southeast. No ore was encountered along these extensions.

The extension of Silver Islet vein for a mile on the mainland and across Burnt island (Fig. 3) was carefully examined at a

number of points and three shafts, with depths of 40 feet, 100 feet, and 60 feet, were sunk on it. In the farthest shaft, about three-quarters of a mile inland, galena yielding 19 oz. of silver to the ton was found at a junction of the vein with another paralleling a dike; this was known as Morgan's junction. Small parallel veins of quartz and calcite occur within or at the margins of nearly all of the numerous dikes crossed by the vein, but Morgan's junction is the only one on which a pit was sunk.

The composition of the vein varied considerably both laterally and vertically, but the general mineral association permitted of the landward extension being recognized as similar to that on the Islet. The gangue minerals were calcite, dolomite, quartz, fluorite and rhodochrosite. The metalliferous minerals were silver; the arsenides, domeykite, macfarlanite, and niccolite; the arsenates, erythrite and annabergite; the sulphides, argentite, sphalerite, galena, pyrite, marcasite and chalcopyrite; the antimonide, animikite; the sulphantimonide, tetrahedrite; and the chloride, cerargyrite. Inflammable gas (probably a distillate from carbonaceous shales affected by the diabase intrusion) was encountered during mining. It probably formed part of the diabase differentiate which contained blocks of carbonaceous shales. Quartz, pink carbonates, and lead and zinc sulphides are the most persistent and uniformly distributed minerals in the vein. The silver ores were the last minerals to be deposited and were for the most part concentrated in two large 'bonanzas' in that part of the Silver Islet vein lying above the 400-foot level, though small aggregates of rich silver ore were found at irregular intervals to the deepest workings in the mine, and silver occurred at a distance of 1½ miles from Silver Islet dike at Morgan's junction.

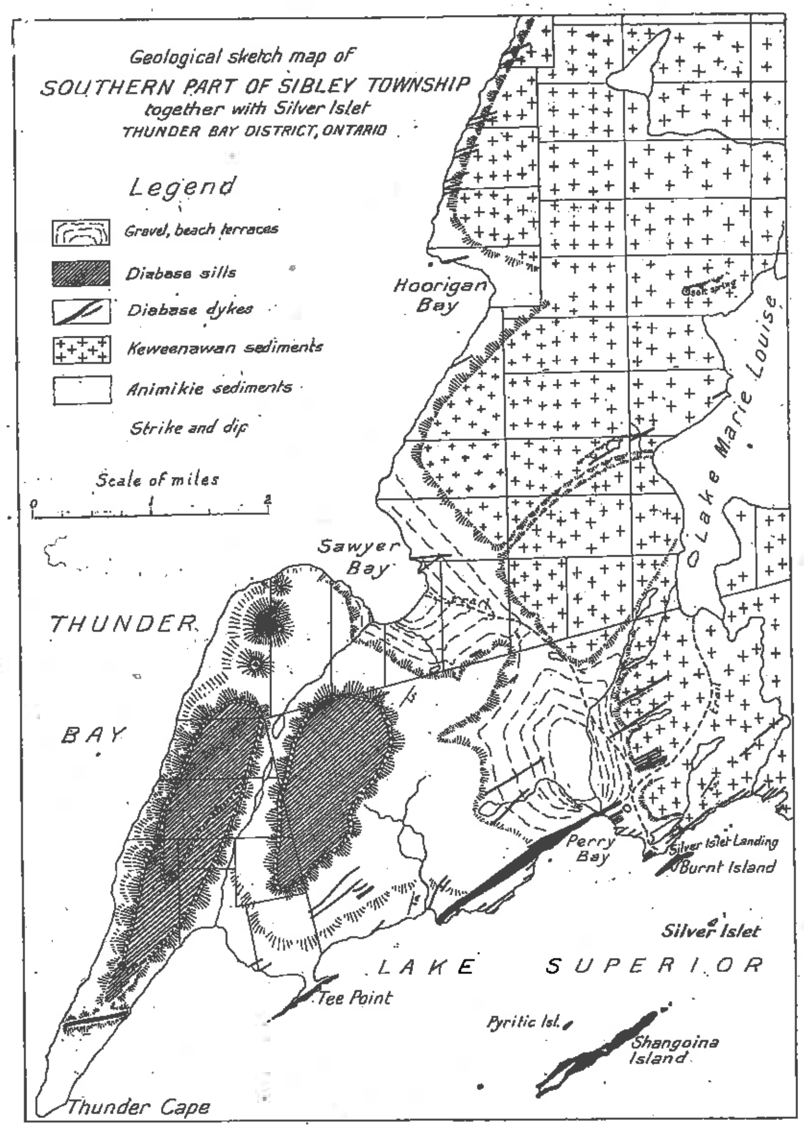
On the first level at Silver Islet, diamond drilling to the east and west was undertaken to find parallel veins; and a small vein was encountered 40 feet to the westward, but subsequent testing by crosscut and drift showed no silver. The mine-waters of Silver Islet were saline.

GENERAL GEOLOGY

Table of Formations

Dikes and sills of diabase and related rocks.	
Keweenawan sediments:	
Red and white tuff and sandstone	100 ft.
Cherty limestone	20 ft.
Red and white tuff and calcareous sandstone	150 ft.
Conglomerate	10 ft.
Animikie sediments:	
Thin-bedded shale and greywacke	50-100 ft.
Grey shale	30 "
Black shale	50 "
Thin-bedded, greenish-grey shale and greywacke or impure sandstone	300+"
Iron formation?	
Conglomerate?	
Basement complex.	

Sufficient fixed data are available to indicate the general geological features of the district (Figs. 4 and 5). Over 1,300 feet of fine-grained, clastic material was deposited on a floor composed, in part at least, of granite. After an interval during which a general uplift took place, a coarser series of normal sediments intimately mixed with fine-grained, volcanic debris was deposited in shallow water over the Animikie with almost perfect accordance. The younger series is assigned to the Keweenawan. Originally both series of sediments were in horizontal layers. Subsequently an extensive system of faults was developed, some of them paralleling the present coast lines of Thunder cape and others at an angle thereto. The sediments have never been folded in the way that characterizes the Keewatin schists, but many local disturbances are responsible for a local mashing in the beds. The general effect of the deformative movements was to give the sediments a regional dip of about 5° to 10° toward the east-southeast. Accompanying the period of faulting, a great volume of diabase was intruded

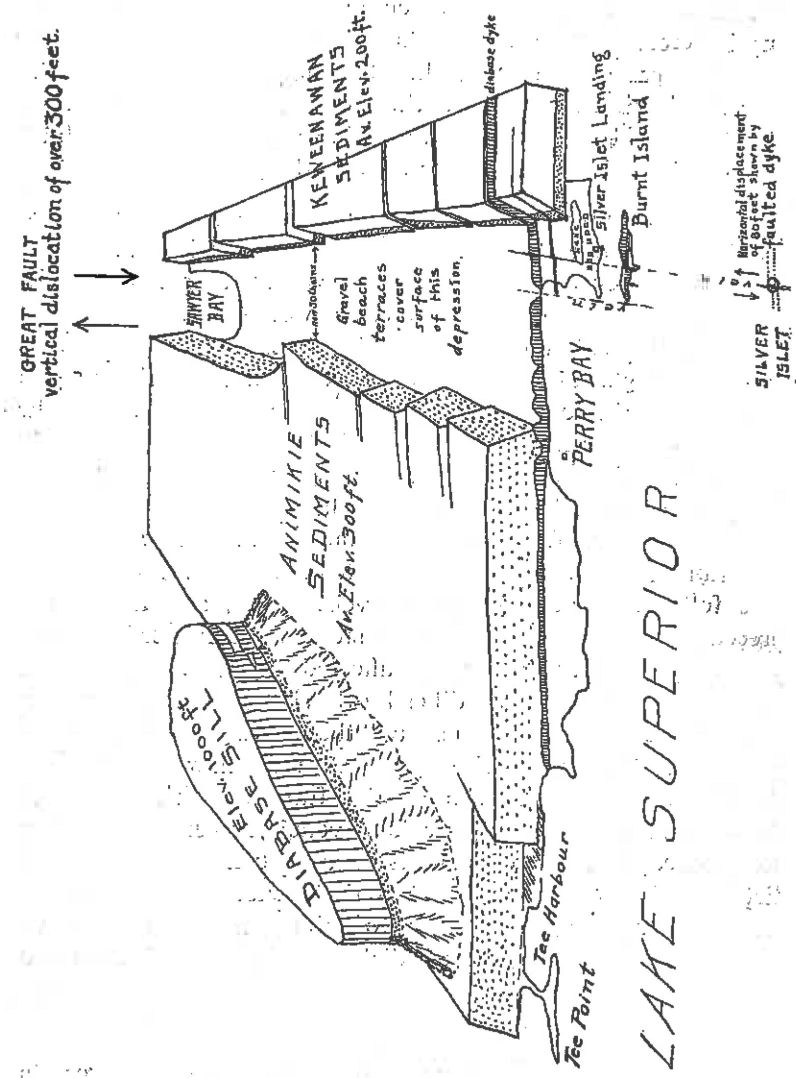


through fissures, and formed dikes in parallel series and sills (sills in this case meaning lateral expansions of dikes in nearly horizontal sheets not necessarily within sediments). It is probable that neither all the faults nor all the intrusions occurred at one time, but much detailed work has yet to be done before the time relations between the faults and intrusions can be determined.

As the diabase cooled, contraction took place, and fissures formed. Most of the fissures, now known to be later than the diabase, trend parallel to the dikes and either lie within them or along their margins, but the general shortening of the dikes induced other fractures which trend across the main parallel series. These fractures were cemented by quartz and calcite and occasionally by other materials conveyed there by solutions.

The diabase intrusions and the great period of faulting of the district took place in late pre-Cambrian time (this age determination being made by correlating the intrusives with those of Minnesota). No clear record of all the geological processes that have gone on since that time can be deciphered in the rocks of the region, except that we see the results of Pleistocene glaciation and erosion of a great water body whose level relative to the land has fluctuated at various times 500 feet or more. A general survey of the district leaves the impression that the great topographic features that have resulted from erosion are of comparatively recent geologic development controlled by structures developed in a remote period.

Topography in Relation to Rock Structure:—The cliffs of Keweenawan sediments that form the western margin of a plain sloping gently toward the east on Sibley peninsula, are regarded as the margin of a great fault. To the north of Sawyer bay the evidence for this is merely that the trend of the cliff front is more even and regular than would be the case if it were formed in any other way, such as by erosion, as in the case of the Niagara-Bruce peninsula escarpment. South of Sawyer bay the direct relation of the cliff front to a fault zone is observable.



The great diabase sills near Thunder cape have, in a measure, protected the underlying sediments from erosion, but undermining by erosion of the sediments has caused the collapse of the marginal parts of the sill, thus forming mesas with irregular margins having cliffs about their edges. The straight-walled west side of Thunder cape presumably represents the same fault as that along the Keweenawan cliffs to the north.

The most pronounced evidence of glacial erosion is evidenced by the lack of talus at the base of certain cliffs where one would expect it to occur. Also the general levelling of a surface, that under conditions of normal erosion would become strongly ridged with upstanding diabase dikes cutting soft strata, is evidence of glacial erosion. A thin deposit of glacial till can be found on the topographic elevations of the district. All the low levels are mantled with stratified silt, sand, or gravel, which were deposited since glacial time. The great depression between Sawyer and Perry bays is an eroded fault zone, and is covered with gravel that formed a succession of beach terraces in both directions from a certain intermediate point as the waters of the lake fell successively to lower levels. Now, and during the preceding period, when the lake stood at a higher level, the pre-Cambrian sediments were affected to a much greater degree than the diabase dikes by the shoreline erosion, and a series of diabase ridges constitute the fringing reefs, T-shaped peninsulas, and upstanding groynes that now characterize the shoreline. The rocks in the fractured fault-zones of the district and any large calcite veins that exist were even more susceptible to erosion than any other rocks of the district. It is to be expected that such features would occupy topographic depressions of about the same relative magnitude as the ridges of diabase. Such depressions are favourable for the deposition of recent unconsolidated materials.

The purpose of this paper is to present briefly the results of an enquiry as to whether other deposits like that at Silver Islet might be expected to occur elesewhere in that general vicinity. Since the time when Silver Islet mine was abandoned other silver deposits have been found and carefully studied elsewhere in Ontario and much additional geological information has been gathered in the Thunder Bay district. This knowledge might be expected to assist in the consideration of the Thunder Cape district.

The enquiry would be greatly facilitated if the genesis of the Silver Islet orebody were known.

Mr. Ingall, writing in 1887, discussed the genesis of the group of silver deposits examined by him in the Lake Superior district. He gives facts to show that the aggregations of silver ore were deposited subsequent to the greater part of the vein material, especially in the cleavable minerals of the vein, and suggests that the silver was leached by circulating ground waters from decomposing diabase intrusives near the veins and that the mineralized solutions deposited silver in those crevices that existed in the adjacent or underlying rock. Carbon in the country rock is supposed to have favourably influenced the deposition of silver. The conditions favourable to deposition then would be: (1) a silver-bearing intrusive with joints or fissures allowing of the infiltration of decomposing waters; (2) fissures or other permeable channels in the neighbouring or underlying rock; and (3) the presence of carbon in some form adjacent to the permeable channels.

This theory favours the assumption of the occurrence of silver deposits on the mainland. The great diabase intrusives exposed near Thunder cape are presumably the exact equivalent of the intrusive which supplied the Silver Islet deposit. Carbon occurs in the Animikie sediments, presumably throughout the district, although at different elevations in the faulted blocks. All the conditions would be fulfilled if what had been permeable channels could be recognized through the carbon-bearing rocks in this section. Though many veins are known to exist on the mainland none has been found cutting carbon-bearing rocks; but presumably all veins that extend to great depth somewhere pass through this formation. Deep veins might be expected to occur in the fault zones of the district.

The silver deposits of Cobalt and Gowganda were probably formed at the same time, and by the same agencies. Theories of ore genesis have been advanced by W. G. Miller, W. H. Collins, and others. The most careful study of the deposits in these districts indicates that the silver was deposited in veins of a certain peculiar type and age and that the minerals that first formed in the veins, quartz, carbonates and certain sulphides, were the end products of differentiation in thick, quartz diabase sills. A series of differentiation products can be recognized in connection with the great diabase intrusives, and a reddish, fine-grained rock, aplite, represents a phase of the differentiate which formed prior to the deposition of vein-filling residue. In the Gowganda district a study of magmatic differentiation did not clearly prove that silver, cobalt, nickel, and bismuth minerals were direct products of the diabase magma, but their occurrence in only those veins that might be regarded as the residual magmatic products suggests that they formed part of residue. Regarding transportation and deposition it is supposed that the contraction of a cooling diabase sill caused fissures to open. These would develop at the upper and lower parts of the sill and extend into the adjoining country rock, while molten material still existed in the interior of the sill; part of the molten material would ooze into these fissures; as cooling and contraction proceeded other fissures would form into which the later phases of magmatic differentiation would flow Presumably the minerals now found in the veins were precipitated from aqueous solutions which were moving (circulating), and these minerals represent only those constituents which, by reason of temperate and pressure changes or chemical reaction, could no longer be carried in solution Carbon does not occur in the rocks of this district as in the Lake Superior district. The rich silver deposits are found mainly within 300 feet of the surface. This may mean that there has been a secondary enrichment of the veins by downward circulating waters,[1] or, as Miller believes, it may be that the silver, at a late stage coming into the veins, which were already occupied by an earlier generation of cobalt-nickel minerals or other minerals having a similar chemical influence on silver solutions, was precipitated by chemical reactions rather rapidly, and hence close to its place of origin in the diabase sill, which position happens to be near the present surface.

[1]Lindgren, "Mineral Deposits," p. 513.

A consideration of this theory of silver-ore genesis with reference to a geological relation in the vicinity of Silver Islet indicates that all the necessary conditions for other silver deposits are present. Remnants of a great diabase sill can be seen near Thunder cape and if one is warranted in projecting laterally for 3 miles a sheet known to be over 400 feet thick, then it might have extended over the site of Silver Islet. If the Silver Islet deposit was derived from the fluid magmatic residue coming downward from the sill, then all veins of the same period of formation as that at Silver Islet, and which occur closer to the sill remnants than Silver Islet, might be expected to carry similar minerals. There may be many veins of the same period of formation as that at Silver Islet, but the data at hand indicate that those that can be assigned to that period with the greatest confidence are the ones occurring in the great fault zone between Perry and Sawyer bays, of which only two have been located; these are the landward extension of Silver Islet vein, and the submerged vein discovered by Capt. J. W. Cross.

The writer's theory regarding the genesis of the Silver Islet deposit is briefly as follows: A great sill, which may be considered as the immediate parent magma of the visible intrusives of the district, cooled at depth. A liquid residue was formed by a process of differentation in it and in those large upward extensions of it which cooled slowly. The volatile parts of the magma, including silver and other metallic constituents, were concentrated in this liquid residue. Fissures, due to crustal movements or contraction of the intrusives which penetrated to any of these liquid residues, would receive magmatic contributions to the other underground waters in them; and the local high temperatures near these points of magmatic contribution would set up strong convection currents. The primary deposition of the mineral-forming magmatic contributions, in so far as they were governed by the circulation of waters in the veins, would occur *above* the points at which they were contributed.

Silver was presumably contributed to all veins that tapped the graphitic aplite of Silver Islet dike before it solidified; but local conditions seem to suggest that some similar source was present at greater depth, extending over a wider area, and existing as a liquid after the solidification of this particular differentiate. Hence the original deposit in Silver Islet vein is supposed to have received contributions from the differentiate of a sill at depth. There is some evidence for believing that the deposition of silver minerals was not influenced by the presence of carbon in country rocks. After the vein was formed, its upper part was exposed for a considerable time to erosive agencies. This implies that it stood up as a land mass and that ground waters containing oxygen and carbon dioxide circulated through it. Through the agency of these waters a secondary enrichment of silver was brought about and native silver was deposited at successively lower levels as the surface was eroded. This inference is based on the following considerations: the large orebodies occurred only in the upper part of the vein; they were broad near the surface and tapered downward; the silver was deposited later than the gangue minerals and frequently in the cleavage planes of the cleavable minerals. In Lindgren's textbook on "Mineral Deposits" 1st. ed., p. 586, this statement appears, "Native silver is not, as a rule, a primary mineral in the deposits which contain it." The 'floors' in the Silver Islet deposit possibly mark the positions successively occupied by the water table during the period of surface erosion.

Presumably the position of the finally deposited orebodies was largely influenced by the distribution of the primary silver minerals in the upper part of the original vein, and by the permeability of the vein. The erosion which took place during the glacial epoch was locally so intense that all the leached upper part of Silver Islet vein was removed.

If this theory of genesis is correct other silver deposits of the same character as that of Silver Islet might be expected to occur on the mainland in that vicinity. The source, the differentiating diabasic magma underground, presumably underlies all the southern part of Sibley peninsula and a wide area beyond, assuming that the abundant diabase dikes in that vicinity are connected with it. The deep fissures through which the magmatic residues escaped would most likely be found at the present surface within the great fault zone that extends from Perry bay to Sawyer bay, or in any other fault zone of the district along which movement took place soon after the intrusion of the diabase in that locality. Silver deposits due to secondary enrichment might be expected in the more permeable parts of these veins at horizons equivalent in relation to the pre-glacial surface to that at Silver Islet. So far as known the rock surface within the fault zone referred to is practically equivalent in all respects to the surface of Silver Islet.

Under any hypothesis as yet advanced relating to silver deposits in northern Ontario, no particularly unfavourable areas can be delimited for the prospector examining the district near Thunder cape; for if Mr. Ingall's view is correct, i.e., that ordinary underground waters leach silver from diabase masses and deposit bonanzas in carbon-bearing rocks traversed by fissures, then all persistent veins deserve examination, since carbon-bearing rocks underlie the whole area, and the Thunder cape diabase may have extended over the whole area and furnished a source.

If a similar origin to that suggested by Miller for the Cobalt district is applied here, i.e., that mineralized solutions traversed both the upper and lower fissured surfaces of a great sill, the same ubiquity of distribution might be expected provided that one pictures an extension of the Thunder Cape sill.

If the writer's suggestion, that the ores were contributed by a deep-seated parent sill of great extent as well as by its associated large dikes, is correct, then every vein which lies in a major dislocation, or otherwise gives signs of extending to considerable depth, is likely to be proven of interest to prospectors.

The majority of the veins that appear in outcrops in this locality do so by reason of a fortunate combination of resistance in themselves, or in the country rock, to erosion or to special position with relation to the wave action of Lake Superior.

The systematic prospector in this locality should not feel discouraged by the observed barrenness of the special type of veins that are most abundantly exposed, i.e., the narrow veins that trend parallel to the dikes. The type of vein that is sought, if it is to have the character of the Silver Islet vein, is a wide vein, and it will be composed principally of calcite and dolomite. Such a vein would not be likely to form a topographic prominence of itself and, since there is the usual mantle of glacial and recent debris, digging would probably be necessary to expose it. This feature of ease of erosion in the type of vein sought for would be a most discouraging feature if it were not that there are other factors suggesting favourable localities. The known rich vein occupies a fault within a greater fault zone. Other similar faults within this fault zone may be expected and their exact position is delimited to a certain extent by a topographic depression of a restricted area that has not been explored on account of a gravel cover of unknown thickness. The trend of veins within the fault zone will probably be parallel to the course of the Silver Islet vein. There are a number of faults known to occur in the southern part of Sibley township, but no definite suggestion of mineralized veins has been found in connection with any but the Perry-Sawyer dislocation.

To sum up, the fault zone between Perry and Sawyer bays is believed to be a favourable prospecting ground for mineralized veins of the Silver Islet type. Various theories regarding the genesis of the Silver Islet deposit do not affect this conclusion adversely. The most important circumstance that assisted in the locating of veins within the great fault zone was the discovery by Capt. Cross, of Silver Islet, of a wide submerged vein near the west end of Burnt island and the Landing. This datum, coupled with the writer's observations, harmonizes with the general conclusion, and on account of its specific character, adds considerable value to this paper.

MOLYBDENITE IN THE LOWER OTTAWA VALLEY*

By M. E. WILSON

Annual General Meeting, Toronto, March, 1920.

The statements emanating from various sources that molybdenum-steel possesses important advantages for certain special purposes, together with the fact that numerous requests with regard to the available supply of molybdenum in Canada are now being received from Europe, seem to indicate that a more active demand for molybdenite may be anticipated in the near future. It may be of interest, therefore, to inquire just how much molybdenum ore is actually available in Canada at the present time, and what the future possibilities of the industry will be in case the demand for the mineral should increase. The present paper is an attempt to answer these questions for that part of Canada from which the greater part of the war-time production of molybdenite was derived, namely, the Ottawa valley.

Geology.—The region in which the molybdenite deposits of the Ottawa valley occur lies along the southern border of the Laurentian highlands and is underlain for the most part by those ancient pre-Cambrian formations, which, because of their highly deformed or otherwise metamorphosed condition, are generally described as a basal complex. The detailed succession of formations within this complex has not yet been completely worked out, but the results of investigations so far completed indicate that it is composed of three principal groups of rocks: (1) a highly metamorphosed group of marine sediments known as the Grenville series: (2) a group of igneous pyroxenic and hornblendic rocks, gabbro, anorthosite, pyroxene syenite, hornblende schist, etc.; and (3) batholithic masses of granite and syenite gneiss intrusive into groups 1 and 2.

TYPES OF DEPOSITS

The molybdenite deposits occurring in the Ottawa valley belong to the following classes:

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(1) Segregations of pyrite, pyrrhotite, fluorite, quartz, etc., in quartz syenite.

(2) Veins of pyrite, pyrrhotite and quartz in granite-syenite gneiss.

(3) Pegmatite dikes and feldspathic quartz veins, and

(4) Contact metamorphic deposits.

Segregations of Pyrite, Pyrrhotite, Fluorite, Quartz, etc., in Quartz Syenite.—The border of the Laurentian highlands to the northwest of Ottawa is marked by a steep rocky northwesterly trending escarpment, which parallels the north shore of the Ottawa river for a distance of approximately 30 miles. At the western termination of this escarpment a stock-like mass of quartz syenite, in which masses of an older porphyritic syenite are included, is exposed throughout an area approximately one mile wide from north to south, and two miles long from east to west. The greater part of the mass lies in the upland above the escarpment, but two outliers outcrop in the lowland below. Within this syenite, molybdenite occurs partly in thin seams and as scattered flakes, but chiefly disseminated in siliceous pyritic aggregates that have apparently segregated from the syenite. Five ore masses of this segregated type in all have been discovered, but most of these occurrences are too small to be of commercial importance, and it is the main deposit outcropping in the low northwesterly-trending ridge at the south end of lots 9 and 10 in range VII, Onslow township, the property of the Dominion Molybdenite Company, from which most of the molybdenite produced from the Quyon district has been obtained. This deposit consisted originally of a large central mass of ore lying in the midst of an elongated zone of smaller scattered deposits. This zone was approximately 500 ft. long, 60 ft. wide, and at least 250 ft. deep; the central mass was approximately 200 ft. long, 50 ft. wide, and from 75 to 125 ft. deep. The average molybdenite content was approximately one per cent in the upper part of the central ore mass, and three-quarters of one per cent or less, in the lower part. The scattered ore masses outside the main mass contain from one to one-half

per cent molybdenite, but are separated from one another by large masses of almost barren rock. At the time the Dominion Molybdenite Company ceased work, practically the whole of the central mass and some of the scattered masses had been mined away. The total amount of molybdenite obtained from this deposit was over 800,000 pounds. (Figure 1.)

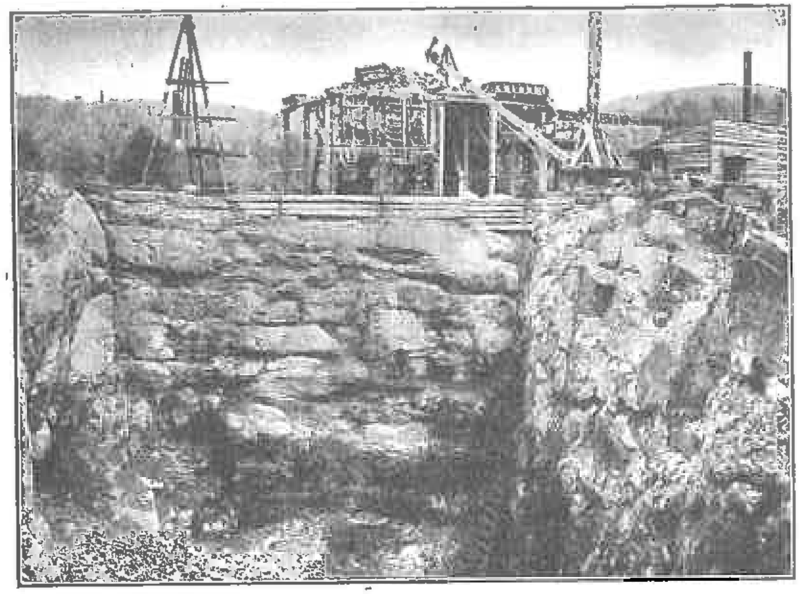

Fig. 1—Pit No. 1, Moss molybdenite mine, lots 9 and 10, range VII, Onslow tp., Pontiac county, Quebec.

Veins of Pyrite, Pyrrhotite, and Quartz in Granite-Syenite Gneiss.—Deposits of this class are very similar to the deposits of the segregated type and differ from them chiefly in form. Examples of the type occur on the properties of the International Molybdenite Company, in Brougham township, and on the property of the Steel Alloys Corporation, in Griffith township, Ontario (Spain mine). The veins are commonly not more than a foot in width, are very irregular and discontinuous, but, in the case of the Spain property, occur fairly close together so that the rock mass as a whole might possibly average one-half of one per cent.

Pegmatite Dikes or Pegmatitic Quartz Veins.—Deposits of this class in the lower Ottawa valley are common, but are usually too low grade or not sufficiently extensive to be worked at a profit.

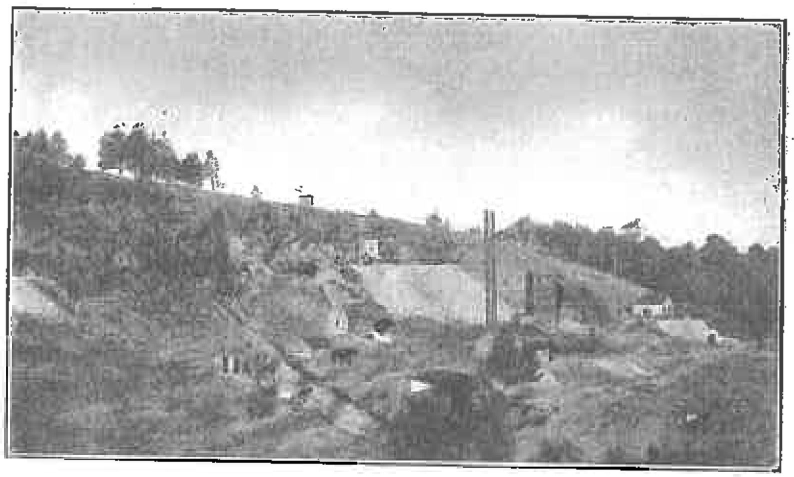

Fig. 2—Mill and other mine buildings, Renfrew Molybdenum Mines, Ltd., lots 8 and 9, con. XI, Brougham tp, Renfrew county, Ontario.

Contact Metamorphic Deposits.—The contact metamorphic type of deposits occurs at points where granite or pegmatite is in contact with the limestone of the Grenville series or where a band or mass of pyroxenite, presumably formed by the metamorphism of limestone, occurs as an inclusion in the granite or pegmatite. The type is most strikingly illustrated by the deposit on the property of the Renfrew Molybdenum Mines, Ltd., in Brougham township, Ontario. This mine is situated on a steep hill slope (Figure 2), the top of which is occupied by a sill-like mass of graphic pegmatite, and the slope by limestone in which masses of pegmatite are included. The molybdenite is of the disseminated type and occurs associated with pyrite and pyrrhotite in a green pyroxenite, which follows the contact

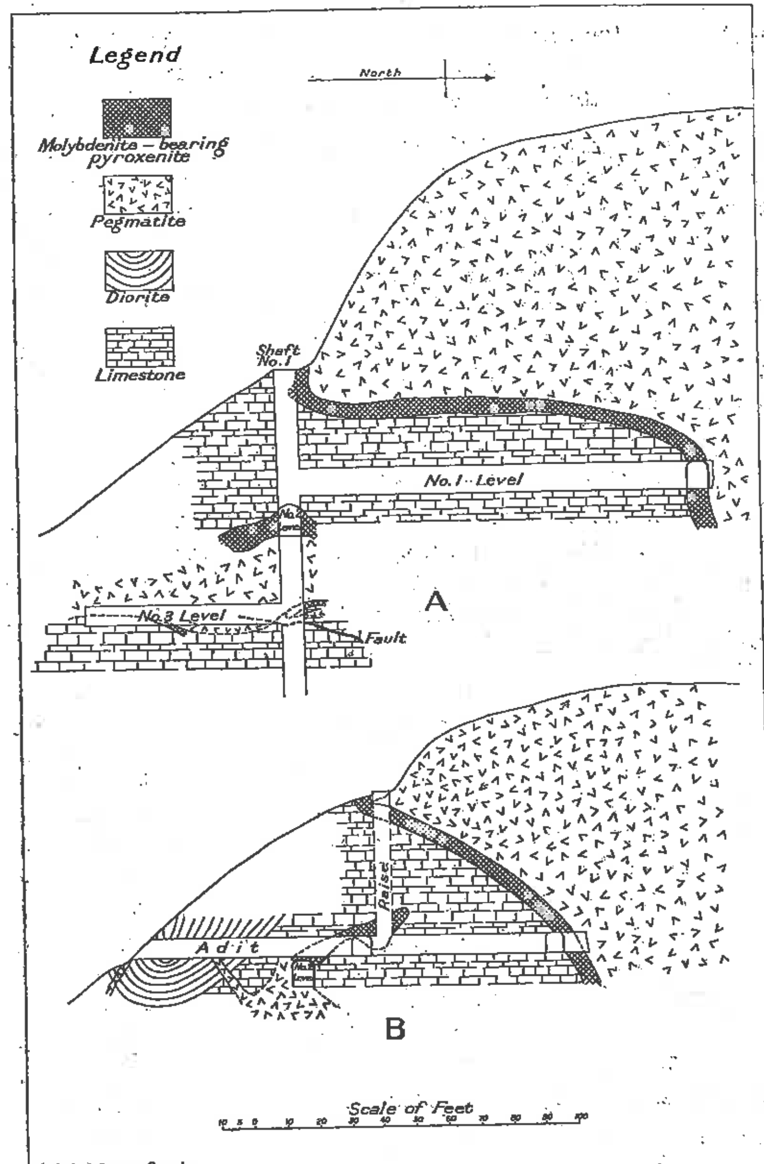

Fig. 3—North-South cross sections showing contact relationships of molybdenite ore-masses, Renfrew Molybdenum Mines, Ltd., lots 8 and 9, con. XI, Brougham tp., Renfrew county, Ontario.

of the limestone and pegmatite in a zone from two to twenty feet in width. The average content of molybdenite in the ore was approximately 0.95 per cent. (Figure 3.)

This property, from the standpoint of output, is the second most important molybdenite mine in Canada, its total production up to the present time being over 100,000 pounds. At the time when mining operations were suspended, however, the known reserves of ore on the property were nearly exhausted; nevertheless, only a small part of the pegmatite-limestone contact present has been prospected; hence it is possible that additional development work might show that large masses of ore still remain in this locality.

In addition to the ore masses on the property of the Renfrew Molybdenum Mines, Ltd., there are numerous other deposits of the contact metamorphic type in the lower Ottawa valley, but most of these are either of relatively small extent or undeveloped. The largest single mass of ore of this class so far observed by the writer was approximately 200 ft. long, 10 ft. wide, and contained an average of about one per cent molybdenite.[1]

CONCLUSION

From the preceding data it may be inferred that in the Ottawa Valley the reserves of molybdenum ore averaging one-half of one per cent, or more, in molybdenite content, are not large. Nevertheless, the prospective conditions on several of the properties are good, and if the market for the product were assured, even at the price prevailing at the present time, development work would be undertaken and some molybdenite, at least, discovered. Furthermore, on the properties on which mills have not been constructed, there is probably sufficient ore averaging one-half of one per cent, or more, in sight, which, by cobbing could be raised to an average grade of two per cent molybdenum

[1]It is obvious that the percentage of molybdenite contained in a deposit of this type cannot be definitely determined except by a mill run on average ore. This percentage is, therefore, merely an estimate determined from a comparison of the ore in this deposit with ore from other deposits of which the average molybdenite content is known.

to supply a small concentrating plant having a capacity of from 25 to 50 tons per day for several years.

It is obviously impossible definitely to foretell what the future production of molybdenite from the lower Ottawa valley will be, when so many important factors bearing on the problem are wanting. At least one company, however, the Steel Alloys Corporation, is carrying on development work; and another company owning a property which the writer has not seen, propose to construct a mill having a capacity of 25 tons per day directly the market conditions warrant the expenditure. It is evident, therefore, that if it were certain that all the molybdenite produced could be sold even at present prices, there would be some production of molybdenite in Canada, and if the price were greater the production would increase accordingly. Up to a certain limit, therefore, the future production of molybdenite in Canada is dependent on the demand rather than the supply available.

THE MICHIPICOTEN IRON RANGES*

By W. H. Collins

Annual General Meeting, Toronto, March, 1920.

Michipicoten district, on the northeast coast of Lake Superior, has a greater variety and quantity of mineral resources than appears to be commonly realized. In 1897, gold was found south of Wawa lake, and has been mined intermittently since then. In 1918, new finds of gold were made a few miles east of Goudreau station, on the Algoma Central Railway. Pyrite mining is becoming an increasingly important industry. A small tonnage of pyrite was recovered, with the iron ore, from the Helen mine, while it was in operation; but since 1913 much larger deposits, near Goudreau, have been developed. A body of about one million tons of high-grade ore has also been found on the Holdsworth property, near Hawk Junction, and several other promising deposits are known elsewhere in the district.

The chief mineral product, however, is iron ore. Since 1900, Michipicoten district has been the foremost producer of iron ore in Ontario and gives substantial evidence of holding that position. Between 1900 and 1918, the Helen mine, belonging to the Algoma Steel Corporation, Sault Ste. Marie, Ontario, yielded 2,823,369 tons of brown iron ore of good quality, and 51,930 tons of pyrite. This deposit is now exhausted, but early in the course of exploitation it was recognized to be simply an oxidized pocket in a much larger body of siderite, and as its approaching exhaustion became apparent, steps were taken to investigate the commercial possibilities of the body of siderite. This deposit was explored in 1916 with diamond drills and found to be an approximately vertical, tabular body, 2,600 feet long, and maintaining a thickness of from 145 to 225 feet to a depth of at least 2,400 feet, as far down as the drill reached. It is estimated that there is almost one hundred million tons of siderite above the 1,800-ft. level.

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This orebody outcrops along the crest of a high ridge that lies in a direction slightly north of east. As a first step towards mining in 1918, a tunnel 12 ft. by 8 ft. in section, with 0.6% grade for drainage was driven into the side of the ridge, so as to pierce the orebody about midway, and 200 feet under the crest. Lateral branches, 8 ft. by 8 ft. in section, into which the ore overhead can be mined when industrial conditions admit of doing so profitably, were driven east and west along the orebody.

The siderite contains approximately 36% iron, 2% manganese, 0.017% phosphorus, 2.50% sulphur, 7% silica and enough lime and magnesia to assist materially in fluxing the ore in the blast-furnace. Before shipping, it has to be roasted to drive off the carbon dioxide and sulphur, leaving a nodulized hematite containing in the neighbourhood of 50% iron, 2.50% to 3% manganese, 0.03% phosphorus, and no water. The finished ore will be hauled about 13 miles over the Algoma Central Railway to Michipicoten Harbour and shipped by lake boats either to Sault Ste. Marie or to Lake Erie ports.

The problem of roasting the siderite, although this method of beneficiation is not employed in America outside Michipicoten district, has already been solved after costly experimentation, by the Algoma Steel Corporation, at the Magpie mine, situated on the Algoma Central Railway, 26 miles from Michipicoten Harbour. The Magpie mine, since it was opened in 1916, has produced about one and a half million tons of siderite, of much the same composition as the Helen ore, from an orebody similar to the Helen deposit in form and geological relationships, though much smaller. Mining has been carried to a depth of 570 feet in a nearly vertical, tabular body 1,400 feet long and from 30 feet to 60 feet thick. The siderite is broken to lumps 3 or 4 inches in diameter and roasted for four hours at a temperature of 1,100° Centigrade in rotary kilns such as are used in cement factories. There are five kilns (each 125 feet long, and with an inside diameter of 6 feet), that are heated from the lower end with a burning jet of powdered soft coal. Three hundred pounds of soft coal and 1½ tons of siderite are required to produce one ton of finished product. The coal is brought from Virginia.

Besides the Helen and Magpie deposits, there are other bodies of siderite, either lower in grade or less favourably situated, which will probably become of value, commercially, in the course of time. Two deposits of brown iron ore of considerable size and of the same nature as the old Helen deposit, have also been found. One of these, at the Josephine mine, close to the Algoma Central Railway, is estimated to contain about a million tons of ore not lower than 50% iron, but the other, near Brant lake, is less conveniently situated and is of lower grade. Deposits of the siderite type are easily discovered, and there does not appear to be any strong probability of further important finds of such ore being made. The brown-ore deposits, on the contrary, can be found only by careful search, and it is by no means unlikely that further bodies will be discovered.

All the minerals, except gold, originate in one kind of geological formation, known as iron-formation. The Michipicoten iron-formations, besides being economically interesting, have always been more or less of an enigma to geologists, and have for these reasons been studied repeatedly, particularly by the geologists of the Ontario Bureau of Mines. Professor A. P. Coleman and the late A. B. Willmot, who were the first to undertake this work and have contributed the most important results, examined the district at various times between 1899 and 1906. Dr. J. M. Bell obtained additional data in 1905, and Mr. A. L. Parsons in 1915. Between 1918 and 1920 the Geological Survey also contributed by collecting additional information for an improved map of the iron-formations in an area 25 miles square near Michipicoten Harbour, and by making a further study of the composition, structure, and origin of the iron-formations. The present paper deals briefly with three conclusions reached from this latest field-study that have a bearing upon the exploration of ore deposits geologically connected with the iron-formations.

For an understanding of these conclusions it need only be said of the general geology of Michipicoten district, that the iron-formations occur exclusively in a great and varied assemblage of early pre-Cambrian volcanic rocks and associated

sediments, known collectively as the Keewatin group. The Keewatin was folded severely in early pre-Cambrian time, was invaded by great bodies of granite, and was afterwards eroded so deeply that the bevelled edges of its various formations now outcrop in narrow, more or less parallel, bands. These bands of iron-formation, being the most resistant of all to erosion, stand up in ridges as high as 250 feet, and are therefore known to prospectors and geologists as iron ranges. In late pre-Cambrian time, the district was affected profoundly by faulting, which broke the belts of iron formation into a complicated arrangement of segments that vary in length from a few hundred feet to several miles.

I.

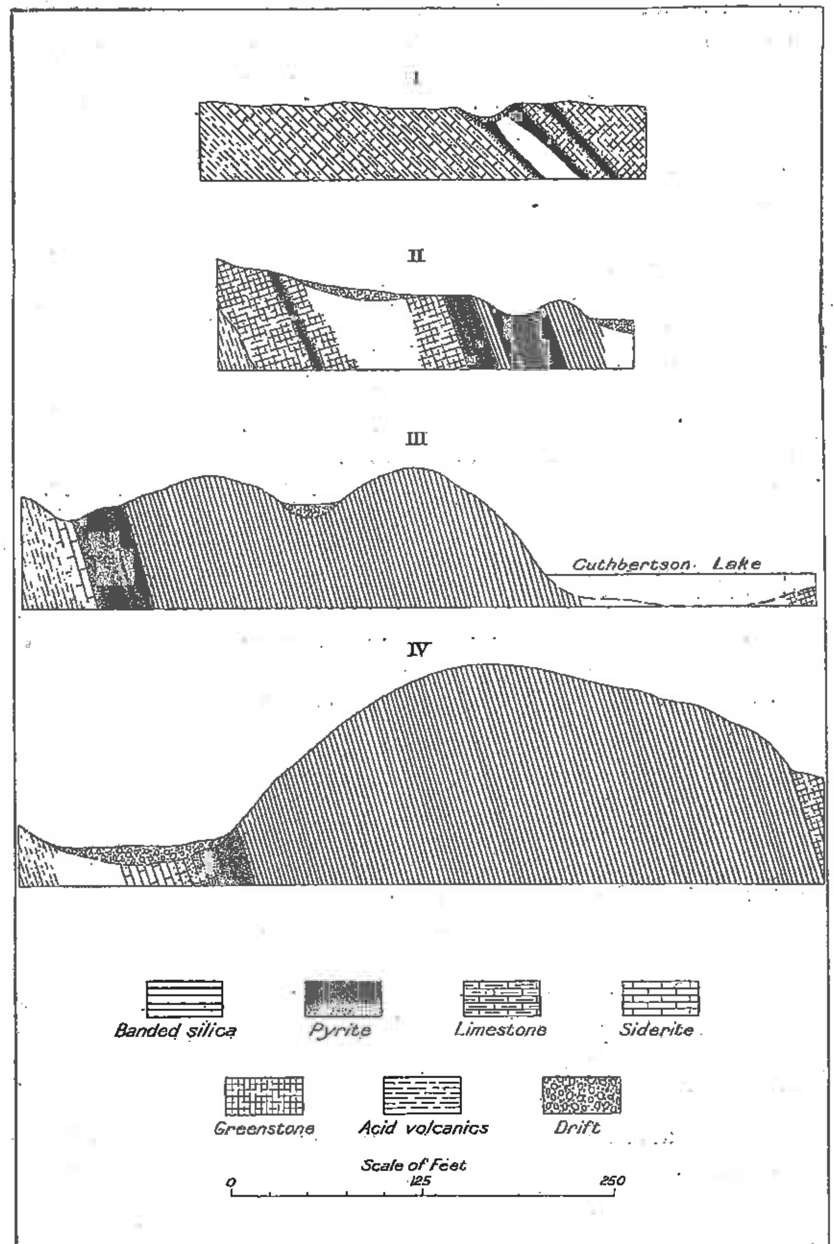
Coleman and Willmot ascertained that each of these ranges, or segments of iron-formation, consists of a banded siliceous (banded silica) member and a massive carbonate (siderite) member composed dominantly of siliceous iron carbonate with pyrite disseminated through it; furthermore, that brown-ore deposits, like that of the old Helen mine, are simply oxidized portions of the siderite member. But they believed, because the iron ranges dip very steeply and have linear outcrops, that they must be closely compressed synclines of iron-formation. Van Hise, Leith, Winchell, Coleman and other geologists had advanced the general hypothesis that iron-formation is a chemical sediment connected in some way with volcanic activity and, of course, laid down in the first place horizontally. But iron-formation bears so little resemblance to ordinary sediments that, after being folded into steep attitudes, it was difficult to tell which side had been the top or the bottom originally.

In 1918 a means was found of distinguishing top from bottom in most though not all cases. The iron formation east of Goudreau is so little folded that it seldom dips more than 70° and in some places not more than 15°; and there is rarely any possible doubt regarding its true attitude. A large amount of surface trenching and diamond-drilling had also been done, affording excellent opportunities for examining the iron-formation in section. It was noticed, in comparing surface and drill-core sections, that the sequences from top to bottom at all places on the Goudreau ranges have a great deal in common. The formation just above the iron-formation is prevailingly an ellipsoidal greenstone. Beneath this and in sharp contact with it comes the banded-silica portion of the iron-formation. This, in turn, gives place, either directly or by a succession of alternating layers, to pyrite. The pyrite, which attains its highest concentration immediately under the banded silica, becomes more and more mixed with carbonate in the downward direction and in this manner grades into the carbonate member. The carbonate member passes downward in somewhat the same gradational fashion to volcanics, of acid composition. To sum up, the downward succession at Goudreau is: (1) a basic-lava flow constituting the cover or hangingwall of the iron-formation; (2) three members, or zones, of banded silica, sulphides (pyrite and pyrrhotite), and carbonates (of iron, lime, magnesia and manganese), constituting the iron-formation; (3) an acid flow or tuff constituting a floor or footwall for the iron-formation.

A more detailed idea of these relationships and of the scale on which the three members of the iron-formation occur at Goudreau, and elsewhere in Michipicoten district, may be got from the accompanying drawing.

This order of succession can be determined in many of the other iron ranges in the district, but all of these are so nearly on edge that it cannot be ascertained from their attitude alone which side corresponds to the original top. In one case, however, the Goetz range, in township 29, range 24, the iron range is paralleled to the north by a band of slate and conglomerate, the pebbles in which are composed principally of banded silica. If it be assumed that the pebbles were derived from the neighbouring iron-formation, the iron-formation must underlie the conglomerate, and its top must face northward; in that case the downward succession conforms with the succession found at Goudreau.

This corroborative test could not be applied elsewhere because of the scarcity of sediments carrying iron-formation



Cross-sections of iron formations showing the stratigraphic arrangement of banded silica, pyrite, and siderite, or sideritic limestone, and the topographic expression of each.

debris; but another, somewhat more speculative, means of testing the uniformity of the order of succession in the iron-formation was found. The original belt of iron-formation, of which the Goetz range is a faulted segment, has a total exposed length of seven miles and an average thickness of several hundred feet. If the width of this formation before folding was at all commensurate to its length, it is likely that, instead of extending downward for an extraordinary distance, the iron-formation would return towards the surface in one or more folds, the truncated outcrops of which would constitute other more or less parallel ranges. If that happened, each adjoining pair of ranges would have to lie face to face and back to back alternately; that is, if the sequence in one range for a given direction were banded-silica-pyrite-siderite it would be siderite-pyrite-banded-silica in the ranges on either side. Now, the dislocated belt of which the Goetz range is one segment is roughly paralleled in this manner by a broken belt of iron-formation lying 6,000 feet to the southeast and by another lying 9,000 feet, or more, to the north.

It has been concluded from evidence of this nature that the iron-formations of Michipicoten have, with subordinate irregularities, a definite and constant stratigraphic sequence. The overlying and underlying volcanics are not everywhere basic and acid respectively; some of the smaller iron-formations lie within greenstones and in a few places there are volcanics of acid or of intermediate composition above as well as beneath. In some cases also, one or another of the three members of the iron-formation is rudimentary or lacking. The Dreany range, for example, has no siderite member and only thin patches of sulphides beneath the banded-silica member; and the Magpie range consists of siderite with no banded silica and only slight local concentrations of pyrite on one side. It is much commoner, however, to find all three members present in the order of banded-silica-sulphides-carbonate, and, in more than half of the ranges, to find an acid fragmental-volcanic beneath and a greenstone-flow above.

Once this stratigraphic feature was known, the iron-formations became an aid rather than a hindrance to working out their own folded and faulted structure and that of the Keewatin. It became comparatively easy to correlate the various faulted segments of iron-formation that had originally been united in longer belts and to trace them across swampy or drift-covered intervals. It was found also that the iron-formations are not all contemporaneous and confined to a single horizon in the Keewatin, but occur at intervals from near the base upwards, through a thickness which is not determinable but may be estimated in thousands of feet. Once it is established that the siderite and pyrite members occur in only one side of the banded silica, trenching and diamond-drilling can be confined to that side of the range. An examination of some of the exploratory work that has been done clearly demonstrates that an earlier knowledge of this relationship would have saved much time and money. Another very general feature observed, which may prove of some use in valuating iron-formation properties, is that the pyrite is most concentrated and highest in sulphur close under the banded silica and tends to become lower in grade and more mixed with carbonates as the carbonate member or footwall of acid schist is approached; likewise, the carbonate (siderite) member tends to decrease in quality and become mixed with replacement vestiges of schist toward the base.

II.

Professor Coleman discovered a number of faults in the Keewatin that have dislocated and displaced the iron ranges since they were folded into their present steep positions. He also observed that many, if not all, displacements were in the same direction, viz., towards the left of an observer looking across the fault. Professor Coleman's observations and deductions have been strongly corroborated by the Geological Survey's recent work. Out of fourteen faults found, in only one is the displacement to the right. As several of these faults have broken the original iron-formation belt into segments and displaced the segments laterally for distances of from 3,000 to 12,000 feet, it is not difficult to understand how greatly they have obstructed exploration of the iron ranges. During the field work of 1919 and 1920, a knowledge of the fact that displacement is to the left proved of distinct aid in discovering faulted extensions of the iron ranges.

The faulting seems to have coincided in time with intrusions of diabase of late pre-Cambrian, probably Keweenawan age, for a considerable share of the fault planes are occupied by dikes of this diabase. It may be mentioned in this connection, that there are other dikes, of olivine diabase, more numerous and intruded later, which have no connection with this faulting and should not be confused with the earlier set.

III.

Most of the known pyrite deposits in the district constitute the middle member of the iron-formation, between the banded silica and the carbonate members (see figure). They appear to be primary in origin, that is, they were laid down as part of the original iron-formation and have not altered in mineral composition to any notable extent since their deposition. This 'range' pyrite has certain distinctive features in all the iron ranges. It is mixed with more or less siderite, depending upon the degree of concentration, and does not usually carry more than 35% of sulphur. It is really a mixture of about 90% pyrite and 10% pyrrhotite and magnetite, and it carries from 1% to 2% of both manganese and zinc.

Besides these primary pyrite deposits, there appears to be another group, of secondary origin, derived from the former, and of an importance that is only now becoming realized by prospectors and operators in the Michipicoten district.

The first of these secondary deposits, recognized by the writer in 1918, occurs in the Rand Consolidated Company's open pit at Goudreau. It is a bed of loose pyrite grains several feet thick, resting upon the glaciated surface of the Keewatin—in fact, one specimen of the pyrite from the bottom of the bed shows a cast of the grooved and polished Keewatin rock surface. The pyrite is overlain by a thinner bed of friable, pure white quartz, and above this is six feet or more of stratified sand and fine gravel firmly cemented with limonite. The water-level in

the pit at the time of observation coincides closely with the boundary between the pyrite and overlying quartz. The water, which drains off an immediately adjacent range of primary pyrite, is so impregnated with iron sulphate that encrustations of this salt are common on the walls of the pit. There seems little doubt that the loose pyrite has originated in post-Glacial time from these surface waters that leach pyrite from the Keewatin iron-formation.

The geological environment of this deposit is sufficiently unlike that of the primary 'range' deposits to distinguish them. The ore has also distinctive qualities. It has no admixed siderite, pyrrhotite or magnetite, only small fractions of 1% of manganese and zinc, and a sulphur content as high as 50%.

Pockets of a similar loose, granular pyrite usually associated with white quartz-sand were found in the old Helen mine, the pyrite in the pockets being quite sharply defined from the surrounding brown iron ore. A sample of this pyrite, supplied by Mr. G. S. Cowie, of the Algoma Steel Corporation, was analyzed and found to resemble the Goudreau secondary pyrite very closely in the above-mentioned respects. The mine water has also been analyzed and found to contain large amounts of ferric sulphate in solution. There is little reason to doubt that the pyrite was derived by the oxidation, solution, transportation, and re-deposition below water-level of disseminated pyrite in the siderite which, by its oxidation, formed the Helen iron-ore deposit. The similarity (physically and in mineral composition) of this pyrite to the Goudreau pyrite-sand is also so close that there seems little doubt of their common mode of origin.

The Holdsworth pyrite deposit probably originated in like manner. It consists of two, and possibly more, upright lenses arranged at short intervals along the strike of the enclosing Keewatin schists, with each one offset a little to the left of the preceding one, probably by a process of drag-folding analogous to the faults of the district. Green schist forms the north wall, and a light sericite schist the south wall. Each of the two explored lenses is about 500 feet in diameter, 25 feet to 30 feet thick and contains about a half-million tons of ore. The walls of the orebody are sharply defined, and there is neither banded silica nor siderite in association with it. The pyrite contains no siderite, only 3% of magnetic matter, only a fraction of 1% of manganese and zinc, and about 47% of sulphur. In all these respects it is much more like the secondary pyrite at the Goudreau and Helen mines than 'range' pyrite. It is a solid fine-grained ore, unlike the loose, sandy pyrite at the Goudreau and Helen mines, but this difference may well be due to its greater age and corresponding tendency to become cemented. It seems likely, owing to the attitude of the lenses and their offset relationships, that the ore was deposited during Keewatin time before the Keewatin formations were folded into their present positions. The Holdsworth deposit is thought to be a secondary accumulation of pyrite, originating, like the Goudreau and Helen pyrite-sand, by a process of leaching and reprecipitation of sulphides from iron-formation, differing from them only in having been formed in pre-Cambrian instead of more recent times.

These secondary pyrite deposits are much higher in sulphur than the 'range' pyrite and have a correspondingly greater economic value, but they are much less easy to discover. The Holdsworth deposit, for example, was represented at the surface only by a few exposures of a rusty, iron-stained band of schist a few feet wide, and by still smaller and rarer streaks of crumbly, black, oxide of iron, called 'black sand' by the prospectors. Unlike the 'range' deposits, a dip needle over this pyrite registers no exceptional magnetic effect. The indications are so easily overlooked that one naturally wonders if there are not other higher-grade secondary deposits in the vicinity of the iron ranges that may yet be found by careful prospecting.

DISCUSSION

MR. BRADLEY STOUGHTON: To what extent are the iron ores beneficiated?

DR. W. H. COLLINS: Mr. Cowie, an official of the Algoma Steel Corporation, is present, and doubtless will be willing to

give precise information on that point. I may say, however, that the siderite averages about 35% metallic iron; by roasting the iron content is raised to 50% or 51%, and the manganese present is saved. The lime and magnesia in the roasted ore make it almost self-fluxing. It is, therefore, a most desirable product.

MR. G. S. COWIE: Dr. Collins' statements in respect of the beneficiation of the ores of the Magpie mine are correct. We raise the iron content from 38% to 50%–52%. We not only do not lose any of the manganese content, but raise this also. Unroasted ore assaying 2.4% to 2.5% manganese, after roasting will assay 2.8%, and sometimes as high as 3% manganese. The ore, as Dr. Collins has remarked, is a very desirable one, 16% to 18% of it being self-fluxing. It may also be mentioned that another large deposit of siderite occurs in the district. It is estimated by Dr. C. K. Leith, of Wisconsin, that there is now in this deposit from 100,000,000 to 150,000,000 tons of ore, and we estimate the available ore to be 100,000,000 tons.

MR. W. P. ALDERSON: I would like to ask Dr. Collins what relation the gold veins bear to the iron formation?

DR. W. H. COLLINS: My intention in mentioning in this paper the existence of gold in Michipicoten district was simply to call attention to the variety of minerals in the district, and their importance to the prospector. As a matter of fact, it is only the iron ores and pyrite that occur in the iron formation. Gold is found in the volcanics associated with the iron formation.

LOST PLACERS OF ONTARIO

By A. P. Coleman

Annual General Meeting, Toronto, March, 1920.

In most gold regions development begins with placer mines, after which lode mines are opened, establishing a more permanent mining industry. The latter stage of development may be lacking, however, as in the Klondike. On the other hand, there are two striking instances in which lode mining has not been preceded by placers. The greatest gold region in the world, the Rand, is practically without placers; but in this case the reason is clear. The gold, though free, is so fine as to be hopelessly lost when the enclosing 'banket' decays. The other important exception is the gold region of Northern Ontario, where the gold is largely coarse and nuggety, so that the explanation given for the Rand will not apply. The lack of known placers in Northern Ontario must, therefore, be explained in some other way.

It has been suggested by the present writer and others that former placer deposits may have been swept away by glacial action during the Ice Age and so mixed up with barren materials as to be lost. There are some facts which appear to support this theory, the most important being the finding of gold for about forty miles along what have been called the Vermilion placers to the south of the Porcupine region. At several points along this stretch, colours of gold can be panned (sometimes even a hundred or more), but so fine as often to be visible only with a lens. The largest nugget found during several days' panning had a value of only four cents. These deposits are in glacial materials more or less assorted by lake action; old river gravels have not been found and bed-rock has not been reached in the small amount of sluicing done during the short-lived placer boom.

It is possible that this gold was brought by the ice-sheet from the Porcupine district about ninety miles to the north; but, if so, the coarse gold of the quartz veins must have suffered

tremendous grinding to be reduced to the fine dust of the Vermilion placers.

On the other hand, it must be remembered that Porcupine is hundreds of feet lower than Meteor lake, where the most northerly placer gold is found, so that rivers could not have transported gold in that direction. On the contrary, the drainage is, and probably always has been, to the north toward James bay. Placer deposits resulting from the weathering and destruction of the gold-bearing rocks of Porcupine must have accumulated along the drainage slope to the north and not toward the watershed to the south. If such placers were formed, have they been completely destroyed or is there a possibility that portions of them still remain? If they remain where should they be looked for?

First, perhaps, one may naturally inquire as to the original magnitude of such placers. Porcupine has already supplied more than $50,000,000 worth of gold; and one mine reports reserves valued at $40,000,000. The probable total production of the region should run into hundreds of millions of dollars, since some of the veins are known to reach a depth of 1,500 feet or more. The rocks of the region show steeply tilted, often vertical, structures, and it may well be that the gold-bearing veins originally extended far above the present level. The mines now being worked may represent merely the lower half of the deposits as they were formed in the beginning, and it is certain that hundreds if not thousands of feet of rock with the enclosed quartz veins have been destroyed by weathering and superficial erosion. What has become of the hundreds of millions of dollars worth of coarse gold set free in that way?

The major part of this destruction took place almost certainly before the deposition of the Cobalt series, when the mountains due to the Algoman elevations were reduced to a peneplain. It is possible that the Huronian Ice Age swept away these earliest placers; but, as shown by Miller and Knight, residual materials underlie the tillite at various places, and it may be that remnants of the pre-Huronian placers still remain somewhere beneath the conglomerate.

However, there has been great erosion of both Huronian and older rocks in the vast period of land conditions since that time, so that Palaeozoic, Mesozoic, or Tertiary placers may have been formed on the slope toward James bay and may still remain in places beneath the sheets of stratified clay which cover so much of the region to the north; or beneath the Palaeozoic rocks on the shores of James bay; or even beneath the Cretaceous sediments found by Keele under the boulder clay on some of the rivers.

The purpose of this paper is to call attention to the fact that in Northern Ontario, perhaps, half of the original gold-bearing quartz has been destroyed, setting free enough gold to form more than one Klondike, and yet no placers of importance have been found. If such deposits still exist they are probably to the north of the Porcupine region and buried beneath later sediments. They may occur beneath any of the sedimentary rocks from Pleistocene clay to Huronian conglomerate, and as they may be disclosed somewhere in the river valleys on the Hudson bay slope, explorers should be on the look-out for them.

DISCUSSION

MR. J. W. EVANS: Some years ago when prospecting in the Vermillion district, we found that the first six inches of sand, below the grass roots, was alone auriferous. There was no gold in the underlying gravel. It would be interesting if Dr. Coleman would explain why the gold was confined to the upper stratum of sand.

DR. A. P. COLEMAN: There are spots where gold is met with at greater depths than that mentioned. I have panned gold from about twelve feet below the surface at Meteor lake, but in many cases, it is true, the gold is found only near the surface. But I do not believe bed-rock has ever been prospected in that locality. It is conceivable that some of those old placers could yet be mined profitably—that is, with the very cheap

methods of dredging they have introduced in the Klondike and other places. Further, I think that the deposits are really of glacial rather than of river origin. That is, the waters in front of the ice have caused the present distribution, which is quite unlike that found in gold-bearing streams.

MR. T. W. GIBSON: While Dr. Coleman is on his feet, I might perhaps ask him to explain why we have not got silver placers. A great deal of silver in the Cobalt region has been removed by glacial action. Where has it gone?

DR. A. P. COLEMAN: Look for it under the clay there.

DR. W. G. MILLER: In prospecting south of Cobalt you find erratic boulders on almost every claim, some quite large boulders. The first specimen of native bismuth I ever got in the Cobalt district was from a boulder about two or three miles south of where the mines are. The boulders had been blasted in the railway cut there. Of course, you might find a large silver-bearing boulder almost anywhere in this country, or even down across the line. I made a rough estimate of the erosion in Cobalt, and I think there has been more ore carried away from there and destroyed than has ever been mined, and now they have mined about 300 million ounces. I would not be surprised if there had been 600 million ounces carried away.

LABOUR TURNOVER IN INDUSTRIAL PLANTS AND ITS MINIMIZATION.

By A. W. Macdonald

Annual Meeting, Mining Society of Nova Scotia, Glace Bay, May, 1920.

It is only within the past few years that the question of this factor in the labour problem has received very much attention from executives. When it is realized that, in producing, labour cost is far greater than material cost, it behooves companies employing large numbers of men to devise ways and means of reducing the 'turnover' of labour in their plants to the lowest figure possible. It has to be admitted that placing 'green' men on work is one of the most fruitful causes of a high accident rate.

Comparatively few employers have realized that for every man on their pay-roll they are probably hiring at least one new man every year. This shows a labour 'turnover' of 100% and should be a sufficient argument to cause any employer to study the reason for such a condition. Any manager or superintendent of an industrial or other plant employing large numbers of men, who is still of the opinion that the methods that obtained ten years or more ago are good enough today, as far as the hiring and discharging of workmen is concerned, is making a mistake that is costing his company dearly.

It may be well to give some consideration to what it costs a company to make a new employee efficient, that is, to bring him from the stage of a 'green man' to that of a productive workman.

Some of the sources of this cost are:

(a) The difference between a standard day's work and that which a new man does while accustoming himself to his work.

(b) The extra supervision required by a green man.

(c) Interference with the work of other men.

(d) Accidents caused by green men.

Various executives have computed the total amount of the costs according to their own conditions, and the results range from \$25 to \$100 per man as the cost to a company of making a green man a productive workman.

It is evident, therefore, that the cost of replacing workmen is enormous, and the necessity of taking drastic measures to prevent labour 'turnover' is of first importance.

Of course, to get rid of an employee is far easier than to help him 'make good,' and to allow him to leave is less troublesome than to find out in advance any condition of dissatisfaction and to attempt to remedy it.

One of the chief reasons why this matter has received so little attention from companies in the past is that they have no records to show what their labour turnover was, and therefore they did not know what it was costing them.

In the last analysis, this hiring of men to replace men that have left the employ, or in other words labour turnover, is simply wasting labour, particularly at this time when men are scarce and most companies have already passed from the position of buying labour to selling employment, and it is clearly a matter of sound business for industrial organizations to hold out inducements for capable men to enter their employ, and to develop fully the capabilities of those they engage.

A company engages in business for the sole purpose of making profits, and any activity of the company that does not tend toward that end is poor business. Large companies developed slowly from the primitive stage of industry in which each employer was his own foreman, and the tradition has been handed down that in order to preserve discipline the foreman must be able to hire and discharge his own men.

It has been pointed out in a foregoing paragraph that the labour cost of production is very much in excess of the material cost, probably in some instances two and one-half or three times as much, and we find that every company purchases its material through a well-organized purchasing department, while in the majority of cases its labour supply is picked up haphazard. More attention is certainly paid to the purchasing of material and the designing of equipment than to the selection of workmen. This, in no small measure, contributes to unrest and dissatisfaction among workmen, and consequently is a dominant factor in labour turnover.

Work of all kinds can be analyzed to determine the qualifications necessary for its performance, and in some places efforts are being made by executives along the line of selecting and training workmen. The time is not distant when this will be a regular part of industrial activity. The majority of men are neither lazy nor unwilling to work. The trouble is that they are picked up without any effort at selection and placed at work for which they are unfitted.

The question of hiring and discharging workmen, on which alone rests the labour turnover of a plant, may be placed under two headings.

(1) Hiring and discharging by foremen.

(2) Hiring and discharging through a properly organized employment department.

In the first place, hiring by the foremen frequently leads to practices that are detrimental to all concerned. It very often leads to the building up of racial and other cliques in a department that will cause trouble later. This is inevitable, as the only source of labour supply that is open to a foreman is through his relatives and friends and the friends of the men in his gang.

The only basis of judgement the foreman has in interviewing an applicant for work is the impression the man makes on him at the time. If the applicant is unknown to him personally, he cannot tell anything at all about his ability, and he has not the time or the means at his disposal to make the enquiry necessary to determine even approximately the fitness of the man for the work that has to be performed.

We can now consider the centralized hiring of employees through an employment department.

One of the objections to this method was that it interfered with the authority of the foreman and superintendents. As a matter of fact it does nothing of the sort. It does not mean that they cannot dispense with the services of any man that is not doing his work in an efficient way. Instead of the faulty workman being discharged from the plant to the street, he is sent back to the employment office where a further effort will be made to have him placed on a job where he will fit. It is evident that, although a workman may not suit one job, he may prove satisfactory on another. If, after a reasonable number of trials have been made, the man persists in being 'no good', he should be discharged. In a modern organization, no man should be allowed to leave the service without a searching enquiry into his reasons for doing so. These interviews with men that are leaving the employ frequently reveal objectionable conditions that in many cases can be easily remedied, and that are actually a disadvantage to the company as well as to the workman. The modern employment bureau is the only agency that is equipped to handle the problem of distinguishing between the man who is 'no good' and the man who is wrongly placed.

Certain things are essential to the successful centralized hiring of workmen. In the first place, the employment manager should be personally familiar with the various works for which he has to secure workmen. He should have notice, as far as possible in advance, of the number and class of men required, in order that he can provide them when needed. He should have sufficient assistants so that his time will not be used up doing work that can be equally as well done by an ordinary clerk. He should have the standing and the authority in the organization that will enable him to perform his duties satisfactorily in order that the company for which he is working will secure the best possible results.

SUMMARY

(1) The cost of labour turnover in industry is so large as to justify the adoption of any means to bring about its reduction.

(2) The most important and efficient method is the establishment of a properly administered employment department.

(3) Hiring through an employment department does not impair the authority of the foreman or superintendent immediately in charge of the work.

(4) He hires his men from the employment department and discharges them back to it instead of to the street.

(5) The employment department is able to give special attention to the applicant for work, and properly conducted, it is able to devote special skill and knowledge to selling employment in the organization to the workman.

GENERAL INDEX

NOTE.—The names of authors of papers are printed in small capital letters; the titles of papers in italics.