MINES BRANCH

DEPARTMENT OF THE INTERIOR.

HONOURABLE FRANK OLIVER, M. P., MINISTER.

EUGENE HAANEL, PH. D., SUPERINTENDENT OF MINES.

REPORT

OF THE

COMMISSION

APPOINTED TO INVESTIGATE THE ZINC RESOURCES OF BRITISH COLUMBIA AND THE CONDITIONS AFFECTING THEIR EXPLOITATION.

OTTAWA, CANADA,

1906.

OTTAWA, 27th September, 1906.

SIR,—

I have the honour to submit herewith the "Report of the Commission appointed to investigate the Zinc Resources of British Columbia and the conditions affecting their exploitation."

The investigation, results of which are embodied in the Report now presented, was undertaken in response to petitions from the Silver-lead Association and Associated Boards of Trade of British Columbia.

I was directed by you to prepare a Memorandum on this subject outlining the work to be done and submitting names for your approval of the staff who were to be entrusted with this examination.

On approval of this Memorandum, Mr. Walter Renton Ingalls, Editor of "The Engineering and Mining Journal," New York City, and author of an extensive work on the Metallurgy of Zinc and a treatise on the Occurrence and Distribution of Zinc Ores, the commercial and technical conditions affecting the production of Spelter, etc., was appointed Chief of staff of the Zinc Commission with Mr. Philip Argall, M.E., of Denver, Colorado, and Mr. A. C. Gardé, of Nelson, B.C., as his assistants, the former taking charge of the field work in connection with the developed mines of the Province, the latter acting as Mr. Argall's assistant.

To render the report regarding the Zinc Resources of the Province as complete as possible, Dr. A. E. Barlow and Mr. Joseph Keele, of the Geological Survey Department, were detailed to investigate the undeveloped zinc ore deposits.

Through the courtesy of the Honourable the Minister of Mines of British Columbia information was also received from the Gold Commissioners of zinc ore occurrences in their respective districts.

The metallurgical investigation of the samples of ore collected was conducted by Mr. Henry E. Wood, of Denver, Colorado, under the supervision of Mr. Argall and in consultation with Mr. Ingalls.

I have the honour to be,

Sir,

Your obedient servant,

EUGENE HAANEL,

Superintendent of Mines.

HONOURABLE FRANK OLIVER, M.P.,

Minister of the Interior,

Ottawa.

LETTER OF INSTRUCTION.

OTTAWA, 14th August, 1905.

DEAR SIR,—

You are hereby authorized to make an investigation of the Zinc Resources of British Columbia and their commercial possibilities.

This examination is to cover:—

1st—Examination of the present development of the mines to determine approximately the tonnage of zinc ore immediately available, its occurrence and character and the future prospects, together with the cost of mining.

2nd—Examination of the present methods of milling.

3rd—Investigation of the adaptability of the ores to the new methods of concentration (magnetic, electrostatic, etc.).

4th—Study of the conditions affecting marketing of the concentrate, including the question of smelting in the Province or elsewhere in Canada.

5th—Investigation of the possibility of special utilization of the zinc ore of high silver content.

Mr. Philip Argall, M.E., of Denver, Colorado, and Mr. A. C. Gardé, of Nelson, B.C., will act as your assistants, the former taking charge of the field work, the latter acting as Mr. Argall's assistant. These parties are to report to you the results of their investigations, made in accordance with full instructions to them from you.

Your recommendation that the investigation of the adaptability of the ores of British Columbia to magnetic and electrostatic concentration, etc., be undertaken by Henry E. Wood, of Denver, Colorado, is hereby accepted.

Upon the completion of the field work, or sooner if advisable, you are directed to make a tour of the zinc districts of British Columbia to obtain such personal view of the economic conditions as will enable you to form a sound judgment regarding the establishment of zinc smelters, fuel supply and strategical railway locations and such other data as are necessary to arrive at proper conclusions affecting the development of the Zinc Industry of British Columbia.

Your report dealing with the economic features of the enquiry is to contain an analysis and summary of the data collected under your direction by your assistants in the field and in the concentration laboratory; the individual reports of Messrs. Argall, Gardé and Wood are to appear in the full report.

Yours very truly,

EUGENE HAANEL,
Superintendent of Mines.

WALTER RENTON INGALLS, Esq., M.E.,
Editor "Engineering & Mining Journal,"
New York, N.Y., U.S.A.

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REPORT

ON THE

ZINC RESOURCES OF BRITISH COLUMBIA

AND

THEIR COMMERCIAL EXPLOITATION.

BY

WALTER RENTON INGALLS.

Editor of the Engineering and Mining Journal,
Editor of The Mineral Industry.
Member American Institute of Mining Engineers.
Member Institution of Mining and Metallurgy.
Member Society of Chemical Industry.
Member American Chemical Society, Etc.

1906.

SIR,—

Herewith I beg to present my report on the Zinc Resources of British Columbia and Their Commercial Exploitation; and the accompanying reports of Mr. Philip Argall, Mr. Alfred C. Gardé, Dr. A. E. Barlow, Mr. Henry Harris and Mr. Henry E. Wood, together with reports by certain of the Gold Commissioners of British Columbia, received through the courtesy of the Minister of Mines for that Province.

Upon receipt of my appointment by you to undertake the investigation of the zinc resources of British Columbia, and their utilization, plans for the work were immediately inaugurated. Mr. Philip Argall, to whom the charge of the field work was entrusted, left Denver for British Columbia on Sept. 1, 1905, and remained in the field until Nov. 20, during which time he examined the developed mines and mills in operation in the West Kootenay and Southeast Kootenay districts. In the West Kootenay he was assisted by Mr. Alfred C. Gardé, who also examined certain mines independently, which work is described in his separate report.

It was originally contemplated to examine only the developed mines of the Province, but upon its appearing advisable later on to investigate the undeveloped mines and some of the reported occurrences of zinc in various parts of the Province, Dr. Alfred E. Barlow, of the Dominion Geological Survey was commissioned to undertake this part of the work, in connection with which he left Ottawa, with Mr. Joseph Keele as assistant, on Oct. 7, 1905. Owing to the lateness of the season and the great scope of territory to be covered, Dr. Barlow was unable to visit many of the reported occurrences of zinc ore, but he examined prospects on Vancouver Island, the Coast and in the Slocan, Boundary and Southeast Kootenay districts. Additional information as to the undeveloped mines was received from the Gold Commissioners of British Columbia.

As the work of examining the mines approached completion, I visited British Columbia, leaving New York Oct. 19, and returning Nov. 30, and inspected the principal mines and silver-lead and zinc smelting works of that Province and Alberta, and investigated the general mining, transportation and metallurgical conditions.

Samples of ore sent from British Columbia to Denver, Colo., for the purpose of metallurgical investigation, reached Denver toward the end of December, 1905, whereupon the experimental work was immediately begun, being continued without interruption through the early months of 1906 and completed about the middle of May, 1906. This work was done by Mr. Henry E. Wood, of Denver, under the supervision of Mr. Argall and in consultation with myself. Mr. Argall personally conducted a large part of the experiments and worked up all the results.

Some experiments with the flotation process were made by me at New York, with the assistance of Prof. W. R. Crane, of the School of Mines, Columbia University, and Mr. H. F. Seifert.

The assaying and chemical work required in British Columbia was done by Mr. Henry Harris, assistant superintendent of the Hall Mining and Smelting Co., of Nelson, B.C.; that required in Denver was done by Henry E. Wood & Co. A considerable amount of special chemical work in behalf of the Commission was done by Mr. W. George Waring, of Webb City, Mo. To all of these collaborators credit is due for able and expert performance of the work entrusted to them.

The results of the work of the Commission are fully incorporated in the following reports.

Yours respectfully,

WALTER RENTON INGALLS.

Dr. EUGENE HAANEL,

Superintendent of Mines,
Mines Branch,

Department of the Interior,
OTTAWA,
Canada.

INTRODUCTORY.

British Columbia, the most westerly Province of the Confederation forming the Dominion of Canada, comprises principally that section of British North America lying to the westward of the summit of the Rocky Mountains. The northern boundary of the Province is the 60th parallel of latitude; its southern boundary is the United States of America, or practically the 49th parallel; on the west it is bounded by the Pacific Ocean, and on the east by the Rocky Mountains as far north as the 54th parallel; beyond that by the 120th meridian of west longitude. The total area of British Columbia is about 382,000 square miles The country is traversed in a north-westerly direction by four more or less continuous chains of mountains, between which lie long and generally narrow valleys. These valleys form the channels of streams, which drain into the Columbia River; and in the southern part of the Province several of them are occupied by long, rather narrow, navigable lakes, affording means for water-transportation, which have been extensively employed. This system of lakes has had an important bearing upon the development of the mineral resources of the Province.

It was not until 1893 that the lode mines of British Columbia really began to be productive, the output from this source during the six years immediately prior to that date, amounting to an average value of only $60,000 a year, was derived from selected rich ores found near the existing lines of transportation.

In 1893, however, the value of the production of the lode mines of the Province rose to $300,000, since which time there has been a steady increase, until in 1901 the output from this class of mining reached a value of $13,683,044. It fell off slightly in 1902, but the decrease was due principally to the lower market value prevailing, and in 1903 an upward tendency again became apparent.*

HISTORICAL.

The discovery of zinc ore in British Columbia was undoubtedly contemporaneous with the discovery of silver-lead ore, because just as in the country west of the Rocky Mountains in the United States, these ores commonly occur in British Columbia in close association. The development of the industry in connection with these ores was precisely similar in both

* Report of the Provincial Mineralogist, British Columbia.

countries. In the early years there was no market for the zinc ore as such, and as a constituent of silver-lead ore it was a detriment to the value of the latter. The object of the miner, in order to conform to the requirements of the silver-lead smelters, was consequently to keep the percentage of zinc in the ore shipped as low as possible. Large quantities of zinc ore were, therefore, removed from the ores by hand-sorting or mechanical concentration and thrown away; in many cases beyond recovery; in a few cases into separate dumps where it could be held pending the development of a market, which in the United States was foreseen by a few mine owners as far back as 1885. Whenever possible, zinc ore was passed by in the mines.

Previous to 1899, the supply of zinc ore smelted in the United States was obtained chiefly from Missouri, Kansas, Wisconsin, New Jersey, and Virginia, with comparatively small quantities from Tennessee, and Arkansas. There was no ore received from the country west of Kansas, except from a group of mines near Hanover, N. M., whence some shipments were made about 1893, and possibly some small, spasmodic shipments from other localities, of which no record has been preserved.*

The ore from Hanover was sent to Mineral Point, Wis., and Waukegan, Ill. The freight rate to those points was $12 per 2000 lb. Under the market conditions of that time, it being a period of general industrial depression and low prices, there was no profit in the business and the exploitation of the mines ceased.

In the summer of 1899, certain smelters in Kansas received small shipments of blende concentrate from Creede, Colo. The real development of the zinc industry west of the Rocky Mountains may be dated from this time. At first, the Colorado ore was regarded askance, although that received from Creede was really a superior ore by any standard save that which existed among Kansas-Missouri smelters, who based their ideas at that time upon the ore of remarkable purity which was afforded by the Joplin district. They considered an iron content of upward of 2% in a zinc ore to be highly objectionable; and in fact, in view of their smelting methods and equipment at that time, it was objectionable. The attempts to smelt even the comparatively clean ore from Creede in 1899 were disastrous.

A combination of circumstances, however, caused the possibility of obtaining an ore supply from Colorado and elsewhere in the far West to be kept in mind. There was at about this time a concerted effort on the part of the miners of the Joplin district to raise the price for ore. Their mines were in fact unable to furnish the supply required except at an enhanced

* The utilization of the zinc resources of the far West was early considered. In 1885, Eugene and Alfred Cowles patented an electric furnace for the reduction of zinc ore, and I believe they had in mind the treatment of mixed ore from New Mexico. H. C. Rudge built Belgian furnaces at Denver, Colo., in 1888 and actually smelted a small quantity of ore from Leadville, but because of ignorance the venture proved a failure. Messrs. Ingalls, Argall and Wood, who have been associated in the investigation of the zinc resources of British Columbia, formulated extensive plans for zinc development in 1889, but these proved to be premature.

price. On the other hand, the price for spelter left the smelter an insufficient margin and he was keenly looking out for supplies of cheaper raw material. European smelters were in somewhat the same position.

The great deposits of mixed sulphide ore at Leadville, Colo., had been worked since about 1885 for the lead content of the low grade of ore, the zinc and most of the iron being thrown out upon the tailings pile. The concentrating mills were of the old conventional design, crushing the ore comparatively coarse and cleaning it chiefly by jigging, but there was no great profit in the operation and it was after a while abandoned. The success in the treatment of similar ore by fine crushing at Broken Hill, New South Wales, the invention of the Wilfley table in 1895 (affording a greatly improved means for cleaning fine ore), reductions in the cost of mining, etc., led to the erection of a type of mill more especially suited to the particular ore, and although these were designed especially for the production of galena concentrate, it was found that a fair grade of zinc concentrate could be made at the same time as a by-product. The advent of an enterprising broker, acquainted with the needs of the European zinc smelters, developed an export business, which in 1900 and in two or three years subsequent attained large proportions. The factors enabling this to be done were the low price at which the miners were willing to sell the ore, and the low freight rate which was obtained, via Galveston, to Swansea and Antwerp. At first, the ore produced assayed about 45% zinc, 12% iron and 6% lead; the average subsequently ran down to about 38% zinc, 17% iron, and 3% lead. The buyers paid a flat price for the ore, at first only $5 per 2000 lb., f. o. b. cars at Leadville, and shipped it to European ports at a cost of $9.50 per 2000 lb. The miners were well satisfied with this price because the ore was distinctly a by-product, and anything realized for it was so much gain. The increasing demand for zinc ore of any kind led, however, to competition for the Leadville ore, a gradual increase in the price for it, and the producers became firm in holding out for the best bargains.

The smelters of Kansas continued their experiments on the smelting of Colorado ores with many discouraging experiences, but after a few years they succeeded in treating them profitably and drove the European buyers out of the Colorado market. The increased price for spelter, the continuing shortage of ore, and the severe competition for what Joplin could supply forced these smelters more and more into the country west of the Rocky Mountains for their ore supply, and raised the prices for such ore, in which the smelters were aided by gradual improvements in their processes.

In order to illustrate the magnitude which the zinc industry west of the Rocky Mountains has attained, I may be permitted to quote from an article by myself, published in the Engineering and Mining Journal of May 12, 1906, and The Mineral Industry, Vol. XIV, as follows:

"Statistics of the production of zinc ore in Missouri and Kansas (Joplin district) and New Jersey are available for a long series of years. Up to a few years ago these were sufficient, inasmuch as nearly the whole spelter output of the United States was derived from those sources. In 1899 zinc ore from Colorado began to appear in the market, and during the last two or three years that ore, together with ore from other States and Territories west of the Rocky Mountains, and from British Columbia and Mexico, has been figuring largely in the market. It is, therefore, important to know definitely as to the production of these sources of ore supply. Such statistics respecting them as have previously been published are incomplete, and of doubtful accuracy.

"Statistics of zinc ore production indicate directly the magnitude of the mining industry, by showing the tonnage of ore produced and moved. In connèction with spelter production, however, it is necessary to examine them with knowledge of what they represent.

The ore production of the Joplin district is of two classes, viz., blende and calamine. The former averages about 58 per cent. zinc; in round numbers, two tons of this ore make one ton of spelter. The calamine of the district is entirely zinc silicate. It may be assumed as averaging a little better than 40 per cent. zinc, three tons of ore being required, roughly, to make one ton of spelter. The total production of zinc ore in the Joplin district in 1905 was 252,435 tons. No attempt was made to classify this as blende and calamine, but in recent years the output of the latter class of ore has amounted to 10,000-16,000 tons per annum, and it may be reasonably assumed that the production in 1905 was something between those figures.

"A small amount of calamine, both carbonate and silicate, is produced in southeastern Missouri, especially by the Valle mines. This ore goes chiefly to St. Louis, and amounts to 3000-6000 tons per annum.

"The zinc ore produced in the States and Territories west of the Rocky Mountains is both blende and calamine, the latter being chiefly zinc carbonate produced in Mexico and New Mexico. The production of Colorado, Utah, Idaho, Montana, and British Columbia is chiefly, if not entirely, blende. This ore varies generally in grade from 30 per cent. to 50 per cent. zinc. In a few cases, as at Creede, Colo., and the output of handsorted, lump ore of one mine in British Columbia, it exceeds 50 per cent., the Creede ore (mill-concentrate) in fact being almost as high in zinc as the average Joplin product, but although low in iron it is higher in lead than the Joplin ore. The average zinc content of the western ore, both blende and calamine, may be assumed at 38 per cent. From 3⅓ to 3 tons of this ore are required to produce one ton of spelter. This sulphide ore is comparatively high in iron and lead; some of it is very high in those elements. It is mostly produced as a concentrate from mixed sulphides, the lead product being shipped to the silver-lead smelters. The Iron Silver Mining Company, however, ships a good deal of hand-sorted lump ore from its Moyer mine, at Leadville.

"Wisconsin produces a blende concentrate, which after magnetic separation, is practically as high in zinc as the average Joplin ore, and when well prepared is comparatively low in iron and lead, the blende itself being only slightly ferruginous and the iron content of the marketed ore being chiefly intermixed marcasite. Wisconsin also produces carbonate ore, which is used at Mineral Point for the manufacture of zinc oxide.

"The large output of zinc ore in New Jersey is entirely from the Franklin mine of the New Jersey Zinc Company. It is the mixed franklinite-willemite, averaging about 20 per cent. in zinc, which is separated into one product (willemite) for spelter manufacture and another product (franklinite) for the manufacture of zinc oxide and spiegeleisen.

"Of the Western zinc-mining districts, the most important single district is Leadville, Colo. Other important single districts are Creede, Colo.; Magdalena, N. M.; Park City and Frisco, Utah; Monterey, and Las Plomosas (near San Sostenes, on the K. C., M. & O. Ry., Chihuahua) Mexico; and the Slocan, British Columbia. Outside of these districts, the zinc ore production west of the Rocky Mountains comes from many scattered localities. In New Mexico, besides Magdalena, Hanover is a small producer, and there are several other promising districts. In Montana, Butte is the principal source. In Idaho, the Wood River district is the most important, although some ore was obtained in 1905 from the Cœur d'Alene. In Utah, the Daly West Mining Company, of Park City, and the Horn Silver Mining Company of Frisco have large zinc resources; the former did not produce in 1905, but the latter shipped 8445 tons. In Colorado, besides Leadville and Creede, zinc ore is produced at Rico, and by many small mines in Clear Creek and Summit counties. In Mexico the Calera mine, of the State of Chihuahua, was a considerable shipper of mixed sulphide ore to Pueblo, Colo. Arizona and Nevada both figured as small producers in 1905. The ore of Magdalena, N. M., was shipped chiefly to Missouri, Kansas, and Wisconsin, for the manufacture of zinc oxide. Other western ores are shipped to Mineral Point, Wis., for the manufacture of zinc oxide.

PRODUCTION OF ZINC ORE IN THE UNITED STATES.

STATE.	1904.	1905.
Colorado	*a* 94,000	105,500
Idaho	Nil.	1,700
Kentucky	*d* 958	*d* 414
Missouri-Kansas	*b*273,238	258,500
Montana	Nil.	2,000
New Mexico	*c* 21,000	17,800
New Jersey	*d*280,029	*d*361,829
Utah	Nil.	9,265
Wisconsin	*c* 19,300	32,690
Other states *f*	*a* 4,500	*e* 6,000
Totals	693,025	795,698

a Estimated. *b*, Production of Joplin district, plus output of Southeastern Missouri, the latter as reported by the State mine inspector. *c*, According to H. F. Bain, "Contributions to Economic Geology,' 1904. *d*, Report of State Geologist. *e*, Partly estimated. *f*, Arizona Nevada, Arkansas, Illinois, Iowa, Tennessee and Virginia.

"The statistics of ore production by States are given in the foregoing table, which is based on the production of zinc ore in marketable form, from the standpoint of the zinc smelter, as shipped to and received by the zinc smelter. A certain quantity of low-grade ore, treated at Cañon City, Colo., for the manufacture of zinc-lead pigment, is enumerated separately.

IMPORTS OF ZINC ORE INTO THE UNITED STATES.

Source.	1904.	1905.
British Columbia	2,100	8,561
Mexico	?	*a* 32,164
Totals	?	40,725

a, The actual tonnage of ore imported was somewhat greater than this figure, but it included some mixed ore, which for statistical purposes has been reduced to the zinc ore equivalent.

"There was a small importation of zinc ore from Mexico in 1904, the business with that country having been inaugurated in that year, but statistics concerning it are unavailable.

"The total supply of zinc ore in 1905, so far as can be enumerated, was 795,698 tons from domestic sources and 40,725 from foreign, a total of 836,423 tons. The situation is clarified if the production of New Jersey be deducted leaving 474,594 tons, to be compared with the production of 190,294 tons of Western spelter. The Rocky Mountain ore used for the manufacture of spelter amounted to 160,000 tons.

"The total—795,698 tons—understates the actual production of ore to some extent, certain ore consumed for the manufacture of oxide being omitted. The deficiency is chiefly in the representation of the outputs of Wisconsin and "Other States."

"The United States Smelting Company, at Canon City, Colo., treated 33,000 tons of ore, averaging 22.7 per cent. zinc, and 8.8 per cent. lead, all of which, except 800 tons from Arizona, was obtained from Colorado. This ore, used for making zinc-lead pigment, has not been included in the above statements.

"The exportation of zinc ore from the United States in 1905 was 30,448 tons, against 35,333 tons in 1904. This was chiefly New Jersey willemite.

"I estimate that out of the 190,294 tons of Western spelter produced in 1905, about 124,000 tons was derived from ore mined in the Joplin district, about 12,294 tons from ore mined in Wisconsin, Kentucky, southeastern Missouri and Arkansas, and about 54,000 tons from ore mined west of the Rocky Mountains, including British Columbia and Mexico. The quantity of spelter originating in the far West certainly shows a remarkable growth for an industry that is only five years old. In 1904, about 128,000 tons of spelter was derived from Joplin ore. The total production of Western spelter having largely increased in 1905, the relative position of Joplin was materially reduced."

The beginning of the zinc industry in British Columbia was practically contemporaneous with its origin in Colorado, the first shipments having been made in 1899. These were chiefly from the Lucky Jim mine, near Sandon. A few tons came from the Bosun mine, near New Denver. The Lucky Jim mine, between June 30 and Nov. 30, 1899, shipped 1,728 tons of ore, assaying from 30 to 52% zinc. Most of this was ore assaying about 50% zinc, 1 to 2% lead, and less than 5 oz. silver per ton, but some of it was mixed ore assaying 30 to 35% zinc, 20 to 30% lead, and this was generally rather high in silver, containing 25 to 30 oz. per ton. The exportation of zinc ore proper from both the Lucky Jim and Bosun mines amounted to 1,600 tons, which went partly to Antwerp, and partly to Ellesmere-Port, on the Manchester Ship Canal, where the Smelting Corporation, Ltd., was exploiting the Fry process of smelting, an undertaking which subsequently met with disaster.

After this inauguration of zinc ore production there was a period of inactivity, which is not surprising inasmuch as in the United States the market for the impure ores had not yet become well established. American smelters were beginning, however, to feel the pinch in the conditions under which they were operating, and in the summer of 1901, the Lanyon Zinc Co., of Iola, Kan., which had received interesting samples of ore from British Columbia sent Thomas Jones thither to investigate the conditions and purchase ores for its account. Jones bought ore in 1902-1904, chiefly in 1903, purchasing about 4,000 tons in all. In 1903 and 1904 buyers for other smelters appeared in the market, but up to 1905 the shipments of ore were not large.

STATISTICS OF PRODUCTION.

The total shipments of zinc ore from the Slocan district, according to a statement obtained by Mr. A. C. Gardé from the Canadian Pacific Railway office at Nelson, B. C. are shown in the subjoined table, which include the ore passing over both the Canadian Pacific and Kaslo & Slocan roads.

NAME OF MINE	1902	1903	1904	1905
Bosun	580	681	....	....
Wakefield	35	181	151	....
Payne	667	610	1001	98
Whitewater	....	....	....	101
Ivanhoe	....	256	902	713
Hartney	....	21	....	....
Wellington	....	....	....	33
Bound Main	....	60	....	....
Lucky Jim	....	48	....	2462
Idaho-Alamo	....	....	30	60
Slocan Star	....	686	....	3978
Canadian Smelting Works	....	....	....	260
American Boy	....	21	....	129
Last Chance	....	....	....	22
Blue Bell	....	....	....	37
Total	1282	2564	2084	7893

The statistics for 1905 cover only the first 10 months of the year.

According to information furnished by Mr. Mackintosh, collector of the port, the shipments of ore passing through Kaslo, destined to the United States, from Jan. 1 to Oct. 31, 1905, were as follows:

NAME OF MINE.	TONS	GRADE OF ORE.			REMARKS.
Lucky Jim	2785	53% Zn.	0.5% Pb.	3 oz.Ag.	Lump ore.
Whitewater	101	47 "	3.0 "	33 "	" "
Wellington	32	50 "	3.0 "	55 "	" "
Last Chance	64	35 "	4.0 "	66 "	" "
American Boy	140	42 "	7.0 "	9 "	" "
		47 "	2.0 "	15 "	" "
Slocan Star	3566	32 "	3.0 "	32 "	Concentrate.
		35 "	2.5 "	45 "	"
Silver Bell	37	49 "	0.5 "	4 "	Lump ore
Total	4125				

The statements in the above table as to the grade of the ore should be accepted only as approximations, not as the settlement assays, but they are probably fairly indicative.

The statistics furnished by the Canadian Pacific Railway are short with respect to the shipments from the Lucky Jim mine (possibly because of some lots of ore not reported at the time of preparing the statement) since the actual shipments from that mine in 1905 were 3619 tons, besides 108 tons lost by the sinking of a barge. The mine also had on hand, Nov. 1, 1905, about 1500 tons of ore unsold. Part of this ore was produced in the fall of 1904, but it is impossible to make an accurate separation between the two years. Considering the whole as production in 1905, the total is 5227 tons, of which 744 tons were concentrate and the remainder hand-sorted lump ore. The concentrate was made between Nov. 9 and Dec. 9, 1904, and may be added to the production of that year.

Mr. G. O. Buchanan, Dominion Inspector of Lead Bounties, Kaslo, B. C., communicated to me, under date Dec. 15, 1905, the following statistics of zinc ore production in 1905, compiled from reports received from the owners and managers of the respective mines:

NAME OF MINE.	TONS.	ASSAY IN ZINC	REMARKS.
Slocan Star	4093	33.5%	Concentrate.
Lucky Jim	4600	54.0 "	Lump ore
" "	745	48.0 "	Concentrate
Ivanhoe	541	46.0 "	"
Ruth	1000	38.0 "	"
Jackson	1150	38.0 "	"
Bell	130	50.0 "	"
Idaho	61	37.4 "	"
All others (estimated)	1000	40 0 "	"
Total	13,320	42.0 "	

Under "all others" are included the production of the Vulture, Wellington, Last Chance, American Boy, Payne, Hartney, Monitor, Wakefield and Grey Copper mines.

The production of zinc ore in British Columbia in 1905, according to Mr. W. F. Robertson, provincial mineralogist, was 11,623 tons.

The discrepancies between the above statements are to be explained by difference in the bases of computation, which in certain cases is the actual production; in other cases, the actual shipments, which may be quite different, inasmuch as ore is sometimes held back at the mines. I consider that the shipments constitute the more reliable basis for statistics of this industry.

All of the zinc ore shipped from British Columbia in 1905 was destined to the United States. I obtained reports from all of the smelters who received this ore, which foot up to a total of 8561 tons.

From the foregoing data, it may be safely estimated that the shipments of zinc ore from British Columbia since the begining have been approximately as follows:

Year	Tons	Year	Tons
1899	1600	1903	2564
1900	none	1904	2828
1901	none	1905	8561
1902	1282		

The actual production in 1905 was possibly 3000 to 4000 tons more than the shipments, but this is largely ore held for treatment by magnetic separation for the enrichment of its grade in zinc, which will correspondingly reduce its weight.

CHARACTER OF THE ORE.

The zinc ore so far produced in British Columbia has been blende. No calamine * has been shipped, and the existence of any important supply of that class of ore has not been reported. The blende has been shipped partly as hand-sorted lump ore; partly as mill-concentrate. The former has been of the higher grade in zinc. The grade of the mill-concentrate is reduced by the intermixture of siderite (spathic iron ore) and pyrites (pyrite and pyrrhotite), which occur commonly with the blende of the Slocan and cannot be satisfactorily separated by ordinary mechanical concentration.

The blende of the Slocan is commonly of the variety known as "black jack," and is generally of bright luster. The black coloration of blende is not necessarily an indication of high content in combined iron; for example, the blende of Wisconsin, is sometimes black, but is low in combined iron.

* Metallurgically and commercially, "calamine" includes both the carbonates and both the silicates of zinc, although the U. S. Treasury Department has recently ruled to the contrary, which ruling is to be tested in the courts.

Blendes which are black in color and of brilliant luster are apt to be high in combined iron, but not always. The Slocan blendes do not appear to be high in combined iron as a general thing; nor are they as a rule high in cadmium.

A selected piece of blende from the Lucky Jim mine, black and of dull luster, was found to contain 59.8% zinc, and 4.56% iron.

A selected piece of blende from the Slocan Star mine, black and shining, was found to contain 60.75% zinc, and 4.45% iron.

These samples represented practically clean mineral, analysis showing no insoluble silicious matter. Neither specimen contained as much as a trace of copper, lead, or manganese. Nor did either specimen contain any considerable amount of cadmium, although they showed distinct traces of that element, possibly 0.1 to 0.2%.

The above analyses were made by Mr. W. George Waring, of Webb City, Mo. They indicate a ratio of zinc to iron in the pure blende of approximately 100: 7.5.

The following analyses of Slocan zinc ores were communicated by Mr. George Huston, editor of the Sandon Mining Standard, Sandon, B.C.

SLOCAN STAR MINE.

	1	2
Sulphur	29.84%	*a*
Zinc	56.70	43.40
Iron	6.23	*a*
Lead	0.14	1.00
Lime (CaO)	trace	*a*
Gold	"	*a*
Silver	2.80 oz.	159 oz.

a. Not determined.

RUTH MINE.

	1	2
Sulphur	28.03%	25.11%
Zinc	52.25	39.42
Iron	7.70	13.80
Lead	0.16	3.65
Lime (CaO)	0.12	trace
Gold	trace	"
Silver	3.40 oz.	9.40 oz.

1. Hand picked ore. 2. Concentrate.

LUCKY JIM MINE.

	1	2	3	4
Sulphur	30.65%	31.06%	31.03%	31.37%
Zinc	60.80	61.10	59.60	53.78
Iron	3.84	3.73	3.73	7.75
Lead	1.75	1.65	2.50	0.61
Lime	0.10	none	0.15	1.00
Silver	2.00 oz.	2.00 oz.	2.70 oz.	3.70 oz.

1. Hand picked ore. 2. Hand picked ore. 3. Hand picked ore. 4. Average ore.

AMERICAN BOY MINE.

	1	2	3	4	5	6
Sulphur	26.94%	26.55%	26.36%	23.68%	23.38%	23.86%
Zinc	49.40	49.00	48.70	44.90	44.50	45.25
Iron	6.10	6.20	6.10	5.96	5.86	5.98
Lead	3.45	3.15	3.80	2.20	2.90	2.45
Lime	0.22	0.30	1.32	1.67	1.67	1.46
Silver	11.80 oz.	12.60 oz.	12.80 oz.	6.70 oz.	6.50 oz.	6.86 oz.

OTHER MINES.

	a WELLINGTON.	*b* WAKEFIELD.	*c* IVANHOE.	*d* WHITEWATER.	*e* WHITEWATER.
Zinc	51.20%	48.00%	38.00%	58.00%	45.40%
Iron	0.07	5.00	......	......	......
Lead	6.40	3.90	5.00	1.00	7.50
Cadmium	0.50	0.29	......	......	......
Antimony	1.08	0.19	......	......	......
Arsenic	trace.	......	......	......	......
Silver	155 oz.	32 oz.	45 oz.	10 oz.	667 oz.

a. Crude ore; assay made by Karl Sander, Prayon, Belgium. *b.* Concentrate. *c.* Concentrate. *d.* Average of crude ore. *e.* Special.

The samples showing the remarkably high silver values are hardly to be considered as typical ores.

A carload (20 tons) of ore from the Bosun mine, assayed by Johnson & Son, London, showed:

Zn	46.25%	Fe	6.82%	Mg	0.33%
Si	8.25	Mn	1.08	Ca. O. etc.	8.45
Pb	1.02	As	0.30		
Cu	0.45	S	25.80		
Ag	0.25	Al	1.00		
Total					100.00

This ore contained 83 oz. silver and 0.5 oz. gold per ton.

A sample of a large lot of ore from the Payne mill, near Sandon, assayed by Ledoux & Co., of New York, Dec. 14, 1903, showed

Zinc	51.40%	Tin	0.17%	
Lead	2.36	Cadmium	0.46	
Iron	6.04	Arsenic	0.04	
Silica	4.66	Silver	9.40	oz. per 2000 lb.
Sulphur	21.84	Gold	0.07	" " " "

The presence of tin in this ore is noteworthy, although it is not greatly to be marvelled at, in view of the close association of the veins of the Slocan with granitic rocks, which are especially kindly to the formation of tin minerals. Other analyses of the Slocan zinc ores do not show tin, but this is probably because that element has not been particularly looked for and small percentages may easily escape observation in the ordinary commercial assays. I have been informed by Mr. Jules Labarthe that traces of tin occur in the base bullion (work-lead) produced at the Trail smeltery, B.C., and it is to be remarked that galena ores from the Slocan, obtained from the same veins as the zinc ore are extensively smelted at that works.*

The occurrence of tin in zinc ore is unusual, but not extraordinary. Crystals of cassiterite have been isolated in the blende mined at Freiberg, Saxony, and the spelter produced there has shown an appreciable percentage of tin, while tin has been detected in some of the old brands of spelter made in New Jersey.

It will be remarked that in many of the analyses of Slocan zinc ores the percentage of sulphur is insufficient to satisfy the zinc, iron, and lead. This is explained by the occurrence of carbonates in the ore, especially spathic iron, which does not, however, appear to contain zinc; at least not in any of the samples examined by the Commission. This subject, together with other points bearing upon the mineralogy and composition of the ores, is discussed at more length by Mr. Argall in a section of his report, to which reference should be made.

The analyses of many Slocan zinc ores show that they contain few objectionable impurities, and none of them in excessively large quantity. The percentage of iron is moderate and the percentage of lead is low. As a general thing, the ores are rather low in cadmium. Some of them contain a little manganese, but not much. The percentage of lime is very small. Fluorspar is entirely absent. Arsenic and antimony are present in some of the ores. The coarser portions of the solid zinc ores can be hand-sorted up to a tenor of 50% zinc. By combined wet and magnetic concentration ore assaying as high as 55% zinc can be produced, and 50% zinc ought to be a fair standard of practice with regard to a large class of the ores.

MARKET FOR ORES.

The zinc ore which has been heretofore produced in British Columbia has been marketed chiefly in the United States, the smelters at Pueblo, Colo., and at several points in Kansas having been the principal buyers. A comparatively small quantity of ore has been exported to Europe. Since about the end of November, 1905, a smelter at Frank, Alberta, has been in the market for these ores. There are, therefore, three markets open to the ores of British Columbia, viz.: (1) the American; (2) the European; (3) the Canadian, which, however, is still in a tentative condition. With respect to these markets widely different conditions obtain.

American Smelters.

The smelters of the United States are divided into two groups, viz.: (1) those which make high grade spelter from special ores, this group in-

* Analyses of Trail base bullion given by A. G. Betts in *Transactions*, American Institute of Mining Engineers, XXXIV, 182, show from 0.0118 to 0.0474 per cent. of tin

cluding the New Jersey Zinc Co. and the Bertha Mineral Co. producing at Bethlehem and Palmerton, Penn., and at Pulaski, Va., spelter from the remarkably pure ore of New Jersey; and (2) those which make spelter from Western ores. The Western spelter up to about 1901 was made almost entirely from ore mined in Wisconsin and the Joplin district of Missouri-Kansas, and was marketed chiefly under the name of "prime Western." Certain smelters, purchasing selected ores made superior brands. Since 1901, the quantity of ore from west of the Rocky Mountains has attained large proportions, and the "prime Western spelter" now produced is chiefly derived from that class of ore. Many of the smelters separate the first draw of metal, which being distilled at the lowest temperature of the operation, is lower in certain impurities (especially lead) than the second and third draws, and market it as "special." The major part of the Western spelter is produced, however, as ordinary brands, and the prices for ore are based on the prices of good ordinary brands of spelter. In this respect the same condition exists in Europe.

The producers of Western spelter in the United States are divided into two principal groups, viz.: (1) those in the Illinois coal field; and (2) those in the Kansas natural gas field. Outside of these groups are the Grasselli Chemical Co., which receives zinc ore at Cleveland, Ohio, and after roasting it there for the manufacture of sulphuric acid out of the roast-gases, ships the burned ore to Clarksburg, W.Va., for smelting; and the United States Zinc Co. (a subsidiary company of the American Smelting and Refining Co.) at Pueblo, Colo. The Grasselli Chemical Co. has not yet done any business in British Columbia.

The United States Zinc Co. makes a speciality of the recovery of silver and lead contained in zinc ore, obtains its entire supply of ore from the Rocky Mountains region of the United States, Mexico and British Columbia, and has been one of the largest purchasers of ore in the Slocan. The Illinois smelters have not yet figured to any considerable extent in the far Western market. The Kansas smelters have been important factors therein, and it is they. together with the Pueblo works, who have established the conditions under which the trade has been heretofore carried on.

The smelters now operating in the United States are the following:

Grasselli Chemical Co	Cleveland, Ohio.
Grasselli Chemical Co	Clarksburg, W. Va.
Matthiessen & Hegeler Zinc Co	Lasalle, Ill.
Illinois Zinc Co.	Peru, Ill.
Sandoval Zinc Co.	Sandoval, Ill.
Mineral Point Zinc Co. (1)	Mineral Point, Wis.
Mineral Point Zinc Co.	North Chicago, Ill.
Mineral Point Zinc Co.	Depue, Ill. (building).
Hegeler Bros	Danville, Ill. (building).
Edgar Zinc Co.	St. Louis, Mo.
Edgar Zinc Co.	Cherryvale, Kan.
Lanyon Zinc Co.	Iola, Kan. (3 works).
United Zinc and Chemical Co.	Iola, Kan.
Cockerill Zinc Co.	Iola, Kan. (2 works).
Cockerill Zinc Co.	Altoona, Kan.

Cockerill Zinc Co	Pittsburg, Kan.
Cockerill Zinc Co	Nevada, Mo.
Cockerill Zinc Co	Rich Hill, Mo.
Granby Mining and Smelting Co	Neodesha, Kan.
United States Zinc Co	Pueblo, Colo.
Caney Zinc Co	Caney, Kan.
New Jersey Zinc Co. (2)	Bethlehem, Penn.
New Jersey Zinc Co. (2)	Palmerton, Penn.
Bertha Mineral Co	Pulaski, Va.
Ozark Zinc Oxide Co. (1)	Joplin. Mo.
Ozark Zinc Oxide Co. (1)	Coffeyville, Kan.

1 Produces zinc oxide. 2 Produces both spelter and zinc oxide.

European Smelters.

The smelters of Europe who are chiefly in the market for American and Canadian ores are those of Wales, receiving their supplies through the port of Swansea; and those of Holland, Belgium, and the west of Germany, receiving their ore supplies through the ports of Hamburg, Rotterdam and Antwerp, but chiefly through Antwerp. Other European smelters purchase these ores, but to far less extent than those mentioned above, who have been for many years dependent upon foreign mines for nearly their entire supply of raw material. The system of receiving these supplies has therefore become well organized through long experience.

This organization has made Antwerp the favorite port of entry. There are at that port convenient docks for unloading and trans-shipping the ores; agencies for the sampling of the ores as received; brokerage houses for the consummation of transactions in custom lots; and public assay offices for the determinative work that may be required. Antwerp is, moreover, conveniently situated with respect to the principal smelters of Holland, Belgium and the Rhine, has excellent railway connections, and also is the terminus of an extensive canal system, by which cheap transportation to the works can be secured.

The smelters operating in Great Britain, Belgium, Rheinland-Westphalia, France and Holland are the following:

GREAT BRITAIN.

Brunner, Mond & Co	Winnington.
Central Metal and Smelting Co., Ltd	Glasgow.
Dillwyn & Co	Swansea (Llansamlet).
English Crown Spelter Co	Swansea (Port Tennant).
Williams, Foster & Co.	Swansea.
Pascoe Grenfell & Sons	Swansea.
Swansea Vale Spelter Co	Swansea (Llansamlet).
Villiers Spelter Co	Swansea (Llansamlet).
Vivian & Sons	Morriston, Swansea.
John Lysaght, Ltd	Netham, near Bristol.
Dynevor Spelter Co	Dynevor, near Neath.
H. Kenyon & Co	Warrington.

BELGIUM.

Société Anon. de la Vieille Montagne	Valentin-Cocq, Hollogne-aux Pierres.
Société Anon. de la Vieille Montagne	Angleur (Chenée).
Société Anon. de la Vieille Montagne	Flône, Hermalle-sous-Huy.
G. Dumont et Fréres	Sart-de-Seilles, Seilles.
Soc. Anon. de la Nouvelle Montagne	Engis.

Soc. Anon. Austro-Belge	Corphalie lez Huy.
Soc. Anon. metallurgique de Prayon	Prayon, à Forêt.
Soc. Anon. metallurgique de Boom	Boom.
L. de Laminne	Antheit.
Soc. Anon. d'Escombrera-Bleyberg	Bleyberg, à Montzen.
Soc. Anon. des metaux d'Overpelt	Overpelt, near Neerpelt.
Soc. Anon. des fonderies de Biache St. Waast	Ougrée.

RHEINLAND AND WESTPHALIA.

Société Anon. de la Vieille Montagne	Berge-Borbeck.
Rheinisch-Nassauischen Akt. Gesellschaft	Eschweiler.
Rheinisch-Nassauischen Akt. Gesellschaft	Stolberg.
Akt. Gesell. f. Bergbau, Blei. u. Zinkhüttenbetrieb zu Stolberg u. in Westphalen	Dortmund. Stolberg.
Markisch-Westfälischen Bergwerksverein	Letmathe.
Akt. Gesell. f. Zink Industrie vormals W. Grillo	Hamborn-Neumühl.
Aktiengesellschaft Berzelius	Bergisch-Gladbach.

FRANCE.

Société Anon. de la Vieille Montagne	Viviez (Aveyron).
Compagnie Royale Asturienne des Mines	Auby (Nord).
Usine à Zinc de St. Amand	St. Amand lez Eaux (Nord).
Société des Mines de Malfidano	Noyelles-Godault.

HOLLAND.

Société de la Campine	Budel.

The European vs. the American Market.

The conditions governing the sale of zinc ore in the European and American markets are widely different, and these have a decidedly important effect on the producer of ore, who is selling in them. These differences are due partly to custom, partly to natural conditions.

The European smelters receive the major portion of their ore supplies by water; they come in large lots; and are contracted for delivery a long time ahead.

The American smelters receive their entire ore supply by rail. The ore came formerly in small lots, purchased from week to week, and even at the present time a large tonnage is received in that way. This is because the Joplin and Wisconsin ore is produced by a multiplicity of small mines, which individually make only a small output and according to old custom adjust their accounts weekly. Since the appearance in the market of the ore supplies from west of the Rocky Mountains, afforded in many cases by important mining companies, capable of supplying a large and regular output, the receipt and settlement of ore on a contract basis has greatly increased, and the smelters desire to transact business in that way; in this respect approaching the European practice.

Either the European or the American smelter will buy small lots of ore. Both, however, aim so far as possible to insure the supply required for their works as far ahead as possible, and will make better terms, in other words will figure more closely, on a contract for the regular delivery of a certain tonnage than they will on occasional small lots. This refers, of course, to normal conditions. It may happen that a great shortage in ore supply, as recently in the Joplin district, will induce the smelters to pay fancy prices for whatever is offered. However, while the American smelters are tending to the same policy as the European with respect to regularity of ore supply, the ore business of the two will always be distinguished by the direct receipt on the one hand of the ore in railway cars of 25 to 30 tons capacity; and on the other hand by vessel, involving several trans-shipments. This inevitably has an important bearing upon the calculations of the exporter. In the case of the large exporters of Greece, Italy, Spain and Australia, the ore is dispatched by the ship-load, and is delivered on board ship after a comparatively short railway carriage. In the case of large shipments from the western part of the United States, or from British Columbia, the ore must necessarily be carried over a long line of railway, involving the lapse of a good deal of time, during which the cars of a lot are likely to become separated, causing trouble in making up the cargo. If the ore be shipped in small lots it may have to be sacked, adding considerably to the expense.

The exportation of large quantities of zinc ore from Franklin, N.J., and Leadville, Colo., the latter being low grade material, the tonnage in both cases being large, has been successfully carried on. In both cases, however, the business has been done under special contracts. The exportation of zinc ore from the Joplin district, several times attempted, has not proved a success. The settlements have developed a multiplicity of charges and losses, important in the aggregate, which have added heavily to the returning-charge specified in the contract.

An example of these results is afforded by the experience of the Miners' Association of Missouri and Kansas, which early in 1903 exported 1,022 tons (of 2,000 lb.) of ore to Belgium, the prime object being to reduce an accumulation of ore then existing in the district. The ore was shipped in two lots of about 500 tons each. It was shipped in bulk by rail to Galveston, and thence to Antwerp by steamer. The freight rate was $5.80 per ton. The loss of ore in transit amounted to 2.4% of the weight at Joplin, and was equivalent to $0.72 per ton. Drattage, which is the European custom of allowing 3 kg. overweight per 1,000 kg. amounted to 0.29%, or $0.087 per ton. Reception charges, cables, interest, insurance charges, etc., amounted to $0.259 per ton. The total of these items, necessary to deliver the ore c.i.f. Antwerp, omitting the freight was as follows:

Loss in transit	$0.720	per ton.
Draftage	0.087	"
Incidentals and interest	0.259	"
Total	$1.066	"

In the exportation of an ore concentrate from west of the Rocky Mountains, worth approximately $25 per 2,000 lb. f.o.b. cars, an allowance of $1 @ $1.25 per ton must be added to the freight rate to destination, and the brokerage also must be added if the business be done through a broker, in order to obtain the basis c.i.f. European port, where the value of the ore is computed after deducting the returning charge of the smelter. Or, calculating back from the gross value of the ore at the works for which purchased, there is first to be deducted the smelter's charge (this including freight, etc. from port of entry), and then the commissions, insurance and freight from point of original shipment, plus an allowance (as indicated above) for extras, chief among which is the loss in transit. This loss may be greatly reduced by sacking the ore, but that adds to the expense more than the loss may amount to in bulk shipment. In the exportation of 4,300 tons of galena concentrate from the St. Eugene mine in the year ended Sept. 30, 1904, the cost of sacks and sacking amounted to $1.25 per ton.

In shipping ore to American smelters, the latter commonly make the price to the shipper on the basis of f.o.b. cars. The smelter himself has to stand the loss in transit, which comes out of his margin. The miner can ship in car load lots of 25 to 30 tons, and the business is in all respects far simpler than with the European smelters. Before going into the relative advantage of the terms which can be offered by American and European smelters, it is necessary to consider the questions of tariff conditions and the determination of the value of zinc ore in general, and especially the silver and lead contents of certain zinc ores.

Tariff Conditions.

In all of the European countries which import zinc ore, it is admitted free of duty.

In the United States, where the tariffs on imports are now established by the Dingley Act, there was in 1905 a dispute between the American mining interest and the smelting interest with respect to the status of zinc ore from foreign countries, which originated through the importations from British Columbia, but was precipitated by the importations on a much larger scale from Mexico, which in 1905 assumed large proportions. The importations from British Columbia were exclusively blende; those from Mexico were partly blende, but were largely carbonate ores, carbonate ores being commercially and metallurgically included with the silicates of zinc under the name of "calamine." Mineralogically, however, the name "calamine" is employed by the best authority (Dana) as the species-name of the hydrous silicate of zinc.

In the Dingley Tariff, calamine is specifically put on the free list. Blende is not mentioned, nor are zinc ores in any form, except calamine, as above stated. The case of these ores came therefore under some general clause of the Act. Unfortunately there were two or three conflicting clauses of that character.

It was decided by an order of the Secretary of the Treasury, Feb. 10, 1906, that "calamine" in the tariff referred only to the hydrous silicate of zinc, and that sulphide and carbonate ores were dutiable at the rate of 20 per cent. *ad valorem*. This decision is not final, inasmuch as the smelters intend to take the matter to the Courts. Pending the action of the latter, any zinc ore imported into the United States from British Columbia is obliged to pay the duty of 20 per cent., the British Columbia ore being exclusively blende. The portion of this report dealing with the marketing of ore was written before the Treasury decision of Feb. 10, 1906 was rendered, when American smelters were the chief buyers of Canadian ore, wherefore the fullness with which the conditions of the American market are discussed. The analysis will, however, continue to be useful in determining the possibilities of the Canadian and the European markets.

THE VALUATION OF ZINC ORES.

The value of a zinc ore depends chiefly upon its tenor in zinc and objectionable impurities; especially iron, manganese and lime, which increase the corrosion of the retorts; and lead, arsenic and antimony, which contaminate the spelter. The value of the ore is also affected by its character, whether oxidized or sulphide, or a mixture of both; the sulphide ore must be roasted, but yields a diminished weight for the subsequent treatment, which is the more expensive part of the process; the oxidized ore escapes preliminary treatment, unless it be carbonate, but suffers no diminution in weight. The preliminary treatment of ores which are mixtures of sulphides and oxides is often troublesome. The value of an ore is, moreover, affected by its physical character. Lump ore is subject to an additional expense for crushing; fine slimes are more expensive and troublesome to roast than coarser concentrates. Some ores roast and distil easily; others with more difficulty. All these factors are given consideration by the zinc smelter. The chemical composition of the ore is, however, the most important factor in determining its value.

In determining the treatment charge on the ore purchased, the smelter starts with the cost of smelting a ton of the ore of average composition, that is to say the mixture on which he proposes to operate his furnaces. It is aimed to have all the furnaces on the same charge, for various reasons. To this smelting charge he adds the profit that he ought to make to obtain a proper interest on his investment, allowing for the necessary amortization of his outlay in plant.

The further addition of the freight on the ore to his works, and on the spelter product to its market, with allowances for the cost of buying the ore and selling the spelter, gives the returning charge which he must make against the ore in buying it on the basis of f.o.b. cars at the mine or mill where produced.

The ores purchased will be of various kinds. Few will correspond exactly with the ore which it is aimed to charge into the furnaces. Some will be higher in zinc; others will be lower. Some will be too high in iron; others too high in lime. The very desirable ores can perhaps be purchased only at a small margin. The deficiency must then be made up from the price of the less desirable ore. Inasmuch as the various kinds of ore may not be bought contemporaneously, the smelter effects this balancing in price by arbitrary additions to the returning charge on certain kinds of ore according to the percentage of objectionable impurities contained. It may be necessary under certain contingencies to put a less advantageous charge into his furnaces, when the cost of smelting will be directly increased and the percentage of metal extraction decreased, by greater destruction of retorts, higher zinc tenor of the residues, or some other factors which have a powerful influence on the ledger.

It is the custom of European smelters to pay for ores according to a sliding scale, which combines three elements, viz., the price of spelter and zinc content of the ore, which are variables, and the returning charge per ton of ore which is fixed. This sliding scale is embodied in a convenient formula, from which any seller of ore can readily compute the value, the returning charge having previously been agreed upon in the contract.

American smelters compute the value of ore in practically the same way, but in purchasing custom lots of ore they make usually a direct bid of so much per ton, and in purchasing ore on contract they frequently employ an involved sliding scale, which is generally equitable, though less simple than the European.

The smelters of Belgium, Holland, France and the West of Germany employ generally the formula $0.95 \, P \left(\frac{T-8}{100}\right) - R$ in which P is the price of spelter (good, ordinary brands), at London, T the units of zinc in the ore, and R the returning charge per ton of 1,000 kg. This formula gives the value of the ore per ton of 1,000 kg. The value is given in pounds sterling, marks, francs, or dollars according as P, the value of spelter at London, is reckoned in pounds sterling, marks, francs or dollars.

This formula is scientific and fair. The freights and cost of smelting per ton of ore are substantially the same irrespective of the grade of the ore (within certain limits). The returning charge R is therefore constant, as it should be. The percentage of metal extracted falls off as the grade of the ore falls off, because the losses in smelting are to a large extent constants, i.e. a ton of certain ore may contain 1,000 lb. of zinc and a ton of another ore 800 lb. of zinc, but the loss of metal in smelting will be approximately 125 lb. in each case. The formula takes account of this by the uniform deduction of eight units from the zinc content of the ore as shown by assay. If the ore assay 60 units of zinc, the payment is for 52 units, or 86 $\frac{2}{3}$% of the zinc in the ore. An ore assaying 50 units, with eight units deduction, returns 84%; and an ore assaying 40 units, with eight units deduction, returns 80%. These percentages are not very much below the actual metallurgical extractions. In being a little under the actual extraction it tends to increase slightly the smelter's margin as the value of spelter rises, as does also the discount of 5% from the London price, but this is equitable.

Sometimes in the case of a desirable ore, the smelter will propose to buy the ore with a deduction of only seven units from the assay, instead of eight, as usual. The percentage of zinc paid for, on different grades of ore, becomes then as follows:—

60 units : 88.33%
50 units : 86.00%
40 units : 82.50%

Sometimes, if it be anticipated that lower grades of ore than contemplated in the returning charge, named in the contract, will be offered, it is provided that the returning charge be increased with decrease in the grade of the ore. For example, it may be provided that for each unit of zinc below 50, the returning charge shall be raised 2.5 francs per 1,000 kg. of ore, which is equivalent to 43.86 cents per 2,000 lb. If therefore the returning charge were named at $11.50 per 2,000 lb. for ore assaying 50 to 55% zinc, with increase of 43.86 cents per unit below 50, the charge on ore assaying 48% zinc would be $12.38 per 2,000 lb.

The formula $0.95P\left(\frac{T-8}{100}\right)-R$ works out as follows in the case of an ore assaying 48% zinc, the London price of spelter being assumed at £28 per ton of 2,240 lb., and the returning charge £2 12s. 6d. per ton of ore.

$$0.95\,P = 0.95 \times £28 = £26.6.$$
$$\left(\frac{T-8}{100}\right) = 0.48 - 0.08 = 0.40.$$
$$R = £2\ 12s.\ 6d. = £2.625.$$
$$£26.6 \times 0.40 - £2.625 = £8.015 = \$38.95.$$

The value per 2,240 lb. of ore is $38.95. Dividing that result by 1.12 gives the value per 2,000 lb. of ore, thus $38.95 ÷ 1.12 = $34.78.

The same result is obtained when the price of spelter and the returning charge are converted into terms of dollars and cents per 2,000 lb. Thus, £28 per 2,240 lb. × 6.075 cents per pound = $121.50 per 2,000 lb.; 0.95% of this is $115.43; the returning charge of £2 12s. 6d. per 2,240 lb. is equivalent to $11.39 per 2,000 lb. The value of the ore per 2,000 lb. is consequently:

$$\$115.43 \times 0.40 - \$11.39 = \$34.78.$$

Another method, which is very convenient, is to convert the expression T–8 into pounds and multiply by the price of spelter per pound. Thus, and

$$T-8 = 40;\ \text{and}\ 40 \times 20 = 800\ \text{lbs.}$$
$$0.95\,(6.075 \times 800) - \$11.39 = \$34.78.$$

The returning charges which have been made by European smelters on American, Australian and Canadian ores during the last two or three years have ranged from $11.40 to $13.16 per 2,000 lb., these figures corresponding to 53 @ 60 marks or 65 @ 75 francs per 1,000 kg.

The terms offered by European smelters include various provisions, which must be given consideration. The basis of settlement is always c.i.f. at some specified port, usually Antwerp. A percentage of the value of the consignment, say 75% or 80%, may be drawn upon remittance of the bill of lading; the remainder upon arrival of the consignment, and determination of its assay value. In connection with the latter, the zinc must be determined by the Schaffner method (sodium sulphide titration). Differences of one unit or less between the assays of the buyer and seller are split. Differences in excess of one unit are umpired. The umpire must be an European chemist (specified in the contract). His result is accepted as final if it fall between those of the buyer and seller. If it fall outside, the original assay which is nearest to the umpire's is taken. The party whose assay is furthest from the umpire's pays the expense of the latter. These conditions as to umpire assays are the same as obtain in the settlement for ores in Canada and the United States.

The price for spelter is the average for good ordinary brands at London, as reported by the *Public Ledger*, during the month of the arrival of the shipment. In the case of contracts covering the output of a year or more, it is sometimes provided that the settlement basis shall be the average price of spelter for the year. Provisional settlements are made on the monthly averages, but at the end of the year a final computation is made and the difference between it and the sum of the monthly settlements is debited or credited as required.

As previously remarked, the settlement for ores in Europe is based on the London price for spelter. In the United States it is based on the St. Louis price, or the New York price, there being generally a constant difference between the quotations at St. Louis and New York, corresponding to the difference between the freight rate from Kansas works to St. Louis, and to New York. This difference was formerly about 20 cents per 100 lb; now, and in recent years, it has been about 15 cents. In statistical investigations it is necessary to refer to the New York price, rather than to the St. Louis, because there are authoritative statistics going farther back. Contracts are commonly based on the quotations for spelter in the *Engineering and Mining Journal.*

The difference between settlement on the London and New York markets is a very important consideration to the Canadian exporter of zinc ore. The range of the two markets is shown in a table which is presented elsewhere in this report. It appears, therefrom, that although the London price has been at certain times in excess of the New York price, the average over a long series of years has been lower than the average at either New York or St. Louis. Whenever the London price has been higher, the United States has exported spelter, and this movement of metal has reduced the London price below the American level, or has forced the American price to rise above the European level. There is no shortage of ore supply available to European smelters. On the contrary not only the ore supply, but also the smelting capacity, are going to be immensely increased by the operations of strong companies owning enormous reserves of ore at Broken Hill, New South Wales, which are already developing their plans. So long therefore as the American Government maintains a duty on the importation of spelter, it is to be anticipated that the London price for spelter will continue to average a little lower, in the long run, than the American price.

The western smelters of the United States, in their ore contracts, employ sliding scales of the following character:

(1) Basis of settlement:—Ore delivered at smelting points in Kansas. When spelter is at 6c. per lb. at St. Louis, pay $25 per ton (2,000 lb.) for ore containing 47% zinc, plus 75c. per unit for zinc in excess of 47 units, less 75c. per unit for zinc below 47 units; with 35% of the variation in spelter.

(2) Basis of settlement:—Ore delivered f.o.b. cars at mines. When spelter is at 6c. per lb. at St. Louis, pay $24.50 per ton (2,000 lb.) for ore containing 53% zinc, plus $1 per unit for zinc in excess of 53 units, less $1 per unit for zinc below 53 units; with 42.5% of the variation in spelter.

The above were the terms of contracts for Canadian ores in effect during 1905.

The inferiority of a contract so expressed, insofar as a clear understanding is concerned, in comparison with the European contracts is quite obvious. Taking the latter of the two for example, the meaning is that when spelter is worth 6c. per lb. at St. Louis, ore assaying 53% zinc is worth $24.50 f.o.b. mine. Ore assaying 54% zinc is worth $25.50; ore assaying 52% zinc is worth $23.50. A variation in the price of spelter changes the basis price to the extent of 42½% of the variation per ton of spelter, i.e. if the price for spelter be 5.7c, per lb. at St. Louis, a reduction of 0.3c. per lb.=($6 per ton) from 6c., or $6×0.425=$2.55, is made from the basis price of the ore, wherefore the value of ore containing 53% zinc becomes $24.50—$2.55=$21.95. Ore with 54% zinc is then worth $22.95; ore with 52% zinc is worth $20.95.

The equity of such a contract can be determined by a consideration of its operation under different conditions. Converting $24.50: 6c.: 53% zinc into an expression of P $(\frac{T-8}{100})$—R, it appears that R=$29.50, this including all freights, smelting charges, profits, etc. Tabulating the results for different grades of ore computed by P $(\frac{T-8}{100})$—$29.50 in the column B, and the results computed according to the sliding scale of the contract under the column A, the comparison appears as follows when spelter is at 6c. per lb. St. Louis:

% ZINC IN ORE.	A	B
45	$16.50	$14.90
50	21.50	20.90
55	26.50	26.90
60	31.50	32.90

When spelter is at 5c. St. Louis, the comparison becomes as follows:

% Zinc in Ore.	A	B
45.	$ 8.00	$ 7.50
50.	13.00	12.50
55.	18.00	17.50
60.	23.00	22.50

If spelter should rise to 7c. the comparison would be as follows:

% Zinc in Ore.	A	B
45.	$25.00	$22.30
50.	30.00	29.30
55.	35.00	36.30
60.	40.00	43.30

Aside from the question of the returning charge there is nothing in this sliding scale which is unfavorable to the miner. At 5c. for spelter the price received for ore of different grades is approximately the same in each case as the St. Louis value of the spelter extractable from the ore less a uniform returning charge. At 6c. for spelter the margin to the smelter diminishes with decrease in the grade of the ore below 53%; with increase above 53% the margin would increase; these conditions would be emphasized if the price of spelter should attain 7c. In the particular contract the ore was likely to run a little under 53% rather than over it, wherefore less risk was taken by the miner than by the smelter upon fluctuations in the grade of the ore and in the market for spelter.

A comparison between the terms offered by American and European smelters may be made in the case of this ore. With spelter at 6c., St. Louis, the American smelter paid $24.50 per 2,000 lb. f.o.b. mine for ore containing 53% zinc The European smelter would pay $39.64 c.i.f. Antwerp. Deducting the freight to Antwerp, $13 per 2,000 lb., and an allowance of $1 for incidentals and loss, as previously discussed, the price f.o.b. mine would be $25.64. However, at the time of the contract the London price for spelter was considerably lower than the New York price, wherefore the American price figured out the better.

The average price of spelter at St. Louis in 1905 was 5.730c.; at London, 5.518c. On these prices the value of the ore taken as an example would have been $22.20 per 2,000 lb. f.o.b. mine under the American contract and $21.50 under the European contract.

It will appear from what has previously been said, together with the data as to cost of smelting, etc., which will be found in a subsequent section of this report, that as between the American and European smelters, previous to the recent Treasury decision the former were in a position to command the market for zinc ore in British Columbia if they chose. The price that they would offer for ores was limited by the best terms which European smelters could make, but otherwise was governed by competition among themselves, their requirements for ore, and to a large extent by the condition of the Joplin ore market, which is still their most important source of supply, although since 1901 its relative importance has been diminshing, and probably will continue to do so. However, the Joplin district will doubtless continue to have a dominating influence on the general ore market for a long while to come, and its prices will to a large extent be used as a basis. It is rather important to consider how prices for other ore work out on this basis.

The average price of ore at Joplin over a long period of years is given in another section of this report. The average for 1905 was $47.40 per 2,000 lb. in the bins of the mines. Assuming an average cost of 40c. per ton for carting and loading the ore on board cars, the cost per ton delivered at Kansas smelting points would be about $48.75. This is for ore assaying 60% zinc, and low in iron, say not more than 2% An old rule of the district is to deduct $1 per unit of zinc under 60 and $1 per unit of iron over 2. The ore that has previously been taken as an example contains about 7% iron. The value of such an ore delivered at the smelting works on the Joplin basis would be therefore $48.75—$12=$36.75 on the basis of 5.73c. for spelter at St. Louis. Reckoning $10 railway freight from the Slocan, 20c. freight on moisture, 30c. loss in handling, 30c. duty on lead contents and 20c. for incidentals, the value of $36.75 per ton delivered at smelting works would correspond to $25.75 f.o.b. mine, against the value of $22.20 according to the terms of the contract. This difference is not on the face unfair, inasmuch as the computation is based on a price of $47.40 for ore assaying 60% zinc at Joplin, but that price leaves only a small margin to the smelter, in fact scarcely a living margin, when spelter is at 5.73c. St. Louis. The deduction of $1 per unit for iron in excess of two units however, is, too much, being more than the increased cost of smelting such an ore, and the smelter could have afforded to have paid a better price than $22.20 for the ore and would have done so if he had been compelled thereto by competition.

It will readily be seen from the foregoing discussion what effect the American tariff of 20% *ad valorem* will have on the position of the American smelters in the market, and what will be the position of the European smelters and the single Canadian smelter. The possible yield to the producer of ore is affected by the silver and lead content of his ore, but anything that may be realized from them does not alter the competitive position of the buyers. The general subject of silver and lead extraction, which is of great interest to the producers, is more appropriately discussed further on in this report.

COST OF SMELTING.

The cost of smelting ores is but imperfectly understood by those who have not had experience in the business. This does not refer only to the smelting of zinc ore, but also to the smelting of other kinds of ore. I may go further and say that it is but imperfectly understood even by many metallurgists who are practically engaged in the business. The operating cost per ton of ore smelted is one thing; this is comparatively easy to determine, but it tells only part of the story. The total cost of smelting is quite another thing, and is not so easy to determine; this includes not only the direct operating cost, but also the necessary allowances for administration of the business, interest on the money invested, and amortization of the capital laid down in plant construction. The capitalists who invest their money in a metallurgical plant not only expect to receive a proper interest on it, but also expect to preserve the principal intact. This subject has been discussed so thoroughly in industrial economics that it is unnecessary to go into it here to any further extent than will outline the fundamental principles.

There have been instances where plants have been erected for the specific purpose of working up a definite supply of material, only that and nothing more. Assuming it to be contemplated that such a purpose will be consummated in two years of time, it is obviously necessary to charge each year's operation with one half of the cost of the plant. Ordinarily, however, the calculation for amortization is not so simple, because the life of the plant can not be so definitely foretold, but even under the most favorable prospects as to continuance of the supplies of raw material, and the conditions which govern the operation of the works and the marketing of its products, there are considerations as to the life of the plant, both in part and as a whole. Some of its apparatus, in spite of the most liberal outlay for repairs and renewals, will wear out and become useless, often in a comparatively few years . Other parts may perhaps be kept in excellent condition for many years, but may become unprofitable through advances in the art, whereby the competition of more modern and superior methods or machines may render the old ones practically useless. This consideration, which has aptly been referred to as "depreciation due to the advance in the state of the art" applies not only to particular machines and methods, but also to the plant as a whole. Anyone who will look around in the industry with which he is most familiar, and will observe the number of plants of no more than 10 years' construction, which have become idle and out of date will appreciate the importance of this industrial calculation.

It is a common practice among engineers in considering the probable standing of a new metallurgical project to reckon an amortization period of 10 years, i.e. the first cost of the plant must be reimbursed in that time, not necessarily with view to the distribution of the original investment among the subscribers, but rather with view to the maintenance of the value of the plant to the possible extent of a complete replacement at the end of the time reckoned. Prudence seldom permits the estimate of a longer amortization period than 10 years. In some cases it is unsafe even to reckon so long a time as that.

The assumption of an amortization period of 10 years implies that 10% of the first cost of the plant must be added annually to all the cost of direct operation, and moreover, a certain interest charge must be added in order to to arrive at the actual cost of production. Reduced to the basis of a ton of ore the relative magnitude of these charges depends upon the cost of plant per ton of annual capacity. For example, the cost of a first class copper smelting plant of 1,000 tons daily capacity, or say 350,000 tons per annum is about \$600,000, which is about \$1.70 per ton of annual capacity. The cost of a silver-lead smelting plant of the same capacity is about \$800,000, which is about \$2.30 per ton of annual capacity. The cost of smelting a ton of copper ore in such a plant is roughly about \$2.50; the cost of smelting a ton of silver-lead ore is roughly about \$3.50*. In the former case 17c. per ton should be added to the smelting cost for amortization; in the latter case 23 cents per ton; in both cases the amortization charges are only about 7% of the direct operating cost. In zinc smelting the conditions are quite different, not only in this, but also in other respects.

In the first place the design of zinc smelting plants is by no means so well standardized as are those of copper and silver-lead smelting plants The latter conform to certain well established lines. The new plants which are built now-a-days differ but little, one from another. Zinc smelteries on the other hand, even of modern construction, show wide differences (although there is a strongly growing tendency toward uniformity) and there is an equally wide diversity in their costs. Plants erected in the United States within the last five years have cost from \$7 to upward of \$20 per ton of annual capacity; the former for a plant using natural gas as fuel; the latter for a plant using coal (gas firing and regenerative furnaces). These figures are, however, extremes. A plant costing only \$7 per ton will require immediate additions, which will raise the figure; while a plant costing \$20 per ton implies an unnecessary outlay of money.

The first cost of a zinc smelting plant has a very important bearing upon its operating cost and the percentage of metal extraction which it will effect. Representing by M the value of the metal extractable from a ton of ore of given zinc content, by O the cost of the ore, by C the direct operating cost per ton of ore, by P the first cost of the plant per ton of ore and by X the net profit per ton of ore.

$$X = M - (O + C + 0.1\ P + 0.06\ P)$$

in which 0.1 P is the allowance for amortization and 0.06 P for interest on the first cost of the plant.

Practically, X should be the larger, the larger is P up to a certain point (because M will be larger and C will be smaller) in order to justify the increase in P. This is indeed the result of experience. In comparing natural gas and coal gas plants, however, of which the first cost is widely different, the above formula would show very much in favor of the lower value of

* These figures are based on the ton of charge smelted, including ore and flux. In good practice in silver-lead smelting the ore is about 80% and the flux about 20% of the charge. These figures, together with that as to the cost of silver-lead smelting are representative of American conditions. The conditions in British Columbia are widely different.

P if amortization were reckoned at the same percentage. But no one would reckon on so low an amortization as 10% in the case of a plant whose existence would be dependent upon a fuel supply so uncertain and precarious as natural gas.

The cost of a modern zinc smeltery in Belgium or Rheinland, for the treatment of blende, i.e. with full roasting capacity, is about $14.50 per ton (of 2,000 lb.). Such a plant will be equipped with gas producers and Rhenish regenerative furnaces. A similar plant in Canada or the United States would probably cost $17.50 to $18 per ton. The plant at Pueblo, Colo.' which practically duplicates the one at Overpelt, Belgium, cost more than $18 per ton, but there were various misfortunes in connection with its construction. American engineers would not, however, duplicate the details of European construction, nor would they follow the European lines as to machinery and various features of the design of the works. They would erect a plant, of at least equal efficiency in all respects and superiority in some important matters, which would cost about $16 per ton. A corresponding plant for natural gas fuel would cost about $10 per ton. The difference is accounted for chiefly by the higher cost of the regenerative furnaces, with their accessories, including gas producers, extra boiler capacity, etc., and the means for storing and handling the large quantity of coal required. The ordinary plant in the Kansas gas field costs $7 to $8 per ton. These estimates are for plants of capacity for 25,000 tons of ore per annum, which is a convenient size. Smaller plants cost proportionately somewhat higher.

The comparative cost of smelting a ton (2,000 lb.) of zinc blende ore under different conditions as existing at present is approximately as follows:

ITEM.	A.	B.	C.	D.	E.
Labor	$4.25	$4.52	$4.80	$3.20	$3.24
Fuel	.80	1.69	.80	3.75	3.32
Reduction material	.80	.80	.80	.67	.55
Clay	.44	.40	.44	.50	.72
Other supplies	.20	.20	.20	.20	.20
Repairs and renewals	.48	.75	.47	.65	.64
Administration	1.00	1.10	1.00	.70	.60
Total	7.97	9.46	8.51	9.67	9.27
Say	8.00	9.50	8.50	9.67	9.25

A. Well designed natural gas plant in Kansas, equipped with mechanical roasting furnaces. Cost of gas reckoned at 2c. per 1000 cu. ft.

B. Estimate for well designed coal plant in Illinois. Gas firing. Mechanical roasting furnaces. Rhenish regenerative distillation furnaces. Cost of fuel reckoned at 75c. per ton, 2.25 tons required per ton of ore.

C. Ordinary natural gas plant in Kansas, equipped with hand roasting furnaces.

D. Estimate for well designed coal plant in Rheinland, Germany. Hand roasting furnaces. Gas firing. Rhenish regenerative distillation furnaces. Cost of fuel reckoned at $2.50 per 2000 lb., 1.5 tons required per ton of ore.

E. Estimate for well designed coal plant in Rheinland, Germany. Hand roasting furnaces. Gas firing. Rhenish regenerative distillation furnaces. Cost of fuel reckoned at $2.60 per ton, 1.28 tons required per ton of ore.

Zinc ore is being smelted to-day, in spite of increased cost for both labor and gas, more cheaply in Kansas than it can be in Europe. Even with coal, it can be smelted in the United States for approximately the same cost as in Europe, although this is hardly being done at the present time. European smelters have the advantage of very much cheaper labor than the American, and unlike the experience in some other arts, low-priced European smelter-men has in zinc smelting practically as high an efficiency as the high priced American. The difference in the labor cost per ton would be greater were it not that the American smelters economize in labor through the use of mechanical roasting furnaces and other mechanical devices, wherever they can be installed, to a greater extent than the European. The European smelter has, moreover, an advantage in various other items in the cost of smelting, but these are all offset by the cheaper cost of fuel in America, although the inferior character of the coal in the chief zinc smelting districts requires the consumption of a greater quantity of it per ton of ore.

The final element in the valuation of zinc ores, namely the percentage of metal extraction, is difficult to generalize, inasmuch as it depends greatly upon the character of the ore-mixture that is smelted and the metallurgical practice. An extraction of 90% of the zinc in the ore is sometimes effected, but this is above the average. On a good grade of ore an extraction of 88% would be a very fair result, and with a poor grade of ore and in inferior plants the actual result will be a good deal below that figure. In the early treatment of Colorado ore in Kansas, the extraction was only about 72%. The retorts in use would not stand the high temperature necessary to distil off the last of the zinc, and residues assaying high in zinc were discharged from the furnace. Improved practice, including the manufacture of a superior retort, has improved that result, but the extraction from these ores is still inferior to that which is yielded by the clean ores of the Joplin district.

Estimating an extraction of 88% in an European plant and 84% in an American plant, the profit to the smelter in purchasing in British Columbia an ore assaying 53% zinc, with spelter at 5.73c. St. Louis and 5.52c. London, may be calculated as follows:

DR.	KANSAS.	BELGIUM.
Ore cost per 2000 lb.	$22.20	$21.50
Freight	10.00	13.88
Loss and incidentals	1.00	1.25
Smelting	8.50	9.25
Amortization	1.50	1.40
Interest	.54	.84
Freight on spelter	1.14	.76
Selling expense	.50	.50
Total	45.38	49.32
Cr.	$51.00	$51.50
Spelter	890 lb. at $5.73	933 lb. at $5.52

In the above estimates, no allowance is made for the recovery of by-products (sulphuric acid, silver, and lead) which the European smelters generally recover to more or less extent; but which American smelters recover only in a few instances.

The metallurgy of zinc is frequently spoken of as a backward art, the smelting process still being expensive as compared with that of lead and copper ores, while the proportionate extraction of metal is greatly inferior. This idea rests, howeyer, on false standards of comparison. The zinc smelter, as a rule, deals with ore which has already been enriched to a high degree, so that his practice is comparable to that of the smelter of galena concentrate, or of black tin ore, rather than to that of the silver-lead or copper smelter, who has to treat a very large quantity of ore for a comparatively small production of metal. In other words, while the copper smelter makes commonly a concentration of 20:1 and even 50:1, the zinc smelter makes a concentration of only 2:1 or 3:1. In handling the less quantity of raw material it is generally permissible to utilize wheelbarrow and shovel to a greater extent; but if the cost of the process be referred to the basis of the crude ore raised from the mine, the smelting expense may not appear unduly heavy.

VALUE OF ARGENTIFEROUS BLENDE.

The addition which is lent to the value of zinc ore by a silver content is a matter of both interest and importance in connection with the ores of British Columbia, inasmuch as they are generally silver-bearing; occasionally silver-bearing to a high degree. The highly argentiferous ore, however, is much smaller in proportion than the ore which is comparatively low in silver. In some cases the ores run very high in silver, but such occurrences exist elsewhere, especially where tetrahedrite (grey copper ore) or similar silver minerals are associated with the blende. These high silver-zinc ores, however, are commonly considered not as material for the zinc smelters, but for the silver-lead smelters, to whom they are disposed of.

In the treatment of a zinc ore by the usual smelting process, a content of lead and silver (and also gold) in the ore can be recovered if it be desired. As to whether the recovery will be attempted, or not, is chiefly a question of extractions and costs. The first step in the metallurgical treatment of zinc blende is a roasting of the ore down to about 1% sulphur. In this process the loss of zinc is small, being seldom more than 2% in bad practice, and ordinarily in good practice considerably less than 1%. The losses of both lead and silver, however, are heavy; the silver loss is likely to be 10 to 12% in the roasting of a 10 oz. ore, while the lead loss is fully 10%.

In distilling the roasted ore there is a further loss of lead by volatilization; some of that which goes over into the condenser escapes; some is condensed along with the spelter, which it contaminates, necessitating a refining in order to make the spelter marketable. The behavior of the lead in this respect depends largely upon the manner in which the distilla tion is conducted. If the retorts be run at a relatively low temperature, comparatively little lead will go over into the spelter, but the extraction of zinc will be low. On the other hand, if the retorts be run at relatively high temperatrue, in order to insure the best possible extraction of zinc, a comparatively large proportion of lead will go over into the spelter. In the eyes of the zinc smelter the extraction of zinc is the prime consideration; the other metals are merely by-products; the zinc is the chief product and the process is directed mainly to its extraction. In any case, the loss of silver (and gold) during the distillation is insignificant.

After the distillation of the zinc has been completed, the residue raked out of the retort is a cinder composed of unburned coal mixed with the gangue of the ore, the latter being fused to a more or less degree, according to its ingredients. This cinder contains the major portion of the lead (to some extent as metallic globules, to some extent as silicate and other compounds) and practically all of the silver of the roasted ore. The cinder also contains a varying percentage of zinc, ordinarily from 4 to 7%. The proportional weight of the cinder with respect to the roasted ore charged into the retorts varies with the character of the ore and the practice in its distillation. In the distillation of ordinary ore, assaying 45 to 50% zinc, the residue generally amounts to 60 to 70% of the weight of the ore charged, say 66⅔% as a mean. Under certain conditions, however, as at Cockle Creek, New South Wales, it may amount to 100%.

The residue having been discharged from the distillation furnace, and assuming that it contain sufficient lead and silver to make their recovery worth while, there are three alternatives open, viz.:

(1) The entire quantity of residue may be passed on to the lead smelter.

(2) The residue may be treated as a crude ore, by crushing and jigging, the concentrate being passed on to the lead smelter.

(3) The residue may be crushed and jigged only for removal of the unburned coal, all the remainder being passed on to the lead smelter, while the coal recovered may be utilized in a variety of ways.

The second method is the one commonly employed in Belgium and the West of Germany. The first method has long been used at Freiberg, Saxony where the treatment of rich silver-zinc ores has been a specialty, although conducted on only a small scale; it has also been practised in the United States. The third method is practised both in Europe and in the United States.

The results achieved by the three methods are quite different, and their relative efficiency depends largely upon local conditions. For the purpose of illustration, comparison may be made of the results that might be expected under certain conditions from an ore containing 10 oz. silver per 2,000 lb. and 5% lead, or 100 lb. of lead per ton. In roasting there will be a loss of 1 oz. of silver and 10 lb. of lead. The roasted ore will weigh approximately 85% of the weight of the raw ore. In distillation there will be no important loss of silver; but the 90 lb. of lead contained in the roasted ore may be expected to suffer a loss of $16\frac{2}{3}\%$, or 15 lb. Assuming the residue to be about 65% of the weight of the roasted ore, or 1,100 lb., that quantity of material will contain 9 oz. of silver and 75 lb. of lead, corresponding to 16.4 oz. silver per ton and 6.82% lead.

If this residue were passed on directly to the lead-smelting furnaces, a recovery of 95% of the silver and 90% of the lead would be expected. The ultimate extraction would be, therefore, 8.55 oz. silver and 67.5 lb. of lead. Reckoning the silver at 60c. per ounce and lead at 4.5c. per pound, and allowing 1c. per lb. for freight and refining charges on the bullion, the gross value of the products would be $7.49.

The net value in this case can not easily be generalized, because the cost of smelting the residue will vary widely according to local conditions. It is not an easily smelted material. It contains a large proportion of fines and is bulky both for that reason and the large amount of unburned coal which it comprises. This coal does not reduce the percentage of coke required on the charge, and indeed is a drawback to economical smelting. The retort residue will never be self-fluxing, the conditions of zinc smelting precluding any such possibility, and suitable ores and fluxes must be available to make up a proper smelting charge.

At Iola, Kan., the smelting of retort residue containing 10 to 15 oz. silver 0.03 oz. gold, and about 6 per cent. lead was abandoned as unprofitable after a long trial, although 91 per cent. of the silver, 92.5 per cent. of the gold and 92 per cent. of the lead were recovered. It may be assumed therefore that the cost of smelting was considerably in excess of $10 per ton.

On the other hand, in a very favorable location, such as Pueblo, Colo., where there are silver-lead smelters treating an immense tonnage of ore, with which the zinc residue can be advantageously mixed in comparatively small proportion, the cost of smelting it may be quite moderate, say $5 per ton.

From the gross value of the by-products, $7.49 as previously computed, a deduction of $2.75 to $5.50 must therefore be made, leaving $4.75 to $2 as the net value per ton of crude ore, the former figure being attainable only under very favorable circumstances.

If instead of smelting the residue directly it were concentrated mechanically by crushing and jigging, there would be in the first place a further loss of both silver and lead in that process. This would amount probably to one-fifth of the lead and one-third of the silver, wherefore the concentrate would contain 6 oz. of silver and 60 lb. of lead, assuming 5 tons of residue to be reduced to one ton of concentrate. The final products from the smelting of the concentrate would be 5.7 oz. of silver, worth $3.42 and 54 lb. of lead, worth $1.89, a total value of $5.31 from which must be deducted the cost of concentrating 0.55 ton of residue, which may be taken as 55c., and the cost of smelting 0.11 ton of concentrate, which may be taken as 55c., giving a total deduction of $1.10 and a net yield of $4.21.

If on the other hand the residue were washed simply to remove unburned coal, of which it may be estimated that 20% can be separated and returned in one way or another to the smelting process, there would be no important loss of lead or silver and the weight of the product to be smelted would be reduced to 880 lb. The value of the lead and silver ultimately recovered would be therefore $7.49, less 55c. for concentrating and $2.20 for smelting or a net result of $4.74 per ton of crude ore.

These computations are sufficient to show how under certain conditions it is cheaper to smelt the entire residue, or all of it except the coal which can be washed out; and how under other conditions it is cheaper to concentrate the residue. On an ore of grade assumed, when conditions are favorable to cheap smelting (silver-lead) as at Pueblo, Colo., it is the more profitable to smelt the entire residue, or the entire residue minus the excess coal. On the other hand, where the conditions are unfavorable (as at Iola, Kan.) this process proved unprofitable on the same grade and kind of ore as is smelted successfully at Pueblo. It will be readily perceived that the necessity of shipping the residue to some other point for treatment, involving a freight and treatment charge rising to $10 or more, per ton will reduce the net value of the silver and lead extractable from a ton of ore to a low figure.

Thus it will be clear that in the treatment of a zinc ore for silver and lead recovery, either the losses are high or the cost is high. This is the reason why the zinc smelter can not afford to pay for silver and lead more than a comparatively small proportion of their assay value. In general, the European smelter is not disposed to pay more than 60% of the assay value of silver in the ore, and that rate only on the silver in excess of 150 grams (about 5 oz.) per 1,000 kg. If the price of silver be 50c. per oz., or 8.5 centimes per gram, the rate paid will be about 5 centimes; with silver at 55c. per oz., or about 10 centimes per gram, the rate paid will be about 6 centimes. An ore assaying 10 oz. per ton would therefore suffer in the first place a deduction of 5 oz., and the remaining 5 oz. would be paid for at the rate of 36c. per ounce (silver being 60c. per ounce in the market), making the silver value of the ore to the miner $1.80 per ton.

The European smelter will not pay for lead at all unless it be in excess of 8%. Lead in excess of 8% is paid for at 50% of the price quoted in London.

It should be remarked, however, that the price paid for silver and lead can not fairly be considered by itself, but must be viewed with regard to the terms under which the zinc in the ore is paid for. For example, a smelter may make a comparatively low returning charge on the ore, contemplating that a concession in that respect will be offset by a profit on silver for which he pays nothing to the seller of the ore.

A contract made for a certain ore from British Columbia in 1904 at the low returning charge of 50 s. per long ton, equivalent to $10.85 per short ton, stipulated that the ore should contain 10 oz. of silver per ton. Each ounce of silver short of that guarantee reduced the value of the ore 25c. per ton; on the other hand, the vendor received a similar credit for silver in excess of 10 oz.

There are comparatively few American smelters who are equipped, or are in a position to extract silver and lead from the zinc ore which they treat. There will doubtless be, however, an extension of interest in this direction. In such cases as the smelters are especially in the market for this class of ore, the practice in purchasing is similar to the European. Thus, one contract for Canadian ore provided for the payment of silver in excess of 8 oz. per ton at the rate of 50% of the New York quotation for silver. On an ore assaying 20 oz. silver per ton, 40% zinc, and over 5% lead, a price of $9.50 per ton f.o.b., Sandon, was made in August, 1905; with a zinc variation of 75c. per unit above or below 40; and a silver variation of 40 c. per oz. above or below 20 oz.

WET PROCESSES OF ZINC EXTRACTION.

A vast amount of ingenuity, energy, and money has been spent, since 1860, upon the development of hydrometallurgical processes of zinc extraction, the idea being to bring the zinc into solution (usually as sulphate); separate the solution from the residue (which will contain the silver and lead of the ore, together with the gangue); precipitate the zinc by suitable reagents and pass the product on to the smelting furnaces, or precipitate it in metallic form by electrolysis; and pass the insoluble residue on to the lead furnaces.

Electrometallurgical processes of this character are hopeless, save under certain especially favorable conditions, primarily because of the high amount of power that is inevitably required to electrolyze solutions of any of the salts of zinc. Vast sums of money have been expended in proving by the erection of large works for the operation of such processes the fundamental principles, which could have been worked out in the office, and the practical behavior of the scheme of working, which could have been determined in the testing-laboratory for one per cent. of the money. The conditions under which processes of this character may prove workable are: (1) in the case of processes affording a useful anode reaction, the application at places where advantage can profitably be taken of it ; and (2) the availability of very cheap water power.

With reference to the first condition, an example is afforded by the use of the Höpfner process at the works of Brunner, Mond & Co., at Winnington, near Chester, England. In this process a solution of zinc chloride is electrolyzed. Zinc of great purity is deposited at the cathode and chlorine is set free at the anode. The chlorine is conveyed into chambers wherein it is utilized in the manufacture of bleaching powder. The electrolysis of zinc chloride furnishes a product at the anode, which in connection with a chemical works can be profitably used, and the value of this by-product offsets what would be a deficit in the deposition of zinc alone.

With reference to the second condition, extremely cheap power (for instance power as cheap as $5 per h.p. per annum) might enable the profitable deposition of zinc without regard to the anode reaction.

In general, projects for the development or exploitation of wet electrometallurgical processes of zinc extraction are to be looked at askance. *

Among the numerous hydrometallurgical processes involving chemical precipitation of the zinc, which have been patented, many have been tried in practice, but only the Parnell process has served for the treatment of any considerable quantity of ore. This was used in Swansea from 1880 to 1883. The ore was roasted and leached with dilute sulphuric acid. The solution of zinc sulphate thus obtained was boiled down to a paste, mixed with a proportion of zinc sulphide, and heated in a muffle, whereby desulphurization was effected and a product obtained which could be easily smelted for zinc. The process was abandoned because of failure of ore supply, and and possibly because it did not prove sufficiently profitable under the conditions at that time. It has never been revived; but in view of the mechanical improvements of which it is capable, it is worthy of reconsideration, and on the whole offers the best prospect of any process of this class.

One of the most serious difficulties in wet processes of zinc extraction pertains to a point which has commonly been overlooked by inventors. They say in their specifications that a solution of the zinc (usually as the sulphate) is to be obtained, and then proceed to outline the following steps, but it is precisely the dissolving of the zinc which it is difficult to effect satisfactorily. If a pure zinc blende, like that of Joplin, be roasted to oxide, the latter dissolves quickly and completely in dilute sulphuric acid. But in the case of a mixed sulphide ore, containing iron, it is found that after roasting the extraction of zinc is very imperfect, even with strong acid. It appears that during the roasting a compound of zinc and iron is formed, referred to more or less correctly as zinc ferrate, which is insoluble in acid. In numerous experiments on ore from Leadville, Colo., the highest extraction of zinc that I obtained was 80%, and ordinarily the extractions were less than 70%. I have been informed, however, by Mr. C. E. Dewey, that he has recently been obtaining extractions approximating 80% on a regular working scale.

This result, namely a zinc extraction of only 70 to 80% has been confirmed by the results of so many metallurgists, working on a great variety of ores, that it is a conservative conclusion that nothing better is to be expected of a ferruginous zinc ore unless some way be found to modify the roasting process to prevent the formation of insoluble zinc compounds, which has not yet been done. Assuming an extraction of 75%, it is obvious that processes of this class start out with a great handicap, inasmuch as aside from further losses that may be suffered in the process, the final product

* It is to be noted that a distinction is made between electrometallurgical processes, by which we have come to understand those in which current is employed for electrolysis, and electrothermic processes, in which current is employed for heating.

is subject to loss in smelting, wherefore the ultimate extraction of zinc in the form of spelter would be but little, if any, more than 66⅔% of what was originally present in the ore.

The American Zinc and Chemical Co. has been recently treating ore at Denver, Colo., by the process invented by C. E. Dewey, which is of this class. The works were partially destroyed by fire last autumn, but it is said that they will be rebuilt.

In the Dewey process the ore is roasted under such conditions as best promote the formation of zinc sulphate. The roasted ore is digested with water into which sulphurous acid gas from the roasting furnace is introduced, dissolving the zinc as sulphate and bisulphite, the latter oxidizing to sulphate. The solution of sulphate is drawn off, evaporated to dryness, and the dry sulphate is calcined to oxide, which is sold to zinc smelters.

The terms offered by the company purchasing ore for this process are indicated by a settlement made on a lot of ore from the Hartney mine, British Columbia, June 24, 1903.

The net weight of the ore was 40,333 lb. = 20,167 tons. Its assay, metal contents and gross value per ton (according to then existing rates of the silver-lead smelters at Denver) were as follows:

ELEMENT.	ASSAY.	CONTENTS	VALUE.	TOTAL.
Gold	0.045 oz.	0.045 oz.	$19 per oz.	$0.86
Silver	56.33 "	56.33 "	50c. per oz.	28.16
Lead	24.75 %	495.00 lb.	32c. per unit.	7.92
Zinc	24.00 "	480.00 "	—50c. per unit.	
Iron	8.40 "	168.00 "	—10c. per unit.	36.94
Silica	15.50 "	310.00 "	excess silica	
Sulphur	18.30 "	376.00 "		

The smelting charge was $3.50 per ton, plus 71c. for 7.1 units of excess silica, and $7 for 14 units of excess zinc, making a total of $11.21, and giving the ore a net value of $25.73 per ton.

After extraction of the zinc the residue weighed 26,890 lb., or 66⅔% of the original ore. The assay and contents of this residue were as follows:

ELEMENT.	ASSAY.	CONTENTS	VALUE.	TOTAL.
Gold	0.065 oz.	0.43 oz.	$19 per oz.	$0.82
Silver	84.00 "	56.00 "	50c. per oz.	28.00
Lead	36.00 %	480.00 "	32c. per unit.	7.68
Zinc	7.20 "	96.00 "	....	
Iron	12.60 "	168.00 "	—10c. per unit.	36.50
Silica	23.30 "	310.00 "	excess silica.	
Sulphur	5.00 "	67.00 "	—25c. per unit.	
			excess sulphur.	

The smelting charge per ton of residue was $1, plus $1.07 for excess of silica, and 50c. for excess of sulphur, making a total of $2.57. On two-thirds of a ton of residue this would be $1.72, making the net value of the ore on this account $34.78 per ton of the original raw material.

.The extraction of zinc was 80%, giving 384 lb. (in the form of oxide) which was credited at 2.5c. per lb., or $9.60, against which there was a debit of $10 for the cost of the extraction process.

This result shows a gain of about $9 per ton of ore to the miner. It is to be remarked, however, that this comparison is made with straight silver-lead smelting. Ore of the character specified would be unmarketable with the zinc smelter unless he were prepared to separate it by some mechanical process. The treatment of this class of ore from that stand-point will be discussed further on in this report.

It is extremely doubtful if any hydrometallurgical or electrometallurgical process, even under the most favorable conditions, could show a commercial superiority over the present combination of standard methods in the treatment of such an ore as has been referred to. Starting with an ore containing 24% zinc and 6% lead, separated magnetically or electrostatically at a cost of $1 per ton, approximately 0.33 ton of zinc ore, containing 50% zinc, or nearly 69% of the zinc in the original ore, and 0.55 ton of lead-iron ore containing 9% lead, or approximately 82.5% of the lead in the original ore, are obtained. Carrying the computation forward to the final extraction of the metals, good practice should yield about 60% of the original zinc and 78% of the original lead at an aggregate cost not to exceed $7 per ton of crude ore, disregarding freights, interest, amortization of plant, etc. A complete analysis of this problem would be quite complicated, but the above figures will give a rough indication of the present state of the art.

THE ZINC MINES OF BRITISH COLUMBIA.

The mines of the West Kootenay and the East Kootenay are described in detail in the reports by Mr. Philip Argall and Mr. Alfred C. Gardé, published elsewhere in this volume. Those of Vancouver Island, the Coast, and certain prospects in the Boundary district are described by Dr. Alfred E. Barlow. Certain mines which are believed to possess zinc resources are further described in a series of reports by the gold commissioners of British Columbia. It is necessary for me only to summarize their reports and present certain observations made in my own examination of the mines of the Kootenays, which included most of the more important from the view-point of zinc production.

There are certain mines in the West Kootenay which are essentially zinc mines. The best examples of these are the Lucky Jim and the Blue Bell. In each case they were worked originally as silver-lead mines, but as such were probably unprofitable—at all events they were allowed to lie idle for many years after the original exploitation. The proportion of galena to the remainder of the ore was small, and the ore was of low grade in silver. The Lucky Jim has recently produced upward of 5,000 tons of zinc blende assaying 50 per cent. and more, in zinc, broken out to a considerable extent from solid bodies of the mineral and shipped in lump form without any culling. The Blue Bell also shows rich faces of blende ore, which although mostly of a concentrating grade will afford considerable that can be selected as comparatively high-grade lump ore by hand sorting.

The majority of the mines of the West Kootenay are, however, essentially silver-lead mines, in which zinc blende occurs as an accessory ore. In this respect they differ in no wise from many other mines in the Rocky Mountains from British Columbia to Mexico, in which zinc blende occurs in association with galena, pyrite and other argentiferous and auriferous minerals. The wide-spread and abundant occurrence of zinc in these ores is indicated by the slags made in silver-lead smelting in the United States, which average about 6 per cent zinc oxide, or approximately 5 per cent. metallic zinc. Estimating the smelting of 2,500,000 tons of ore per annum, which is considerably under the quantity now actually treated, and the product of 0.9 ton of slag per ton of ore, there is annually discarded in this form about 112,500 tons of zinc, which is the result after the zinc ore has been so far as possible culled out by the miner, or left behind in his stopes.

The increased demand for zinc ore during the last few years, which is by all means likely to continue, has made valuable as a by-product in many cases what was formerly an objectionable impurity, to be culled out so far as possible and thrown over the dump. This represents precisely the situation in connection with the majority of the mines of the Slocan. There are comparatively few which can be worked profitably as zinc mines; there are many which are worked for silver-lead ore, wherein the zinc ore will be a valuable by-product, as has already been thoroughly well demonstrated in the case of the Slocan Star and certain other mines. The yield of silver-lead ore will always be the dominating factor in the operation of these mines.

The silver-lead veins of the Slocan are extremely irregular. They are generally narrow and the pay-streaks are thin. Exceptions to this generalization are to be found in certain wide veins, like one at least on the South Fork of the Kaslo, and certain thick stopes of ore such as are to be seen in the Slocan Star mine. The general association of siderite with the ore is highly characteristic of the entire region; there are few of the silver-lead mines which do not show it to some degree. The tendency of the veins toward impoverishment in both galena and blende with depth is strongly marked, the proportion of siderite first increasing and finally the proportion of quartz. The ore shoots are rather short and the stretches of barren vein between them are rather long; the stoping area of the veins is therefore comparatively small and the cost of prospecting is correspondingly high. The silver-lead ore is generally of high grade in silver, which has made it possible to operate certain of the mines at the excellent percentage of profit that has been shown most brilliantly by the Payne and Slocan Star.

However, it has been the failure to recognize the true conditions, which has been the cause of many disasters in the district. Many properties have been developed in too ambitious a manner. Long cross-cut tunnels have been driven at large outlay of money, which has not been justified by the advantage to be gained. Mills of too large capacity have been built and attention in their design has been directed toward reduction of operating cost per ton of ore rather than toward securing the maximum percentage of the valuable minerals of the ore, which should have been the chief consideration in the concentration of these rich and difficult ores. This error in mill design appears to have been due to a mistaken following of the practice in the Cœur d'Alene district of Idaho, where the conditions as to ore deposits, character and grade of the ore are radically different from those which obtain in the Slocan. The large number of mills standing idle throughout the Slocan, in spite of the bounty on lead ore, is the best possible evidence of the mistakes of the past. In future attempts to reopen these mines with a view to augmenting the yield of argentiferous galena by the marketing of argentiferous blende as a by-product, it is important that profit be taken of previous experience. Operations must be inaugurated tentatively. Large outlays in dead-work must be avoided. Prospecting in the veins must be pushed boldly and must always be kept well ahead of stoping. Installation of plant, either mining or milling, must be carefully considered with respect to the probability of reimbursement of the cost of the plant. If there be only 100 tons of ore to be hoisted out of shaft it is both better engineering and better business to raise it by windlass than to buy a steam hoist, and this same principle obtains throughout mine operation.

Neither the average grade of the ore nor the cost of mining in the Slocan can be satisfactorily generalized. The samples taken by Mr. Argall and Mr. Gardé which are referred to in their reports and are summarized in the report of Mr. Henry Harris, indicate the general character of the ore, but it must not fail to be observed that they were taken from only partially developed showings of pay ore, in most cases from narrow streaks, and the grade of ore actually mined from them would be in all probability considerably lower than the assays of the samples. This would depend chiefly on the method of mining and the cleanness with which the ore would be separated underground.

Cost of Production and Productive Capacity.

The cost of mining per ton of material in the West and East Kootenays does not appear to vary greatly from the cost in the Cœur d'Alene district of Idaho, where the conditions are similar. In each district timber is abundant and obtainable at nearly equal cost. The mines are workable generally through adit levels, eliminating hoisting and pumping costs. Rates of wages are practically the same.

At Wardner, Idaho, miners are paid \$3.50 per day; shovelers, \$3; foremen, \$6 to \$7; shift bosses, \$4 to \$6; timbermen, \$3.75 to \$4; hoisting engineers, \$4; head blacksmiths, \$4.50; generally eight-hour shifts. The rate for board is \$1 per day.

At the Blue Bell mine, opposite Ainsworth, miners are paid \$3.25 to \$3.50; shovelers, \$3; smiths, \$4; all per shift of eight hours. The rates in the Slocan and in the East Kootenay are about the same. These figures will be found in detail in Mr. Argall's report.

As illustrative of the cost of mining in large veins at the present time the following table is presented:

ITEM.	A[1]	B[2]	C[3]
1. Miners and helpers	\$0.35	\$0.435	\$0.429
2. Shovelers and trammers	0.41	0.365	0.790
3. Drill sharpening and repairs	0.15	0.120	0.261
4. Compressed air	....	0.120	0.083
5. Maintenance of cars	0.03	0.090	0.038
6. Explosives	0.13	0.145	0.092
7. Timber	0.28	0.110	0.205
8. Timbermen	0.23	0.190	0.067
9. Hoisting	0.23	0.190	......
10. Pumping	....	0.035	......
11. Supplies, n. e. s. [4]	0.04	0.040	0.105
12. Supervision [5]	0.23	0.225	?
Total	2.07	2.065	2.070

1. Cripple Creek, Colo. 2. Centre Star mine, Rossland, B.C. 3. Bunker Hill & Sullivan, Cœur, d'Alene. 4. Including all supplies not elsewhere specified. 5. Including bosses, assaying and surveying.

It will be observed that the cost per ton, on the production of the large quantities of ore upon which these figures were based, comes out the same, but this is merely a coincidence, the advantages of one mine in certain branches of work being precisely offset by the disadvantages in others. Thus, the Bunker Hill & Sullivan, which is operated through a long adit level, has a high cost for tramming ore, but has no costs for hoisting and pumping, which appear in the statements of the two mines operated through shafts. Timber and timbering are always widely variable items in mining. In 1903, the Bunker Hill & Sullivan produced 288,713 tons of ore at a cost of \$1.633 per ton for mining and 11.5c. for general expense. The ore was removed from the mine by electric haulage through the Kellogg tunnel (12,000 ft. long) at a cost of 7c. per ton. In addition to the ore, 47,000 tons of waste was trammed.

There are few figures available as to the cost of mining in the Kootenays, from which any instructive analysis can be made. The St. Eugene is the only company which produces a large tonnage of ore and published an official report of the cost. The report of this company for the year ended Sept. 30, 1904, stated that during five months of operation 58,456 tons of ore were stoped and 50,456 were removed from the mine. The costs per ton figure out as follows:

ITEM.	TONS.	TOTAL COST.	COST PER TON.
Mine development	58,456	$24,913.50	$0.426
Stoping	"	85,772.57	1.468
Compressor operation	"	12,231.82	0.209
Shaft repairs	"	1,264.55	0.022
Total mining	58,456	124,182.53	2.125
General expense	50,456	18,276.84	0.236
Transportation	"	4,121.84	0.082
Milling	"	31,172.95	0.618

Charging the general expense, administration, assaying, surveying, taxes, insurance, etc,. to mining, the total cost of the latter was $2.49 per ton. Deducting the cost of development, which is a widely variable item, the direct cost of producing ore was about $2.06 per ton. This is considerably higher than in good practice in similar veins in the United States, but it is by no means an extravagant figure. The cost of milling also is considerably higher than the cost of similar work in the best practice in the United States, but it does not compare unfavorably with the results attained in many American mills.

The cost of mining 50,000 tons of ore in the Slocan Star mine in 1904-1905 was about $2.50 per ton; the cost of milling the same tonnage was about 41 cents per ton.

A mining cost of $2 @ $2.50 per ton of material may be assumed as a broad generalization of the conditions which obtain in the Slocan. Such a figure is obtainable in a vein of solid ore of fair stoping width, say 6 ft The cost of mining increases inversely as the proportion of pay ore to total vein area decreases, and directly as the occurrence of the ore bodies becomes small and irregular, all of which factors increase the amount of development work that must be done. When the pay streak is narrower than the minimum width of efficient stoping, say 3 ft. 6 in., the cost of production per ton of ore increases directly as the width of the pay streak decreases. If for example we should have a pay streak of solid blende 12 inches wide and 30 inches of quartz beside it, all of which would have to be removed in stoping, a square foot of vein would yield (assuming clean separation of the minerals to be possible, which however would never be entirely practicable) one cubic foot of blende weighing about 250 pounds and 2.5 cu. ft. of quartz weighing about 412.5 lbs., wherefore out of every ton of material broken there would be obtained three-eighths of a ton of blende, and if the cost of mining per ton of material were $3, the cost per ton of blende would be $8. If the blende assayed 60% zinc, the mining of a 12-inch streak would be equivalent to the mining of a 3.5 ft. streak assaying 22.5% zinc, but the solid streak would have the advantage of avoiding the milling expense. The ultimate comparison, however, would depend greatly upon the cleanness with which the ore could be broken down and handled. In the case of ore which must be milled, the cost of production per ton of concentrate is obviously dependent chiefly upon the yield per ton of crude ore and the cost of mining the latter.

In general, the lenses of ore in the veins of the Slocan are so irregular that no broad deductions can be drawn as to the cost of mining, or as to the cost of production, further than has been attempted above. This will be shown clearly in the following results that are available for a few mines.

Up to the end of 1904 the Payne mine is credited with an output of about 50,000 tons of ore and concentrates, worth about $4,000,000, out of which $1,438,000 was paid in dividends, which would indicate a cost of production of a little more than $50 per ton.

The cost of mining in the St. Eugene in 1904 was $2.49 per ton; milling, 62 cents; total $3.11. One ton of concentrate was obtained from 4.7 tons of crude ore. The cost per ton of concentrate was therefore 4.7 × $3.11 = $14.62.

The data of a mill run on ore from the Slocan Star mine, reported by Mr. W. F. Robertson, provincial mineralogist,* are instructive. The data cover the period from July 3 to July 15, 1904. They are as follows:

ITEM.	Tons.	Ag. oz. per ton.	Pb%	Zn%	Fe%	Mn%
Mill feed	1553.0	21.0	5.1	12.0	....	....
Galena concentrate	92.3	95.4	61.0	9.5	....	....
Blende concentrate	334.0	46.5	2.5	35.0	15.0	3.0

This shows a very fair technical result. The galena and blende concentrate contain together 75 per cent. of the silver, 81.5 per cent. of the lead and 67.5 per cent. of the zinc of the original ore. The galena concentrate alone contains 71 per cent of the lead. The blende concentrate contains 62.5 per cent. of the zinc.. One hundred tons of ore yield about 6 tons of galena concentrate and a little more than 21.5 tons of blende concentrate. Disregarding the silver content of the latter, it would not be a marketable, at least not a profitable product, but such an ore can be enriched to a grade of 50 per cent. zinc with loss of not more than 15 per cent. of the total zinc. This would reduce 334 tons assaying 35 per cent. zinc to 198.7 tons assaying 50 per cent. zinc, i. e. 100 tons of crude ore would yield 12.8 tons of concentrate assaying 50 per cent, zinc.

With lead at £16 2s. at London and exchange at $4.87, the value of a pound of lead in ore in British Columbia is approximately 2.5c. The value of galena concentrate containing 61 per cent. lead and 95.4 oz. silver is approximately as follows:

0.90 × 1220 lb. lead at 2.5c.	$27.35
0.95 × 95.4 oz. silver at 60c.	54.38
Total gross value	81.73
Deduct freight and treatment	12.00
Net value	69.73

*Annual Report of the Minister of Mines, British Columbia, for the year 1904, p. 184

The value of zinc ore with 50 per cent. zinc may be reckoned at $13 per ton f. o. b. mine, when spelter is worth 5c. in the market on which settlement is based.

The returns that might be expected from the mining and milling of such ore (5 per cent. lead and 12 per cent. zinc) under somewhat favorable conditions would be therefore approximately as follows:

Dr.	
Mining 100 tons (including development work) at $2.50	$250.00
Transporting 100 tons to mill at 10c.	10.00
Milling 100 tons at 60c.	60.00
Separating 21.5 tons of blende at $1.50.	32.25
Total	352.25
Cr.	
6 tons of galena concentrate at $69.73	$418.38
12.8 tons of blende concentrate at $13.00	166.40
Total	584.78

This shows a gain of $166.40 by zinc recovery at an expense of about $20 additional in milling and $32.25 for separation, a total of $52.25 leaving a net gain of $114.15, disregarding amortization and interest charges (see estimates and comments on pp. 90—91), which would be materially reduced in the case of a mine treating its own ore, and anyway are properly omitted in this estimate wherein similar charges are not made against the mining and milling plant. It is unsafe to base any calculations of zinc ore on a higher price for spelter than 5c. per pound at St. Louis, although the average price in 1905 was considerably in excess of that figure, and the average for 1906 promises also to be high. In this connection reference should be made to the statistical tables on pp. 141 and 143.

The method of development in the mines of the Slocan has been such that at the present time there is comparatively little ore that can be estimated as blocked out. There have been several mines, the most noteworthy being the Payne and the Slocan Star, which have made large outputs in value and a high percentage of profit in the production. It is probable that new and profitable ore bodies will be found in some of the mines, and it is probable, moreover, that many new veins, not now known, will be discovered within the mineralized area of the district. It is, finally, quite certain that intelligent management will materially add to the value of the production of many mines by careful recovery of zinc blende, although the latter may fetch only a small amount per ton.

The present lack of development, together with the irregularity of the ore bodies, make it difficult to formulate any reliable estimate of the zinc producing capacity of the Slocan. That this will bear a definite relation to the production of lead ore may be accepted. The statistics of several mines indicate a probable yield of two tons of blende concentrate containing 50 per cent. zinc to one ton of galena concentrate containing 60 per cent. lead. Unfortunately no statistics of the production of galena concentrate are available. The total production of lead ore in the Slocan during the nine years 1895-1903, according to the annual reports of the Minister of Mines for British Columbia, was 198,207 tons, averaging 40.74 per cent. lead. This is an average of about 22,000 tons per annum. This would correspond to about 15,000 tons of 60 per cent grade, and assuming the blende: galena ratio of 2:1 would indicate the possible production of 30,000 tons of blende per annum, if all the zinkiferous ore were concentrated. The latter is obviously an improbable result. Many of the zinc-lead mines are small affairs, for which it would never pay to provide mills, and are so situated that their ore can not profitably be delivered to a central mill. Many of the lead producers, moreover, have too little zinc in the ore to make its recovery worth while. On the other hand, one mine (the Lucky Jim) has produced a good deal of rich zinc ore with very little lead ore, and it is possible that further chimneys of zinc ore may be found in that property, or similar chimneys in other properties. All things considered, it is probable that 15,000 tons of zinc ore of 50 per cent. grade would be a liberal estimate for the productive capacity of the Slocan.

Mr. Argall considers that the mines of Ainsworth camp can produce from present developments about 54 tons daily of zinc ore of 50 per cent. grade, and may be able to attain an output of 100 tons daily of 45 to 50 per cent. grade in the course of a year or so if the extensive ore deposits be mined and milled on a scale commensurate with their magnitude. These estimates would correspond to 16,000 to 30,000 tons per annum, and in my opinion are extremely liberal.

The two large mines of the East Kootenay, which were examined by Mr. Argall and me, are not properly to be considered as zinc mines at all, although the Sullivan doubtless contains in ore developed more zinc than can be shown by any other mine in the Kootenays, except possibly the "Big Ledge." Its ore is, however, of such character that zinc extraction is almost hopeless. The St. Eugene mine is essentially a lead mine. Zinc ore occurs only at the edges of the ore lenses. A selected specimen of ore from such a position assayed 0.53 per cent lead, 8.72 per cent iron and 44.10 per cent. zinc. The average ore delivered to the mill contains, however, only about 4 per cent. zinc. A considerable portion of this inevitably goes into the lead concentrate, which averages about 6 per cent zinc. In dressing 400 tons per 24 hours, a recovery of about 25 tons of zinc concentrate assaying 20 per cent zinc and 4 per cent lead was effected. A rough statement of the behavior of this ore is therefore as follows:

ITEM.	TONS ORE.	ZINC ASSAY.	TONS ZINC.
Mill feed	400	4%	16.0
Galena concentrate	80	6 "	4.8
Blende concentrate	25	20 "	5.0

The zinc in the blende concentrate is comparatively small in quantity, is finely disseminated, is associated with difficult minerals (e. g. garnet) and is apparently not economically amenable to further concentration. The treatment of this material has already been investigated by the St.

Eugene company, which made some changes in the mill to facilitate its recovery. The tests made by the company failed to show that any commercial product could be made of it, and its recovery was consequently discontinued. However, the tests made by the Commission at Denver, Colo., show that the enrichment of this zinc ore is, not so difficult a matter as appears at first sight (see Mr. Argall's report).

On the opposite side of Moyie lake from the St. Eugene property there is a prospect known as the Aurora which has shown some zinc ore, but according to the account of the owner development work has not yet progressed far. A selected sample from a large sack assayed 2.46 per cent. lead, 5.97 per cent. iron and 56.75 per cent. zinc. This is, of course, a shipping product.

There are many mines and prospects in other parts of British Columbia which according to many accounts contain zinc, as is to be expected in view of the wide occurrence of that element. Their usefulness as sources of the metal depends largely upon their situation with respect to railway.

Methods of Exploitation.

It is obvious from the previous estimate of the probable zinc resources of Ainsworth and the Slocan that the tonnage of ore to be expected is too small to warrant the erection of independent zinc enrichment plants at the several mines. Referring to the Slocan district, an output of 15,000 tons of blende of 50 per cent. grade per annum would correspond to about 25,000 tons of mill concentrate of 35 per cent grade, which would be only 84 tons per day, a quantity that could easily be handled by one central plant of moderate size. The peculiar railway situation in the district, which is entered by two competing railways, one broad-gauge and the other narrow-gauge (preventing any physical connection between them), one leading to Slocan Lake and the other to Kootenay Lake, furnishes justification for two smaller mills, one on each lake, rather than a single large mill. This has already been recognized. The Kootenay Ore Company, under the management of Mr. George Alexander, has equipped a mill at Kaslo, on Kootenay Lake with magnetic separators, and the Monitor & Ajax company, under the management of Mr. Maurice Gintzburger, is planning a similar equipment at its mill at Roseberry, on the shore of Slocan Lake.

The success of these enterprises will depend upon the broadness of the business policy whereunder they are conducted. Business enterprises are always better run under private management than under Government management, and small business enterprises are generally more successful as undertakings of the individual than as corporate undertakings. It is important to the mining interests of the Slocan that the enrichment plants be not in the first place burdened with unnecessary capital charges because of mistakes in design and construction, and be operated in the second place at charges which will return a fair profit on an honest investment and will not unnecessarily or exorbitantly tax the producers of rough concentrate. If these conditions be not complied with it will be worth while for the producers to consider the joint erection of independent plants, which will ensure the efficiency and economy that the conditions of the district absolutely demand.

These conditions must also be recognized by the railways. Zinc ore from regions so remote from the established smelting centers of the world will not stand high freight charges. This will appear clearly from the the discussion of the valuation of ores in a previous section of this report. The zinc ores of British Columbia, Mexico, and the portion of the United States that is west of the Rocky Mountains have been made marketable only by the liberal policy of the railways in making extremely low freight rates on the long hauls. The lower the grade of the ore the less freight will it stand. The local freights on rough concentrates to the enriching plants should be extremely low. A broad policy would be the making of a through rate from the concentrating mill where the ore originates to the final destination of the ore, such a rate to cover milling in transit at the enrichment plants. The railway company is entitled to make a profit on its investment, but the broad view regards the prosperity of the district served rather than the profit or loss on a single class of freight. This principle requires no elaborate argument.

With respect to milling the crude ore, the Slocan has been already surfeited with mills. No new mill should be erected unless a tonnage of ore sufficient to justify its erection has been thoroughly demonstrated. Most of the existing mills will, however, need to be remodelled for zinc recovery. Whenever it be possible, mines unprovided with mills should take advantage of those already constructed. A mine situated like the Lucky Jim, for example, can cheaply put its ore on board cars and the latter can be cheaply unloaded at a mill further along the line, like the Payne, for example. On the carriage of milling ore, the railways should make extremely low rates. The future prosperity of the Slocan depends very much upon the cooperation with which its resources are developed and the advantage which may be taken of its present investments in plant, i. e. railways, mills, tunnels, tramways, etc. Unjustified and unnecessary investments in plant have been a prolific cause of disaster in the past.

To go further backward, this time to the mines, there are many which can neither afford a mill nor can economically deliver their ore to some other mill. For these, the only resource is hand-sorting of blende in the same way that the galena now produced is hand-sorted. The method, however, can be generally much improved by the introduction of some mechanical devices to facilitate the work.

Hand-sorting seldom receives the degree of attention which it deserves. It is one of the simple things that is disregarded in the desire to have a mill, to do everything by machine, and to crowd the ore through. The advantages that may be derived from hand-sorting are coming more and more to be recognized and it is being introduced as an accessory process in some of the largest and most expensively equipped mills. A good deal of the zinc ore of Wisconsin, Leadville, Colo., and Park City, Utah, is prepared by hand-sorting only. No one need therefore be ashamed to resort to this simple process or feel that in doing so he is not up to date and economical. There is perhaps no district in which hand-sorting can be applied with more advantage than in the Slocan, both in connection with the mines which have mills and those which have not, for two reasons, viz., the galena is very rich in silver, wherefore the more it can be separated coarse the higher will be the saving, and the blende is almost always associated with siderite, which can not be separated by jigging and tabling with any considerable degree of cleanness.

Hand-sorting has the advantages of saving clean mineral from unnecessary crushing, therefore avoiding loss in slime; of separating two minerals of equal, or nearly equal, specific gravities, like blende and siderite, or blende and pyrites; of saving in cost of plant. It is not likely to be any more expensive in direct cost than a cheap process of milling, like jigging; it is apt to be a good deal cheaper than some of the more expensive processes, like cyaniding. The cost of picking depends chiefly on the size of the pieces of ore and their specific gravity. It is obviously cheaper to pick out lumps of 2 in. size than $\frac{3}{4}$ in. size, while lumps of a heavy mineral, like galena, weigh up faster for the same expenditure of labor than a light mineral, like quartz. It is apt to be uneconomical to pick over ore smaller than $\frac{1}{2}$ in. size. Ore of 1 in. size can generally be picked to advantage. With labor at 30 cents per hour the cost of picking galena ought not to exceed 60 cents per ton, or 90 cents per ton for blende, and it may be considerably less if the ore be favourable. Even these figures do not compare unfavourably, however, with the cost of milling in the Slocan, which can scarcely be done for less than 60 cents per ton of crude ore if blende recovery be aimed at.

In order to do hand-picking effectively there must be a systematic method. The pickers should not have to do any breaking of the ore, which should be done by a jaw-crusher, if the quantity be sufficient to warrant the installation, or otherwise by men sledging with long-handle hammers. The broken ore may be delivered to a stationary picking table, or to a movable one. The former is the cheaper to install. Among the movable tables, the circular, revolving table is cheap and convenient. A traveling belt is also very convenient to pick from. These devices, together with methods and results of picking, are fully described in Richards' *Ore Dressing* and in my *Production and Properties of Zinc*. It is unnecessary to enter into further discussion of the subject here, but attention may be called to certain important points, viz.:

1. The ore should be rinsed with water, so as to make the minerals show distinctly.

2. The picking-room should be well lighted.

3. The pickers should be able to throw the ore from them; not draw it toward them.

The usefulness of systematic hand-picking in the preparation of the zinc ore of the Slocan can not be too strongly emphasized. The owners of small mines instead of waiting for some one to come forward to build a mill should get to work themselves, pick out for shipment what ore they can, save the cullings, and prosecute the development of the mine until the time comes when a mill is really justified.

The mines themselves are peculiarly well adapted to an extensive application of the leasing system. In small, irregular veins, such as those of the Slocan generally are, the leaser can invariably do better than the company. He has a keener scent for ore than the average foreman; he is free from administrative expenses; he takes risks that the company can not; and when the mine is in *borrasca,* he will work for less than the regular wage in the hope of recouping himself when the mine comes again into *bonanza.* There is many a mine in the United States, which has been unprofitable under company management and has become profitable when turned over to leasers. The general introduction of this system in the Slocan would do a vast amount of good to the district in general and the proprietors of mines in particular.

In this connection steps should be taken to settle local differences and jealousies. Prosperity can be insured only by the inauguration of general activity. Consideration may well be given by the Provincial Government to an adjustment of the system of taxation which would tend to secure the operation of properties that are capable of profitable operation.

ZINC SMELTING IN CANADA.

The local smelting of the ores of a country is always an important consideration. The smelting of zinc ore in British Columbia is especially important for consideration in view of the remoteness of the province from the markets of the world, and moreover inasmuch as a smelting plant is already under construction at Frank, Alberta, for the treatment of these ores. This plant will doubtless be completed before this report is issued from the press.

In the smelting of zinc ore there are two prime considerations. (1) The quantity of fuel required per ton of ore is greater than in any other of the common metallurgical processes. In the best practice of the world, about 1.75 tons of coal are required per ton of ore, while in inferior practice the proportion is much larger. It is therefore cheaper to take the ore to the coal than to take the coal to the ore. (2) The zinc smelting process requires ore concentrated to a rather high grade. Generally, ore to be smelted in Belgian or Rhenish furnaces should contain upward of 40 per cent. zinc in the raw blende. Inasmuch as such ore will yield approximately one third of its weight in spelter, so long as the ore is taken to coal on the direcz line to the final marketing of the spelter, the waste carriage, so to speak, is only on two thirds of the weight of the ore, and it may be actually less than that, inasmuch as the ore is usually rated as a lower class of freight than metal, and a portion of its lead and sulphur contents may be utilized.

The above considerations are the chief factors in determining the location of a zinc smelting works. There are many others, such as the question of local freights, supply of labour and refractory material, market for sulphuric acid, etc., which need not be entered into here. It may be remarked incidentally that the electric smelting of zinc ore is one of the possibilities of the future; in fact it is being practiced already in Scandinavia; and this substitutes water-power for part of the coal required in the process, although it can not replace the coal required as reduction material. I shall refer to the subject of electric smelting in a subsequent section.

Considering the subject of smelting according to the standard practice, the only localities of British Columbia that are in any way adapted to the process, are the Crow's Nest Coal Field* and the Coast, where coal can be obtained from the coal field of Vancouver Island. The Canadian Metal Co., Ltd., has erected its plant at Frank, Alberta, in the Crow's Nest Coal Field, just east of the British Columbia line. The selection of that place appears to have been determined by the acquisition of a coal mine, directly on the railway line, from which the coal can be trammed immediately into the works. Undue weight may have been attributed to the latter advantage, but the direct control of a source of coal was a wise enterprise, especially if the coal can be produced as cheaply as is estimated. It is difficult to see how zinc smelting could be profitably carried on in British Columbia with coal obtainable only at the price asked by the Crow's Nest Pass Coal Co.

The Crow's Nest Pass coal field is situated on the western slope of the Rocky Mountains. According to Dr. George M. Dawson† it has an area of at least 200 square miles, and contains numerous superimposed coal seams, ranging in thickness from 2 to 30 ft. The chief mining in this field is done by the Crow's Nest Pass Coal Company, the history of which is practically the history of the field. This company was incorporated by Dominion charter in 1897. It owns about 250,000 acres of land. It began mining at Coal Creek in 1897. Its production from that time up to the end of 1903 was as follows:—

YEAR	TONS, COAL	TONS, COKE
1898	8,986	361
1899	116,200	29,658
1900	232,345	72,810
1901	425,457	125,085
1902	441,236	120,777
1903	661,118	167,739
Total	1,885,342	516,430

* In using this term, I include the coal formations on both sides of Crow's Nest Pass.
† The Mineral Industry, Vol. VII, p. 200.

Legislative restrictions prevent the Crow's Nest Pass Coal Company from charging more than $2 per ton for mine-run coal at the mines. During the years 1901, 1902, 1903 and to April 1904, the company paid dividends aggregating $968,947, which will furnish a rough indication of the cost of production; anyway, testimony that there is a good profit in selling the coal at $2 per ton. The coal is of good quality. Nine samples taken by the Government mine inspectors in 1902, representing coal as it could be mined for shipment, and not selected samples, averaged: Moisture, 0.91%; volatile combustible matter, 19.1%; fixed carbon, 69.93%; ash, 9.83%; and sulphur, 0.32%.* The mining rate in the field in 1905 was 70 cents to $1.15 per ton of screened coal.

The Canadian Metal Company, at Frank, has a 9-ft. seam of coal, standing vertically and opened by an adit level. Mr. J. J. C. Fernau, manager of the company, stated that he had made a contract for mining at 65 cents per ton of mine-run coal, exclusive of timber and supplies. The coal is bituminous and apparently of good quality. Its total cost may be reckoned roughly at $1 per ton.

The conditions of zinc smelting in British Columbia and Alberta may be considered therefore as based on a coal cost of $1 to $2 per ton. As compared with European conditions this is a low cost for fuel. As compared with American, taking into consideration the quality of the coal, there is no great difference. Few American smelters, if any, are able to obtain mine-run coal as low as $1 per ton, and if they do, it is the inferior bituminous of the West.

Besides coal, the important factors in zinc smelting are labour and refractory material. In Europe, coal is the largest single item in the cost of smelting; in the United States, labour is the largest, the rate of wages being higher than in Europe. While the labour cost in smelting a ton of ore is a fundamental consideration in determining rentability. there is another important consideration which must be reckoned upon in starting a new plant in a new locality. This is the question of obtaining skilled men. The zinc smelter is made, not born. There is no metallurgical process in which so much is dependent upon the skill of the workman as in zinc smelting; nor is it possible to eliminate manual labour by mechanical devices to so great a degree as it is in other smelting process. No matter how perfectly the zinc smeltery be designed; how elaborate its construction; and how modern its methods—poor smeltermen will neutralize all the advantages planned by the engineers. This is one reason why zinc smelting has been for 100 years confined largely to certain localities, where men skilled in the art are obtainable; it is a reason why European smelters are sometimes slow to extend their plants; and it is a reason why certain modern and very costly plants established in new localities have proved disappointing for several years while the workmen have had to be trained, in spite of the proportion that could be imported from older smelting centres. This is a consideration which often is not taken into account in planning a new plant, but it must not be lost sight of.

* Annual report of Minister of Mines for British Columbia, for 1902, page H 262.

It is difficult to estimate the probable labour cost in smelting zinc ore in British Columbia, because the art has not heretofore been practiced there, and there is consequently no precedent to serve as a guide. In a thoroughly modern plant costing about $16 per ton of annual capacity, in the United States, the smelting of 2,000 lb. of sulphide ore would require approximately the labour of 2¼ to 2⅜ men for one day at an average wage of $2; in other words, the labour cost of smelting would be $4.50 @ $4.75 per ton. Wages range from 15 cents per hour to 25 cents per hour ($1.50 to $3 per day). At the lead smelteries of British Columbia wages range from 25 to 50 cents per hour ($2.50 to $4 per day). It would appear to be a low estimate to reckon an average wage of $3 in zinc smelting in Canada, which would make the labour cost per ton of ore come to $6.75 at the minimum, and in all probability it would be nearer $7.50 per ton.

The refractory material, i.e., fire-clay, required for the preparation of the retorts must be of special character, which can be determined absolutely only by practical trial. For this reason it is not worth while to enter into a description of the chemical and physical characteristics of the clays employed for this purpose, which can be found in metallurgical treatises. Zinc smelters are extremely conservative in their use of fire-clay. Among the abundant clay resources of the United States, there is very little employed except the clay dug at St. Louis, Mo. This clay is used exclusively by the smelters of Kansas, Missouri and Illinois. The smelters of Pennsylvania and Virginia, who are remote, also use it. The United States Zinc Co., at Pueblo, Colo., after using St. Louis clay, developed a satisfactory material locally and there is no doubt that many suitable clays exist in the United States, which could be used if the conservatism of the smelters could be overcome. Similarly, it is possible that local clays available for the process. may be developed in British Columbia and Alberta in connection with the coal measures, but so far they have not been looked for, and the company at Frank intends to use St. Louis clay. This must necessarily be costly because of the long railway carriage. It is likely to cost $12 per ton delivered in Canada, and about one-eighth of a ton of clay being required per ton of ore, the clay cost in smelting will be $1.50 per ton of ore.

Reckoning repairs and renewals, supplies, etc. at 10 per cent. advance over Kansas-Illinois conditions, and administration at the same figure as in the United States, the approximate cost of smelting in British Columbia and Alberta would appear to be as follows:—

2.5 tons of coal at $1.50	$ 3.75
2.5 day's labour at $3	7.50
⅛ ton clay at $12	1.50
Supplies	.220
Repairs and renewals	.825
Administration	1.100
Total	14.895

A smelting cost of $15 may be estimated as a round figure.

The Canadian Pacific Railway has made a freight rate of $2.50 per ton on ore from the Slocan to Frank. The rate on spelter to London will probably be $14 to $15 per ton. Assuming an ore assaying 48 per cent. zinc and yielding 800 lb. of spelter per ton, the charges per ton of ore up to delivery of the spelter at London will be as follows:—

Freight on 1 ton ore at $2.50	$2.50
Smelting 1 ton ore at $15.00	15.00
Freight on 0.4 ton spelter at $15	6.00
Total	23.50

If the ore were shipped to Europe for smelting the costs would be:

Freight on 1 ton ore to Antwerp at $13	$13.00
Freight on 1 ton ore to works at $0.88	0.88
Loss and incidentals	1.25
Smelting 1 ton ore at $9.25	9.25
Freight on 0.4 ton spelter	0.65
Total	25.03

It will appear therefore that smelting in Canada is feasible commercially, especially since a part of the spelter produced in Canada can be marketed domestically, saving something in freight and gaining in price. It is possible also for a Canadian smelter to compete successfully with the American so long as the United States assesses a duty of 20 per cent. *ad valorem* on blende. It is to be remarked, however, that this conclusion is based on a plant of high efficiency, in thorough running order (manned competently), and on an estimate for labour that is doubtful. Until the last point has been settled, I should leave considerable leeway to cover unfavourable contingencies, and to anyone contemplating zinc smelting in the Canadian West I would emphasize again the necessity for being content with disappointing results for a considerable period while a sufficiently skillful working force is being secured.

Reference to the statistical tables given elsewhere in this report will show that the consumption of zinc in Canada is comparatively small, and (although on the increase) it is gaining but slowly. The imports (which may be taken as representing the consumption) were 3,154 tons in the fiscal year ending June 30, 1905, against 2,975 tons in 1904, and 2,490 tons in 1903. In 1895 they were 1,214 tons. There was certainly a healthy increase from 1895 to 1905, but the course of the statistics is indicative that it will be a long while before Canada will be able to consume all of the zinc which British Columbia alone is capable of producing. I have already estimated that Ainsworth and the Slocan are now capable of producing (when the necessary milling facilities have been provided) about 30,000 tons of blende per annum of an average (nominal) zinc content of 50 per cent. This would correspond to a spelter production of upward of 12,000 tons, or approximately four times the present consumption of the Dominion. The consumption, moreover, is only about one-half in the form of spelter, the remainder being in the form of sheet and rolled plate. It is to be expected, therefore, that for a long time to come a large portion of the zinc ore or spelter produced in British Columbia will be exported. The spelter cannot enter the United States, because the American tariff of 1.5 cents per pound is prohibitive; besides America itself is at times a considerable exporter of spelter. The plans that are on foot for zinc smelting in Australia would appear to provide for the requirements of the Australian market and much more. The prospects are, consequently, that the major portion of any spelter that may be produced in Canada must be marketed in Europe.

This leads to the inquiry as to the possibility of smelting on the Pacific Coast. Coal exists there. The rail transportation on ore is comparatively short. Deposits of ore exist near the shore, both on the mainland and on Vancouver Island. Water power is available. And spelter can be carried either to Europe or to the Orient by steamships or sailing vessels. If reasonably cheap coal were available, it is likely that a good harbour on the Pacific Coast would be the most advantageous point in British Columbia to conduct zinc smelting on a large scale.

The demand for coal along the Pacific Coast, however, creates a high price for that fuel. It fetches $2.50 to $3 per ton at Nanaimo. Values are not stated separately for the collieries of Vancouver Island in the statistics of the Minister of Mines for British Columbia, but making allowance for the Crow's Nest production at its known value, the average for the Vancouver Island coal appears to have been a little upward of $3 per ton (2240 lb.) in 1904, and the same price was quoted in 1905. The coal of Vancouver Island is of good quality, but it is improbable, in view of the high cost for labour, etc., that smelting could be successfully conducted on the Coast with coal costing $3 per ton. It is reported that coal seams have been discovered on Quatsino Sound, where zinc also has been discovered. It is to be assumed, however, that in spite of the competition of California petroleum, which has already had a noteworthy effect on the coal trade of British Columbia, the demand for fuel on the Pacific Coast will be such as will absorb new supplies at about the present price. The prospect for zinc smelting on the Coast, at least by the standard method, is too remote to merit detailed consideration at the present time. The zinc deposits of the Coast are still of unknown magnitude; they are in fact nothing but prospects. No one would consider any plan for smelting ores from them until they had been extensively developed. The ore that may be taken out in the course of their development will find a ready market in Europe, whither transportation will be comparatively cheap. This is the policy which the owners of zinc deposits on the Coast should pursue, viz., develop the mines and market the ores in Europe.

As previously pointed out, the immediately available zinc resources of British Columbia are those of Ainsworth and the Slocan. In seeking their market, these ores must necessarily pass over either the Canadian Pacific or the Great Northern railways. In either case, the railways obtain the longer haul by taking the ore eastward. The Canadian Pacific is indeed able to carry the ore, through its connecting steamship line, directly to European ports, without dividing freight charges with any other company. It is unlikely therefore that either of those railroads would permit the ore of this portion of British Columbia to go westward. It is understood, however, that a rate of $4 per ton to the Coast has been tentatively made by the Canadian Pacific.

The approximate rate on ore shipments in large quantity from Vancouver to Antwerp is 30 shillings per long ton, or about $6.50 per 2000 lb., but various things have to be taken into consideration; such as whether the ore is to be shipped in sacks or in bulk, the quantity and regularity of the shipments, the time of the year, i. e. whether the busy season or otherwise. All of these conditions affect the rate.

The possibility of the utilization of the abundant water power of the Coast for the electric smelting of zinc ore will be referred to briefly further on.

THE DESIGN OF A ZINC SMELTERY.

The process of zinc smelting is theoretically one of the simplest of all metallurgical processes. Practically it is one of the most difficult. This is because of the variety of steps that the ore must go through, the carefulness with which many details must be observed, and the necessity for carrying out the reduction of the ore in small retorts, made of imperfect and fragile material. No one should think of constructing a zinc smelter without employing a metallurgist skilled in the art. If the plant be of any considerable size, its construction is an engineering problem of magnitude.

It has been pointed out previously in this report that the cost of a zinc smeltery per ton of capacity is relatively much higher than the cost of a copper or lead smelting plant. It has also been pointed out how wide a range there is in the cost of zinc smelting plants of various constructions. The cost per ton varies not only with the type of construction, but also with the size of the plant, a small one costing relatively more than a large one. From one standpoint even a large zinc smeltery may be considered a small plant. A works that is capable of treating 25,000 tons of ore per annum is a large plant, whereas a lead or copper smeltery treating 300,000 tons per annum would be nothing extraordinary. It is constantly to be borne in mind, however, that the zinc smelter deals with ore already concentrated to high degree, and the plant of 25,000 tons annual capacity will produce 10,000 tons of metal, which is as much as the average copper smelter of 300,000 tons capacity produces. In points of ground area occupied and the magnitude of the brick and steel construction there is comparatively little difference between the two plants.

The old type of zinc smelting works common in the United States could be built in small units and at comparatively small cost. There are still in use, both in Europe and America, many of the old direct-fired Belgian furnaces. In some places, where coal is very cheap and labour not excessively dear, they may still be employed with profit, but in general it would be an exhibition of folly to erect such furnaces in a new plant, and if coal and labour were dear it would be suicidal. Economy of coal is so important that regenerative, gas-fired furnaces should alone be considered.

The subject of zinc smelting is so comprehensive that bulky treatises have been written upon it, without by any means exhausting it, and obviously it cannot be entered into to any great extent from the technical standpoint in the limits of this report. It will be aimed only to indicate the general lines of a modern plant, and to give some advice upon the policy of construction.

If a new plant were projected in a district where zinc smelting had already been successfully practised, where the conditions had been well established and where an abundant supply of skilled labour could be relied upon, the wisest policy would be to plan a unit of maximum efficiency, say 25,000 tons of ore capacity per annum (beyond which little or nothing is to be expected in reduction of operating costs) and proceed directly with the construction on that scale. If for any reason, it should appear inadvisable to give the plant its full capacity at the outset, it should nevertheless be laid out on the full lines, omitting a certain number of the roasting and distillation furnaces, which can be added subsequently, as required.

In inaugurating zinc smelting in a new country, it is the policy of prudence to begin with the minimum of outlay for plant, erecting something small that will serve to establish the conditions and will serve to train the men. The chances are that the new plant will lose money at first, and a small plant will lose less than a large one. Thus the Sulphide Corporation began at Cockle Creek, Australia, with a single furnace, although it had very large resources of ore. It appears that the Canadian Metal Co., at Frank, Alberta, may have begun on somewhat too ambitious a scale. However, this is a broad question of policy that is dependant largely on the plans and resources of the promoting company.

It is not at all easy to plan on modern lines a very small plant that will be capable of expansion; or rather it is not easy to do so and keep the original cost per ton in reasonable proportion to the ultimate cost. This is because even in a large plant the quantity of material to be handled daily is comparatively small; 25,000 tons per year is only about 72 tons per day. Now, although the roasting and smelting furnaces for that quantity of ore are naturally divisible into small units, there are other parts of the plant which are not. For example, the retort making machinery for making the retorts for 72 tons of ore per day constitutes a single unit, which must be provided just the same for nine tons of ore per day. There is similar trouble in the design of the sampling and crushing mill, the power plant, and other accessory parts of the works. The only solution of the difficulty is to divide everything into units so far as possible, and leave out such parts as may be temporarily omitted at the expense of increased operating cost for a while, or until the plant has demonstrated its prospect of success.

I will, therefore, briefly indicate the general specifications for a plant of 25,000 tons annual capacity, or 72.5 tons of raw ore per day (estimating 345 working days per annum). The quantity of roasted ore to be smelted daily will be about 62 tons. The yardwork must be, however, so far as possible done without counting Sundays, or say in 300 days per year, which will necessitate the handling an average of 83⅓ tons of ore per day.

Suitable arrangements must be made for unloading the ore, coal and other material as it arrives at the works. There must be well arranged railway sidings and ample room for the cars to stand, with a convenient system of shunting. The ore, coal, and all material should be weighed upon receipt at the works. Immediately after weighing, a sample of the ore should be taken for determination of moisture. As the ore is unloaded it should be sampled for determination of its composition. Proper metallurgical control of the works cannot be maintained unless it be known absolutely what is received. The relation to this of the products which go out shows the percentage of extraction, or the efficiency of the works and their management; nothing else shows it absolutely. The lump ore should be crushed and sampled directly as it is unloaded. The fine ore (concentrates) may be sampled without further crushing, although generally it will have to be crushed before it goes to the roasting furnaces. It is advisable to pass the fine ore first through a dryer (a simple cylindrical dryer is best), which avoids difficulty in winter, when the ore is likely to freeze. For crushing the lump ore, a Blake crusher and rolls should be employed; for the fine ore, rolls only are required. The sampling should be done mechanically. The Vezin sampler is recommended.

The sampled ore is conveyed to storage bins, which should be subdivided to keep the various lots and kinds separate. If the bins are overhead and self-emptying, they save labour in subsequent handling, but they are more costly than bins on the ground. The bin capacity should be liberal —at least enough to hold a month's supply of ore.

From the storage bins the ore is put through the crushing mill for reduction in size to about $\frac{1}{16}$ inch, or so as to pass an 8-mesh screen. In the crushing mill the ore should pass first through a dryer (if it has not previously been dried) and then through rolls. It is much easier to crush and screen dry ore than it is damp ore. The crushing mill should have capacity for 10 tons per hour. Its design will depend somewhat upon the relative proportion of lump ore and concentrate that is expected.

From the crushing mill the ore will go to the roasting furnaces. These should be mechanical. The Ropp and Brown have given good results in the United States, but they are rather uneconomical of fuel. The Hegeler has given good results, especially when the roast gases are to be used for sulphuric acid manufacture, but its first cost is high, and the labour cost of its operation is high per ton of ore unless it be constructed as a very large unit. The Merton furnace has been successful at Swansea and is rationally designed. This furnace has been installed at Frank. Furnaces of the McDougall type are to be recommended. They are economical of fuel and labour, are low in first cost, and are efficient in small units. Twelve furnaces, 12 ft. in diameter and six hearths high, should roast 72 tons of ore per day. A well designed mechanical furnace may be reckoned as having capacity to roast 20 lb. of blende down to 1 per cent sulphur per square foot of hearth per 24 hours.

The roasted ore should be collected in storage bins contiguous to the mixing house. It is advisable to carry a stock of ore between the roasting and distillation departments for two reasons: (1) To insure continuity of operation in case of an accident to a roasting furnace; and (2) to enable the proper mixture of various kinds of ore to be made. Contiguous to the mixing house there should be also a crushing mill for the comminution of the reducing material (coal and coke) to ¼ inch size. The charge should be weighed out into a concrete mixer (a cube or revolving barrel) and therein thoroughly incorporated. European smelters perform both the grinding of the coal and the mixing of the charge in the same operation in a Vapart mill, but I consider this less advantageous than what I have outlined above. The making up of the charge and its thorough mixture is one of the keys to good work in smelting and attention should be concentrated on this department.

Among the smelting furnaces, the most modern are the Rhenish-Siemens, the Ferraris and the Convers & De Saulles. All are good, and it would be difficult to determine which is the best, in the absence of competitive trial under precisely the same conditions. At the Palmerton works of the New Jersey Zinc Co., the Convers & De Saulles (counter-current recuperative) and the Siemens (reversing regenerative) have been tried and the company has decided in favour of the former; it has proved less economical in fuel than the Siemens furnace, but offsets this disadvantage by somewhat higher extraction of metal, because of its more even temperature. In my own opinion, however, the countercurrent type of furnace is inferior to the reversing-regenerative type, especially the Rhenish-Siemens and the Ferraris.. In Europe the Rhenish-Siemens furnace is extensively employed in Belgium and Silesia, with excellent results; it is used at Pueblo, Colo. The Ferraris furnace (reversing-regenerative, with checkers for air only) is used with good results at Monteponi, Sardinia (a comparatively new plant). The furnaces erected at Frank, Alberta, are of this type, but they were designed without the approval of the inventor and embody modifications, which are not to be commended. The regenerarative chambers at Frank, instead of being built under the hearth of the furnace and forming an integral part, of it, are built independently, outside of the furnace house, besides which other undesirable alterations in the design have been made.

Irrespective of the system of heat recuperation, the Rhenish type of furnace is now the standard in zinc smelting. The heat recuperative system may be either reversing-regenerative or counter-current. The Ferraris furnace, as used at Monteponi, is substantially of the Rhenish type, but has an interior middle wall, which is necessary to take the gas flues. The standard Rhenish furnace has 240 retorts, arranged 120 per side, in three rows of 40 each. The retorts are commonly 60 in. long, 7 in. wide and 12 inches high (inside measurements). A furnace of this size takes a charge of about nine tons of roasted ore, and eight furnaces should therefore be reckoned for the works. In modern plants the furnaces are set so that the working floor is about 10 ft. above ground level, the lower floor being entirely open around the base of the furnace. This increases the first cost, but reduces the operating cost and is far better for the workmen. The furnace house must be designed to afford ample room in the front of the furnaces and to insure good ventilation.

The gas producers for supplying the gaseous fuel to the furnaces should set outside of the furnace house, but as near thereto as is possible. Each furnace requires two producers of 8 ft. diameter, or three of 7 ft. diameter. Two of 8 ft. diameter give the equivalent of 100 sq. ft. of grate area per furnace. The modern gas producer, however, has no grate. The producers may be of any standard type, such as the Taylor or Duff. They should be blown with air, together with steam enough to produce a gas rich in hydrogen; and both in design and management should conform to the best modern practice in gas-firing. The coal supply for at least one day should be carried in a hopper over each producer. These hoppers should be kept filled from a railway line, from which the cars can be dumped into them by gravity, the railway line being either of standard gauge communicating with the ground by an incline, or a narrow gauge auxiliary line.

Each pair of distillation furnaces requires a chimney 135 ft. in height and 5 ft. in diameter. The chimney should be divided by an interior vertical partition to a height of 30 ft. to insure equal draught upon the two furnaces.

If lead-bearing ores are to be distilled, the spelter will generally carry 2 to 3 per cent. of lead. By remelting in a simple reverberatory furnace provided with a sump in the hearth, the excess of lead will settle to the bottom, reducing the lead in the spelter to about 1 per cent. A furnace with hearth 16 ft. long and 6½ ft. in width will refine 9 to 10 tons of spelter per day.

The pottery, wherein the retorts and condensers required by the distillation furnaces are manufactured, should be situated conveniently to the furnace house. This is a highly important department, because success in smelting depends upon the excellence of the retorts. The pottery is equipped with machinery for crushing the clay, which generally has to be reduced to about 10-mesh size, and apparatus for mixing it and for molding the retorts.

For clay-crushing ordinary rolls, or dry-pans (edge-runners) may be used. The mixture of ground clays may be effected by hand, the quantity being comparatively small. The mixture (batch) is then kneaded with water in a pug-mill. The type known as the taper-tub pug-mill is best. The pugged clay is compacted into large cylindrical blocks by a hammering machine. The blocks are put into the cylinder of an hydraulic press, from which they are forced out, under pressure of 150 to 200 atmospheres as finished retorts. The hammering machine and press, together with the pumps for their operation are made by C. Mehler, of Aachen, Germany. The Mehler press is used by all zinc smelters who make retorts in this fashion, and is the only machine of its kind. The standard machine has capacity for the manufacture of 80 retorts per 10 hours, which is an ample supply for an eight-furnace plant.

The molded retorts are transferred to the hot room of the pottery, where they are allowed to remain, standing on end, until thoroughly dry. The temperature should be maintained at about 100° F., this being effected either by steam pipes, or by a hot-air furnace. The retorts should be allowed to dry for three or four months. This, together with the daily consumption (which may be reckoned at 3% of the number in the furnaces), determines the floor space and consequently the size of the retort store-house. The pottery should be separate from other buildings and should be well safeguarded against fire.

The power required in the works should be generated electrically for the operation of motors where required. In view of the comparatively large area occupied by a zinc smelting works and the small amounts of power required at many points, this is vastly more convenient, and in the long run more economical than the transmission of power by shafts, belts, wire ropes and steam pipes. The attempt to distribute power in any of those ways, or a combination of them, leads inevitably to a cramped arrangement of the plant, which diminishes its efficiency, prevents advantageous extensions, and introduces other objectionable features.

The best general arrangement for the works is a straight line, beginning with the ore bins, and following in order with the crushing mill, roasting furnaces, mixing house, distillation furnaces and spelter shipping house. Suitable space for any probable extension should be left between each department. Accessory departments are located on parallel lines, the pottery for example opposite the distillation furnaces. The kilns for burning the clay required in the batch for the retorts will be placed conveniently to the pottery, and the tempering furnaces for annealing the retorts before insertion in the distillation furnaces must be situated near to the latter. The works are best laid out on level ground. The ideal site is one, however, which affords large dumping space for residue and enables the latter to be discharged by gravity, irrespective of whether it is or is not to undergo a subsequent milling for recovery of its silver-lead content. If it is to be milled, the plant required is quite similar to an ordinary dressing works, the residue being to all intent and purpose like a raw low-grade ore. If it is to be milled, not for discarding waste, but merely for separation of its constituents, the products should be collected in loosely constructed bins, lined with burlap, through which the water can escape, while all the products, including slime are retained.

THE SMELTING WORKS AT FRANK, ALBERTA.

These works are owned by the Canadian Metal Co., Ltd., a foreign corporation. At the time of my visit (about the middle of November, 1905) the works were still under construction. It is therefore impossible to report fully upon them, although their general outline may be indicated.

The buildings of the works are laid out on two parallel lines. In one line, there is the office; chemical laboratory; and a brick building which is divided into sections, the first being occupied by the crushing mill, the second by the power plant, and the third by the pottery. Beyond this building, there is another designed for the gas producers, to which the coal is to be delivered directly from the mouth of the adit, close by, leading into the coal mine. On a parallel line there is a long brick building which is also divided into sections, the first being occupied by the roasting furnaces, the second by the mixing department, and the third by the distillation furnaces.

The pottery is equipped with a Mehler press, hammering machine, and hydraulic pump.

The roasting compartment comprises five Merton furnaces, which have been constructed five hearths high, instead of the customary three hearths. The hearths are 8 ft. wide and 28 ft. long, besides which there are two, circular finishing hearths, one 8 ft. in diameter, the other 12 ft. in diameter. The hearth area per furnace is consequently $(28 \times 8 \times 5) + 50 + 113 = 1283$ sq. ft., which at 20 lb. per sq. ft. would give a capacity of about 12½ tons per furnace, or 62½ tons per 24 hours for the five, but so high a hearth efficiency is hardly to be expected from the modified design of these furnaces. The furnaces are to be gas-fired. The ore is to be delivered to them by a long screw conveyor extending over the line of furnaces, which set parallel with each other and at right angles to the longitudinal axis of the building. The roasted ore is to be delivered to the mixing room by a long screw conveyor extending under the furnaces. The products of combustion are to pass into an iron pipe, extending longitudinally over the furnaces, parallel with the side of the building, which at the end of the building pitches down into a subterranean flue leading to the main flue to the chimney.

No machinery had been installed in the mixing department. It was said that a mixer of the pug-mill type would be used.

The distillation department was laid out for five furnaces, designed mainly according to the Ferraris system in so far as the superstructure is concerned, but with certain modifications, of which some important features do not commend themselves. The inferior structure is so designed that there are tunnels under the combustion chambers on each side, and no room being left for the checkers under the furnace, the latter are arranged in separate structures outside of the furnace house, but close to the latter, these being of course connected with the flues leading from the furnace and to the chimney. The flues communicating with the chimney lead into a main (5 ft. wide and 6 ft. high) parallel with the furnace house and between it and the pottery. The flue from the roasting furnace department also leads into this main which (together with its branches) is subterranean. Beyond the works, the flue turns off at an angle and leads up hill to a chimney 8 ft. in diameter and 110 ft. in height, the base of which is at considerable elevation (200 ft.) above the works. It is obvious that the flue-system is not of sufficient capacity.

In November, 1905, the distillation furnaces had been completed only to the floor line of the furnace house. The foundations were being laid for five more furnaces beyond the first battery, on the same line.

Each furnace is designed for 240 retorts (120 per side, in four rows of 30 each). The retorts are 7½ x 11 x 50 inches. The fronts of the furnace, are supported by six principal buckstaves, which divide them into five sections, each comprising 24 retorts. Hoods are to be provided for carrying off escaping fume, and there is the usual rail for working shield, etc. The capacity of these furnaces may be estimated at about 8½ tons of roasted ore per day, or say the equivalent of 10 tons of raw ore. The five furnaces will therefore have a nominal capacity of about 50 tons per 24 hours, or say 17,500 tons per annum. These estimates are based on the assumption that no serious difficulties develop because of the peculiarity of the furnace design.

The gas producer plant was only just begun in November, and evidently the plans for it had not been entirely formulated.

A lead smelting furnace for working up the residue was contemplated, but definite plans for it had not been made.

It was obvious that it would still be a long time before this plant would be completed so that operations could be begun.

It was projected to obtain clay for retort manufacture from St. Louis; coke breeze for reduction material from the coke plants in the Crow's Nest field; while the coal for fuel would of course, come from the colliery right at hand.

LEAD SMELTING IN BRITISH COLUMBIA.

Lead smelting in British Columbia will be discussed in this report only in so far as it has a bearing on the production and marketing of zinc ore. It has already been pointed out that the commercial association of the two ores is very close, which is particularly the case in British Columbia. Not only is the production of zinc ore to a large extent dependent upon the production of lead ore, because of their joint association in the mines, but also the zinc ore which is of high grade in silver is capable of treatment either by lead smelting or by zinc smelting, there being a certain dividing line which marks the superior advantage of either process.

There were in British Columbia in November, 1905, three active lead smelting plants and one inactive. The active plants were those of the

Canadian Smelting Works, at Trail; the Hall Mining and Smelting Co., at Nelson; and the Sullivan Group Mining Co., at Marysville. At Pilot Bay, on Kootenay Lake, there was an old plant, now owned by the Canadian Metal Co., which was taking steps to put it again in operation.

The Canadian Smelting Works at Trail, which have been owned by interests closely identified with the Canadian Pacific Railway, but recently have been transferred to the Consolidated Mining and Smelting Co., of Canada, are by far the most important. They comprise not only lead and copper smelting furnaces, but also a lead refinery, which is the only lead refinery in the Dominion of Canada. It is also noteworthy that it was at this plant that the electrolytic refining of crude lead was introduced on a practical scale for the first time in the world, and up to the end of 1905 it was the only electrolytic lead refinery in North America. At present similar plants are in course of construction at Newcastle-on-Tyne, England, and near Chicago, Ill.

In general the practice of lead smelting in British Columbia is similar to that of the United States. The Sullivan smelter treats the ore of a single mine. The Nelson and Trail works are custom plants. The lime-roasting of galena (Huntington-Heberlein process) has been introduced at Marysville, is being introduced at Trail, and is to be introduced at Nelson. In this respect all of the plants are strictly up-to-date. The introduction of the new system ought to be of advantage to the lead-zinc mines of the Province, as will be subsequently pointed out.

The chief difference between the practice of silver-lead smelting in British Columbia and the United States is in the character of the ores offered. In the United States there is an abundance of silicious and ferruginous ores, while lead ores are comparatively scarce, and there is seldom more than 12 per cent. of lead in the furnace charge. Consequently, lead ores are smelted at low rates, often below the actual cost, while both the deficit and the requisite profit on the whole charge is made up out of the silicious and other ores, which are referred to as the "marginal ores." In British Columbia the conditions are more or less opposite. Silicious and ferruginous ores are rather scarce, while lead ore is plentiful. Consequently, silicious ore, like that of Rossland, is smelted at a low price, while a comparatively high price is charged on lead ore, which is necessarily the marginal ore. The percentage of lead in the charge smelted in British Columbia is much higher than in the United States, and because of the invariably high zinc content of the galena ore, the zinc in the charge is also higher than in the United States. The high smelting charge on the Slocan ore has been a good deal of a grievance to the miners. A reduction is to be anticipated as the supply of other classes of ore increases, and as improvements in the smelting practice are consummated.*

Canadian Smelting Works.—The ore received at these works passes through the sampling mill, in which the samples are mechanically cut out

* Since the above was written, the rate of $15 per ton for freight and treatment, which prevailed in November, 1905, has been reduced to $12 per ton.

by Vezin samplers. The sulphide ores and matte, which are to be roasted, have been heretofore delivered to hand-raked reverberatory furnaces, 15 x 68 ft. dimensions of hearth, and to Brückner cylinders. High grade galena, however, has been smelted raw. In November, 1905, a Huntington-Heberlein plant was in process of installation. This was to comprise two Heberlein furnaces, 26 ft. in diameter, and six pots, each of nine tons capacity, the entire plant being rated nominally at 90 tons per 24 hours, which is probably somewhat more than will be attained in practice. This system of roasting should enable the high grade galena, now smelted raw, to be desulphurized economically and without material loss of silver and lead, at the same time increasing the smelting rate of the blast furnaces and reducing the matte-fall, all of which are important advantages.

The raw ores and roasted products are collected in self-discharging bins, from which the constituents of the furnace charge are drawn off into cars. No bedding is done.

There are two lead furnaces, of modern design, with closed tops. Each furnace is 44 × 144 inches at the tuyeres, and smelts 125 to 150 tons of charge per 24 hours, indicating a hearth activity of 2.84 to 3.40 per sq. ft. The percentage of ore in the charge is about 73, the remainder being flux, and the actual ore smelted per furnace is about 100 tons per 24 hours. The charge contains about 32 per cent. lead. The slag produced contains about 10 per cent. ZnO.

The refinery has capacity for about 50 tons of base bullion per day. Its general arrangement is similar to that of an electrolytic copper refinery, and the current and voltage employed are about the same. The electrolyte is a solution of hydrofluosilicate of lead. It was upon the discovery of this electrolyte by Anson G. Betts, of Troy, N.Y., that the electrolytic refining of lead became commercially practicable.

The base bullion is cast into anodes weighing about 300 lbs. each. They are now cast in flat molds, and are rough and irregular. The experiment of casting them in vertical molds is to be tried. The anodes are electrolyzed in wooden vats, coated on the inside with asphaltum. The cathodes are thin sheets of refined lead. During the electrolysis the slime adheres to the anode, from which it is scraped off daily. When the electrolysis has proceeded as far as possible the remainder of the anode, amounting in weight to about 18 per cent. of the original, is lifted out and remelted. The base bullion is comparatively clean, containing about 1 per cent. antimony. It contains comparatively little bismuth. It is especially noteworthy that it contains a little tin, the source of which has not been traced. It is possible that it comes from the Slocan ore, which would not be surprising in view of the close association of that ore with granitic rocks and the detection of tin in some of the zinc ore of the Slocan.

The electrolytically refined lead is of high grade. It is very soft and of excellent character for corroding. It contains ordinarily about 0.4 oz. of silver per ton. The silver content can be reduced to 0.2 oz., but it does not pay to go so low as that.

The anode slime is oxidized in a reverberatory furnace, wherein the antimony and copper are slagged off, the operation occupying four days. Refined doré bars are obtained from this furnace. These bars are then parted by sulphuric acid in the usual manner. An electrolytic process for the extraction of metallic antimony from the anode slime, is now under consideration. If successful, this is expected to result in the production of 800 pounds of metallic antimony per day, which will be a valuable by-product, in view of the present high price for that metal. It is one of the advantages of electrolytic lead refining that antimony and bismuth when present in base bullion can be cleanly extracted as pure metals. Except for this, it is doubtful if the process be any more economical than the process of zinc desilverization, assuming the two to be practised under equally favourable conditions. The cost of electrolytic lead refining at Trail is probably considerably more expensive than zinc-desilverization in good practice, but this is to be attributed largely to the fact that it was the first plant where the new process was introduced and the practical method of operation has had to be developed gradually, wherefore the plant as it now stands inevitably falls short of the degree of efficiency that could be attained in a new plant.

The power required in the Trail works is transmitted electrically from Bonnington Falls, a voltage of 20,000 being used on the main line, which is reduced to 550 in the works. The cost is $45 per horse-power per annum.

Hall Smelting Works. These works, at Nelson, are smaller and less efficient than the works at Trail. They were built, in 1895, to smelt the silver-copper ore of the Silver King mine, but since 1900 have been run chiefly on a silver-lead basis. They are built on a high bench of the hill-side above the West Arm of Kootenay Lake. They comprise two blast furnaces, one 42 × 96 in., the other 44 × 144 in., and when running at full capacity smelt 170 tons of charge per day. The small furnace treating 60 tons and the large one 110 tons. The percentage of ore in the charge is about $66\frac{2}{3}$, the remainder being fluxes. The works have an equipment of hand-worked and mechanical roasting furnaces, including a Merton furnace (recently installed), and the usual equipment of ore bins, sampling mill, briquetting plant, machine shop, power plant, etc.

Experiments have been made with the Savelsberg process for desulphurizing galena, and the installation of a Huntington-Heberlein plant is now contemplated.

The matte from the blast furnaces is separated in a special forehearth, the invention of Mr. Henry Harris, assistant superintendent of the works, which appears to operate efficiently. The slag overflowing from this forehearth is granulated and removed hydraulically. It is claimed that this forehearth makes cleaner slags and saves the labour of one man.

Sullivan Smelting Works. These works, at Marysville, in the East Kootenay, treat the ore of the Sullivan Group of mines, above Kimberley. It is a new plant, completed in 1905, which is of special interest as showing the second installation of the Huntington-Heberlein process in North America* and its application to a particularly difficult ore. The ore to be smelted is of high zinc content, the mechanical separation of which has heretofore defied experiment, and the problem of successful smelting has been to waste this zinc in the slag from the lead furnaces, at the same time maintaining the efficiency of the latter as lead smelters.

This smeltery is situated on a level site, the ground being mostly river sands and probably glacial drift, through which the St. Mary River has cut a channel of about 200 feet in depth; the works are built on the bluff, overlooking this river, the descent to which affords a grand slag-dump. Around the smeltery a small village has collected, now known as Marysville, which consists almost exclusively of houses for the men of the works, a few boarding houses and some stores.

Marysville is connected with the main line of the Canadian Pacific railway by a branch extending from Cranbrook to Marysville. From Marysville, the branch extends to a point near the Sullivan mines, about five miles further up the valley. This road runs three trains per week.

The ore is brought down from the Sullivan mine to the railroad at a cost of 25c. per ton, over an excellent wire tramway, 6,300 feet long, of the Riblet system, which is said to have cost only $18,000, for a capacity of about 20 tons per hour. The tramway discharges into hopper-bottom cars, which are run down to Marysville..

At the smelter, the railway cars are pushed up an elevated trestle, whence coal, coke and limestone are discharged into self-emptying bins and upon the storage floors.

The mine ore is drawn off from the hopper-bottom bins to a belt conveyor, which takes it to the crushing and sampling mill, wherein it is reduced in size to pass a 4-mesh screen (0.18 in. apertures). The mill has a breaker and three sets of rolls, together with elevating, screening and sampling machinery, the whole being so arranged that the ore passes through in a practically automatic process. Two men per shift operate the mill, which crushes about 100 tons in 10 hours. Besides the sulphide ore, the silicious ore and limestone required as fluxes are also crushed in this mill, all being reduced to 4-mesh size. One man trams the crushed material from the mill to the stock-bins.

The silicious ore is obtained from Republic, Wash.; it contains about 85% silica. The limestone comes from Fernie, B.C.; it is of excellent character, being free from magnesia, and containing only 3 per cent. silica.

The ore crushed in the mill, having been received in the storage bins, has to be shovelled up again for transportation to the mixing floor, which requires two men per shift of 12 hours. On the mixing floor, the charges are weighed out, the usual proportion being 100 parts of sulphide ore, 10 parts of silicious ore, and 7 parts of limestone, but the proportions are varied according to the ore. The above is about the average when the sulphide ore

* The first was at Mapimi, Mexico.

contains 27 per cent. lead, 12 per cent. zinc, and 18 per cent. iron. The mixing is done in two shifts of 12 hours each, one man per shift, who shovels the charges upon a belt conveyor, which elevates them to hoppers above the roasting furnaces. The mingling of the limestone, silicious and sulphide ores as they go on the belt, together with the further mixing they receive in the hoppers above the furnaces and the rabbling in the furnaces themselves, is found to be ample, and to give a thorough mixture for the subsequent reactions. For the latter, a thorough mixture is quite essential.

The roasting furnaces are of the Heberlein patent, the latter embodying a circular, revolving hearth, similar to that of the old Brunton and the more recent Godfrey furnace. The furnaces at Marysville are two in number, 26-feet in diameter of hearth, revolving at four revolutions per minute. The hearth is of steel plate and I-beam construction. On top of the plates, a simple covering of twelve inches of crushed coke, forms the hearth of the furnaces; this coke is merely fed in through the hoppers and distributed by means of the rabbles; then the ore is fed in and the furnace placed in commission. These furnaces have one fire box with open ash pit, and immediately opposite it the fixed rabble frame suspended from the roof of the furnace. The ore is fed at the center of the hearth and is gradually removed in spiral-like segments by the fixed rabbles to the perimeter and discharged. On either side of the central ore feed a smoke pipe takes the products of combustion from the furnace. It will be seen that the flame travels from the fire bridge to about the center of the furnace, and thence escapes through the pipes previously mentioned. The power necessary to run a furnace is stated to be 1.5 horse. The ore discharged from the furnace smells strongly of sulphur and is of a dull red heat. It is immediately sprinkled with water. There are two hoppers provided in which the ore can be moistened and a small central one in which hot calcines are held for the purpose of starting operations in the H-H pots. Each furnace treats 45 charges per day consisting of 1,500 pounds of Sullivan ore, 150 pounds of limestone and 135 pounds of silicious ore—a total of 1785 pounds per charge, or say 80,325 pounds per day, wherefore the two roasting furnaces treat about 80 tons of charge per day. The consumption of coal was said to be about 400 lb. per charge. The vault underneath these furnaces is white-washed, the machinery is free from dust and the temperature below the hearth quite comfortable. The furnaces stand rather high, so that the vault is large and the machinery is easy of access. A sand lute is used on the outer edge of the revolving hearth to prevent air coming up from below or dust falling into the vault. The ore fed to the furnaces runs about 25% sulphur, the roasted product 10 to 12% sulphur. One man per shift of eight hours attends to the two furnaces.

The roasted ore after being sprayed with water is trammed to a platform elevator and discharged into hoppers above the H-H pots. Two men per shift are required for this work. They use an ordinary tram-car, holding about 2000 lbs. of ore. There are six H-H pots in use at this smelter, each nine feet in diameter by about four feet deep, supported on trunnions about 12 ft. above the ground. A six-inch blast pipe enters at the bottom, and is dispersed by a 72-inch perforated grating, $\frac{3}{4}$ in. thick, drilled with $\frac{3}{4}$ in. holes, 2 in. centers. A worm and wheel with hand attachment is provided for dumping the pots, on completion of the roasting operation. The pots are covered with conical hoods, provided with slots through which the charging and tamping can be attended to. The six pots are arranged in two parallel lines of three each, with the operating platform between them. One man per shift attends two pots. His principal work is to distribute the charge in the pot as required and tamp up such blow holes as develop. If the latter be not carefully closed, the charge becomes excessively hot around them, leading to loss of lead and silver by volatilization, and the desulphurization of the charge is uneven. The charges average eight tons, although some have reached as high as twelve tons. The time for working a charge varies from twelve to eighteen hours, but on the average is about sixteen hours. The charge used to be started by igniting a little wood on the grate of the pot and then throwing in some hot calcines from the roasting furnace, but there were some irregularities due to this procedure, and saw-dust is now employed instead of wood. The blast is turned on at low pressure, about 2 ozs., and a portion of the charge is then introduced and spread evenly in the pot, filling it to about one third its depth (one-third of the depth from grate to rim). After an hour or so, another portion is dropped in filling the pot about two thirds, and after two or three hours more the pot is charged even full, and the blowing is continued to the end of the sixteen hours. As the pot is filled, the blast pressure is gradually increased, attaining 12 to 16 oz. at the end of the operation.

After the first layer of four to six inches of hot calcines is placed on the perforated bottom of the pots, and the 2-ounce blast has been turned on, when this layer of ore becomes red hot and glowing, the first layer of damp ore is immediately added from the hoppers above, and tamped down around the sides of the pot where the air has a tendency to break through. Blow holes are carefully looked out for by the attendant, who immediately pokes them down with a light bar to the end that the air may so far as possible be equally distributed throughout the charge. In the course of time the layer becomes red hot, and looks practically the same as glowing coals in a grate. At this period the next layer will be added from the hoppers above, and so on, the layers of the moist ore being added as fast as the last one is sufficiently oxidized.

At the conclusion of the process the pot is revolved, and the charge of semi-fused and well roasted material drops on an inverted slag pot and is broken into several pieces. This roasted material contains from 3 to 5 per cent. of sulphur, equally divided between sulphate and sulphide. Some charges go as low as 1.5 per cent. total sulphur. The roughly broken and glowing charge is next sprayed with water, and when cool is broken up by a hammer and bars to suitable size for the blast furnaces. Six men per shift are occupied in breaking these fused charges, the coarse material from which is loaded up by hand, the finer material on coke forks, and the next size is shovelled over a screen. The screenings (amounting to eight per cent. of the original charge) are returned to the pots for further roasting.

The charge dumped from the converter is bright red inside, as appears when it breaks open, and liquid slag in small amount runs out of cavities. Thoroughly desulphurized material has the appearance of slag-roasted galena. It shows prills of metallic lead.

The desulphurized and broken ore is shovelled into cars, which are conveyed mechanically up an incline to the blast furnace charging floor. There is no further making up of the charge, the necessary flux having been added on the mixing floor before the ore went to the roasting furnaces.

There are two blast furnaces. One is 42 × 140 in. at the tuyeres; the other, 42 × 128 in. Each is 17 ft. in height from tuyeres to charge floor, and each has seven tuyeres per side. At the time of my visit only the former was in operation, the capacity of this being in excess of that of the six H-H roasting pots. This furnace was said to be smelting 120 tons of charge per 24 hours, besides 30 tons of slag, which would be at the rate of 2.6 tons per sq. ft. of hearth area per 24 hours for the ore only, or about 3.2 tons, including the slag. The charge was smelted with 11 per cent. of coke. The blast pressure was 18 oz. The blower house has two Connersville blowers, each of 68 cu. ft. capacity per revolution. One blower, at 80 r. p. m., ran the 42 × 140 in. furnace and the six H-H pots.

The sulphide ore smelted in November, 1905, was of approximately the following composition: 7% SiO_2; 18% Fe; 12% Zn; 3% CaO; 27% Pb; and 11 oz. silver per ton. The slag produced assayed 20% SiO_2, 26% Fe; 11% CaO; 4% MgO; 4% Al_2O_3 and 15% Zn. The slag was tapped directly from the furnace into Nesmith two-bowl cars, which were conveyed directly to the dump, only the shells being returned. These showed no matte, whatever, and it was reported that the furnace made no matte, except a lead sulphide product (to be referred to further on), the sulphur remaining in the charge from the H-H pots being almost completely oxidized in the blast furnace. The slag was fairly fluid, and lacked some of the characteristics of a slag high in zinc, settling quietly in the pot, without any crater-like eruptions showing a zinc flame. The slag assayed less than 1% lead (by fire assay) and less than 1 oz. silver.

Practically, the only trouble experienced in the smelting has been the sulphide of lead which accumulates in the well of the furnace, causing considerable inconvenience. This sulphide also rises in the siphon and can be seen on all the bars of bullion. It is said not to give any trouble at the refinery or to interfere with the sampling of the pigs, but it interrupts the furnace campaign and forms a vexation between-product that has to be reworked.

The smelter machinery is operated by water power, having a head of 175 feet. One hundred and fifty horse power is developed from three 6-

foot Pelton wheels, two of which operate the plant and one is held in reserv The maximum that can be developed from this water power is said to be 500 horse power. An auxiliary steam plant is provided to operate the blowers in case of accident to the water power. From the Pelton wheels situated on the St. Mary River, power is distributed by wire rope to the smelter and through the yard as required.

The smoke from the furnaces and H-H pots passes into a concrete dust-chamber, which terminates with a self supporting steel chimney, 8 ft. in diameter and 150 ft. in height.

The total number of men employed in the works is 140. Common labour is paid 25 cents per hour. Blast furnace chargers and tappers receive $4 per shift. Roasting furnace men get $3.50.

The Huntington & Heberlein process, through its efficient desulphurization of the ore and delivery of the ore in excellent physical condition for the blast furnace, has enabled the rapid smelting for lead of a sulphide ore high in zinc, but the zinc is wasted. Zinc can be recovered from such slag by mixing it with fine coal and heating to a high temperature. This was the basic idea of the Fry process, which was tried on a large scale at works on the Manchester Ship Canal, England, in 1898 and 1899, and recently experiments of similar character have been made at Monteponi, Sardinia, but it is doubtful if means will ever be found to extract profitably the zinc from slags containing only 12 per cent. of that metal, which is the average at Marysville; least of all in a region so remote as British Columbia. It might be possible to make a zinc-lead pigment from this ore, as is done at Cañon City, Colo., by the Bartlett process, but the rentability of such a process in British Columbia would be dubious. The extraction of zinc from such ore as that of the Sullivan mine will evidently be dependent upon the separation of the zinc mineral before smelting, but so far as known there is no process which will effect any commercial separation, and of known zinc ores the Sullivan exemplifies the most difficult problem.

Other Smelting Works. Small lead smelting plants have been erected in British Columbia at Revelstoke, and Pilot Bay, but they have not been in regular operation and their production has been insignificant. The first lead smelted in the Province is said to have come from Revelstoke. The Pilot Bay smelter has been idle since 1896, but the Canadian Metal Company, which now owns it, plans to resume smelting there.

SMELTING RATES ON SILVER-LEAD ORE.

The market in British Columbia for silver-lead ore, and other ores that can be smelted therewith, is made by the Hall Mining and Smelting Co. and the Canadian Smelting Works, which are supposed to act in harmony. Since Feb., 1906, the standard rates on various classes of ore have been as follows:—

Galena Ore.—Pay for 95 per cent. of the silver at the New York quotation, and for 90 per cent. of the lead (by fire assay) at the London quotation for soft Spanish, less one cent per pound. Deduct $12 per 2,000 lb. of ore for freight and treatment, plus 50 cents per unit for zinc in excess of 10 units. The freight and treatment rate of $12 obtains on ore containing 50 per cent., or more, of lead. Below that tenor there is a deduction of 20 cents per unit of lead. Thus an ore containing 20 per cent. of lead would stand a freight and treatment charge of $12—(30 × $0.20) =$6. A zinc tenor of 18% in the same ore would increase the freight and treatment to (8×0.50) +$6.=$10. Lead is not paid for at all unless it be in excess of 5 per cent. The freight rate from Slocan railway points to Nelson is $1.50 per ton; to Trail, it is $1.50 by the Canadian Pacific and $2.50 by the Kaslo & Slocan. Consequently the smelter receives $12 minus $1.50 @ $2.50 for his treatment charge.*

Blende Ore.—On ore high in zinc (40% or over) the smelter pays for 95% of the silver at the New York quotation, pays nothing for lead, and deducts $10 for freight and treatment, plus 50 cents per unit for the zinc in excess of 10 units. Ore assaying 40 per cent. zinc would consequently stand a freight and treatment charge of $25 per ton. Some of the ore high in silver has been shipped under such terms.

Silicious Ore.—On silicious, or dry ores, the freight and treatment rate is $6.50 to $10 per ton, depending upon the analysis of the ore and its situation, terms for silver, zinc penalty, etc., being as above.

Iron Ore.—Magnetite is paid for at the rate of 5c. per unit of excess of iron and lime over silica, together with allowance for silver at the regular rate, less the cost of freight to smelter, and no treatment charge.

General.—It is to be remarked that the silver-lead smelters of British Columbia operate under decidedly less favourable conditions than those of the United States. The supply of ore is less abundant, and of less diverse and suitable character. The result of this is that of the charge smelted only 66⅔ to 75 per cent. is ore (the remainder being barren flux, which costs just as much to smelt per ton as ore does) against an average of 80 per cent. in good practice in the United States. A larger proportion of the ore is lead sulphide, which must be smelted raw, in order to avoid the loss of silver and lead in roasting, and thereby increases the matte-fall, and consequently the quantity of material that must be reworked. The lead sulphide ore is moreover zinky, which together with the other zinkiferous ores, causes the smelters to run on a more zinkiferous, and consequently less advantageous slag, their practice being about 10 per cent. zinc oxide in the slag against an average of 6 per cent. in the United States.

The wages for labour in British Columbia also are higher than in the United States. At Trail, the furnacemen are paid $4 for 8 hours work; the tappers, $3; and common labour, $2.50. The same rates obtain at Nelson.

In the matter of coke the British Columbia smelters have a little advantage, since the cost of that fuel to them is only about $5 per ton, against

* Previous to February, 1906, the freight and treatment rate was $15 per ton.

$5.25 @ $6.50 at Denver and Pueblo, and higher figures in Utah and Montana. The smelters of Denver and Pueblo, however, obtain coal at $2 per ton, which is the cost at the collieries in the Crow's Nest field. Power is somewhat cheaper to the Canadian smelters than to the American, inasmuch as the former are able to contract it by electrical transmission at $45 per h.p. per annum, while the cost to the latter, even when they have a good steam plant, is likely to figure out to $75, and frequently is higher. It is to be remarked, however, that the cost of $45 does not include interest and depreciation on the electrical equipment of the works.

The cost of lead smelting in the large plants of the American Smelting and Refining Co. is $3.20 to $4 per ton of charge, which corresponds to about $4 to $5 per ton of ore, including the roasting of the proportion of ore which requires roasting. The latter is about 40 per cent. of the ore smelted, and the cost per ton will fall between $2 and $2.50. As the percentage of ore requiring roasting increases, and as the percentage of ore in the blast furnace charge decreases, the cost of smelting increases.

It is not my intention to enter into a critical discussion of the cost of silver-lead smelting in British Columbia further than to indicate that for the reasons, which I have pointed out above, it is not to be expected that smelting rates on lead ore can ever be so low as they are in the United States, or at least not until equal conditions as to ore supply and labour, etc., are attained, and this has an important bearing on the production of zinc ore, which as I have discussed previously is in most of the mines of British Columbia intimately interwoven with the production of lead ore.

It may be remarked further that the deduction of one cent per pound from the London price for lead is not a clear gain to the smelter, who has to pay out of that the freight to market, cost of refining and expense of selling. On Colorado bullion shipped to Perth Amboy, N.J., for refining, these charges come to $14 per ton, or 0.70 cent per pound, the railway freight rate being 0.32 c. per lb., but if British Columbia bullion had to go to an Eastern market, the freight rate would of course be considerably higher, because of the greater distance. Refiners pay for 98% of the lead in the bullion, pay for the silver at the New York price less one cent per ounce, and for the gold at $20 per oz. The difference between the lead and silver percentages that are paid for gives some margin to the ore smelter, if he does good work.

MECHANICAL CONCENTRATION OF ZINC ORES.

The Belgian and Rhenish methods of zinc smelting, which are the ones generally in use outside of Upper Silesia, require a comparatively high grade ore, i.e., ore that contains at least 40% Zn., after calcination or roasting. A roasted blende which assays 50% Zn. must have contained 45% Zn. before roasting if 10% in weight were lost in that operation. A sulphide ore assaying 45% Zn. contains 67% blende, even if the latter bears no isomorphous iron or cadmium sulphides. There are few blendes which are quite free from iron monosulphide, chemically or mineralogically combined with the zinc, and the foregoing calculation shows the rather high degree of purity that is possessed by a raw sulphide ore assaying 45% Zn. European zinc smelters do not require this grade to be exceeded, and American smelters now buy and work ores of lower grade than they would think of a few years ago. Nevertheless it is desirable from all standpoints that the miner shall make as high a grade of ore as possible, especially when the ore must inevitably stand high transportation charges, as is the case with ore produced in British Columbia. The ability to produce blende concentrate assaying 60% Zn., which is attained in the Joplin district and in the Wisconsin field of the United States is exceptional. Those ores are coarsely crystalline, free breaking, contain but little combined iron, and are associated with a light gangue. It may be assumed that an assay of 60% Zn. indicates a tenor of at least 92% of "mineral," the remainder being chiefly silica. It is obvious that so high a degree of concentration cannot be effected mechanically without extraordinary losses in the tailings except in the case of favourable kinds of ore. It can never be economically attained in British Columbia, because of the higher percentage of combined iron in the blende.

The mechanical concentration or dressing of zinc ore is done practically by manual selection, by gravity separation, by electrostatic separation, by flotation, or by magnetic separation, two or more of these systems often being combined. In concentrating zinc ore it has to be kept in view not only to enrich the ore by removal of the gangue, which is composed usually of light minerals, but also to separate the heavy minerals which may be injurious in the smelting process. For example, all of the lead, iron and manganese minerals are particularly objectionable and ought to be eliminated as completely as can be done economically. Even when this be done blende ore is apt to contain a large percentage of iron on account of monosulphide, FeS, combined isomorphously with the zinc sulphide; thus the shining, black blende of Freiberg, Saxony, sometimes contains as much as 30% Fe; a specimen of similar appearance from Mexico analyzed by me gave 10% Fe. A sample from the Blue Bell mine, opposite Ainsworth, showed 13.5 % Fe. The blende of the Slocan appears to contain 5 to 6%. Fe. This imposes a limit beyond which no kind of mechanical concentration can possibly go.

The dressing of ordinary zinc ores, which are apt to be mixtures of blende and galena with such gangue minerals as quartz and calcite, or calamine with similar gangue, does not offer especial difficulty, owing to the great difference in the specific gravities of the component minerals. The presence of pyrite, marcasite, barite, or siderite complicates matters because those minerals are about of the same specific gravity as blende, smithsonite, hydrozinkite and hemimorphite, (hydrous zinc silicate), and in such cases magnetic or electrostatic separation or flotation must be resorted to as an accessory process. Improvements in the art of ordinary concentration, especially fine grinding and washing on modern shaking tables of the Wilfley type, combined with the new special processes, have enabled many complex ores to be treated successfully, which formerly could not be worked. The mixed ores of Leadville, Colo., and Broken Hill, N.S.W. are important examples.

Manual Selection or Hand Sorting.

When the mineral is coarsely mixed with the gangue it is feasible to separate it to a considerable extent by means of hand sorting. This is an ancient and simple process, which too often receives only slight attention or is considered lightly, being disregarded in the general desire to do everything by machine, without real reflection upon the results. It is of peculiar importance in the handling of the ore of the Slocan, and too much emphasis cannot be put upon the advantage to be derived from its general application.

Hand sorting is generally practised to a certain degree as a part of the mining of the ore, in which high grade ore may sometimes be broken out of the lode separately; sometimes pieces of worthless waste can be picked out of mixed ore and left underground to fill old stopes, etc. As to whether the hand sorting process shall be carried further on the surface depends on the local conditions, especially the losses experienced in mechanical dressing, the cost of dressing and the cost of sorting, of which the loss in dressing is likely to be the most important consideration. If pure mineral be broken, a portion of it will be converted into fines (the percentage depending upon the brittleness of the mineral, the size to which it is broken, and the method of breaking) and in washing with water a good deal of such fines will escape settling, no matter how perfect and painstaking be the process of settling. It is therefore an axiom in mechanical dressing to avoid breaking the valuable mineral any more than is absolutely necessary. These principles are of peculiar application in the Slocan, where the galena of the ore is singularly rich in silver.

The above conditions are met most completely by the process of hand sorting, wherein two classes of ore will usually be made, viz.:—I, clean mineral, divided into (a) blende and (b) galena; and II, mixed mineral and gangue, which is sent to the dressing works. If the cost of dressing be high it is sometimes economical to sort out a third class, III, clean gangue, or waste, thereby saving the expense of crushing and washing worthless material and increasing the capacity of the mill. Under other conditions it may be cheaper to let the waste go through the mill. This can be determined only by a study of the conditions in each particular case. In spite of the high wages for labour, I have no doubt that it would be profitable in the Slocan to pick out waste, considering as waste not only the slate and quartz gangue, but also the siderite, which when clean contains little or no silver.

Hand sorting of ore involves two processes:—I, breaking; II, selection. The breaking is preferably done mechanically by jaw crushers; otherwise it must be done manually with the aid of hammers. The former method is the cheaper, but it produces the more fines; while breaking with hammers is not excessively costly if properly done; and if the ore must not be broken to a very small size. The size to which the ore must be broken depends obviously on what is required to free the pieces of pure mineral for the maximum; and what can be picked over economically for the minimum; the choice between the two extremes will depend naturally upon the grade which it is desirable to obtain for the sorted mineral.

It is not to be doubted that at some of the small mines of British Columbia, which are unprovided with a mill or lack the means to instal a crusher, it would be advantageous to break the ore by hand, rather than not to employ hand sorting at all.

The cost of breaking ore by hand, down to 2 in. size, with labour at 37.5c. per hour for 25 tons per day, will be 66¾c. per ton, a cost which is offset by a very small increase in the extraction of mineral. Rock breaking can be done at that figure, however, only when the men stand up to the work and use the right kind of hammer, which should have a head 6 in. long, weighing about 2.75 lb. (being forged from 1½-in. octagonal bar of the best steel) and a long, springy handle of oak, ash, hickory, or hornbeam.

The ore having been broken, either by machine or by hand, it must be passed over a grizzly or over a shaking screen, or through a revolving screen, to remove the fines (which will be sent directly to the dressing floors) before going to the sorters, and it must be thoroughly cleaned of adhering dirt by means of a water spray. Good sorting cannot be done unless the mineral be bright and clean. In sorting ore by hand, moreover, the success of the operation depends largely upon the convenience of the manner in which the ore is presented to the pickers. This may be done by stationing the latter along a bench on which they may draw from pockets the ore to be sorted; better by discharging the ore on a large, circular or annular table of wood or iron, revolved slowly, around the periphery of which the pickers stand; or better still, by discharging it on a travelling belt from which the pickers can select it.

An annular, revolving picking table, as built by the Allis-Chalmers Company is shown in the accompanying engravings (Figs. 1 and 2). The broken ore is led on the table by a chute, and spreading out is carried by the revolution of the table until, meeting an inclined, stationary scraper, it is swept off into a chute, which delivers it into cars, or a continuous conveyor, to transport it to the next operation. Around the table, men, or boys, stand, who pick out mineral and gangue from the slowly moving layer of ore. The table itself is made of punched iron or steel plate.

Experience has shown that it is easiest for pickers to throw the sorted material in front of them, and this arrangement is easily made with tables of annular form, in connection with which a conical surface may be arranged inside the ring, around the vertical axis, with radial partitions to separate different classes of ore which will slide down the cone into proper receptacles. With stationary tables, and endless belts where picking is done from both sides, it can easily be arranged for the pickers to throw the sorted ore into chutes conveniently placed in front of them, as shown in the accompanying illustration. (Fig. 3).

Endless belt picking tables require no detailed description beyond the statement that they may be made of linked tablets of wood or iron, or billets of wood, or ordinary rubber belting. The Robins system of belt conveyors is easily adapted to this purpose, and the Robins Conveying Belt Co. makes a special picking belt, which is very good. It is heavy and wide, commonly 32 to 36 in., and is supported on idlers which are so shaped as to give the belt a broad, flat surface at the center, with narrow, very slightly raised sides. It is made to travel at speeds of 30 to 60 ft. per minute. Owing to its elasticity the belt will withstand spalling of the ore directly upon its surface. Rubber belts have the advantage over other kinds that there are no links to wear and no crevices wherein pieces of ore can jam.

Mr. Edwin H. Messiter, metallurgical engineer of the Robins Conveying Belt Co., of New York, communicated to me, in response to my request, the following notes on the installation of picking belts:—

"In the treatment of metalliferous ores, sorting conveyors, also called picking belts are used for a preliminary hand-sorting of the ore, as received from the mine, into waste and treatable ore, or into two, or more, classes which are to be treated separately. The ore is usually fed over a grizzly so as to fall on a slightly troughed belt conveyor in a wide layer of such depth that all of the lumps are visible, the purpose of the grizzly being to remove the material which is too small for hand-sorting.

"The material which occurs in greatest quantity is allowed to remain on the belt until the delivery pulley is reached, whence it usually drops into a crusher or directly upon a belt conveyor. The second conveyor is frequently placed under the picking belt, parallel to it, and extended far enough back to receive the fine ore from the grizzly as well as the discharge from the picking belt. The width of belt is usually from 24 to 36 in., 32 and 36 in. being the most common widths. A speed of 35 ft. per min. is satisfactory in most cases, though where material is coarse and easily distinguished the speed may be materially increased, while for very close work it should be reduced. The capacity at a given speed will depend on the size of the ore. The smaller the ore is crushed, the thinner the layer on the belt must be in order that all of the lumps may be exposed to view. With 3-in. lumps a 36-in. picking conveyor running 35 ft. per min. will have a capacity of 35 tons per hour of ore weighing 100 lb. per cu. ft. Under the same conditions a 24-in. conveyor will have a capacity of about 20 tons per hour. For a given width of sorting belt, the capacity may be taken within reasonable limits as directly proportional to the speed and to the average size of lumps in the material carried; e.g., the above conveyor, if run at a speed of 42 ft. per min., instead of 35 ft. per min., would have a capacity of 42 tons per hour instead of 35 tons per hour."

To secure the maximum efficiency in hand sorting, it is necessary to have good supervision (since any system of contract work is difficult to carry out), good light, and all arrangements which may increase the convenience of the pickers.

The cost of culling mineral is largely dependent upon the character of the ore, but it is seldom much in excess of the cost of milling in the general practice of mills of small or moderate size. Under ordinary circumstances, and especially in the case of the Slocan ores, culling is to be advised as a step in the milling process, where all the ore from the mine, having been broken by a crusher to the size determined for the next machine, should pass over a grizzly or through a trommel from which the coarse material would go to the picking table and the rejected stuff from the latter to the next crushing machine. In this case there is no extra cost for crushing, and only the cost of culling, minus the cost of milling the mineral picked out, would have to be considered against the increased saving of mineral. For example, if the cost of milling be 50c. per ton of crude ore and 16 tons be concentrated into one, the cost per ton of concentrates is $8; leaving out of account the question of losses in treatment (very important) and cost of repairs on picking tables, interest, amortization, etc. (comparatively unimportant), it would be an equal thing to produce a ton of culled mineral of the same grade at a cost of $8, insofar as operation only is concerned. Having regard to the recovery of mineral, instead of the 75% which milling may yield (this being probably a maximum), the hand sorting will yield 100%. If the mineral be worth $75 per ton, consequently, instead of one ton obtained by milling, there is 1⅓ ton obtained by culling, or a yield of $100 instead of $75. The gain is therefore $25 + $8 = $33, from which is to be deducted, of course, the cost of culling. According to Richards (*Ore Dressing*, I, p. 488) a man is able to pick from a moving belt a little more than 4.8 tons of galena of 1 in. size in 10 hours, which would make the cost 20½c. per ton with labour at 10c. per hour. With labour at 37½c. per hour the cost would be 76c. per ton. The cost of picking blende on the same basis would be $\frac{74}{40}$ times as much, or $1.40, because of its lower specific gravity. These estimates are established on a theoretical basis, or rather on an experimental basis, and fail to take into account all of the varying conditions. Thus, a man can pick more galena (or blende) from a stream of ore which carries a large proportion of the mineral in clean, free and unmistakeable pieces, than he can from a stream of ore carrying only a small proportion of pieces and these of doubtful classification. Richards' table is, however, valuable in showing how rapidly the quantity of material that can be picked diminishes and the cost per ton increases as the size of the individual particle diminishes. In other words it is much cheaper to pick mineral of 2 in. size than 1 in. size.

A few figures from practice will, however, demonstrate that hand-picking is an inexpensive process even when compared with the cost of milling per ton of crude ore, and often astonishingly inexpensive when compared with the cost of milling per ton of concentrate produced. It is extensively practised on the Rand for removing waste from the comparatively low grade gold ore of that field, although the cost of stamp milling is only about $1.10 per ton. From 10 to 30 per cent. of the ore is picked out at a cost of 14c. per ton of *material picked*, this being done with native labour at 50c. per day. There would obviously be a large saving, even if the wages for labour were $3 @ $3.50 per day. It is to be remarked that this result is accomplished on material of low specific gravity (probably not more than 2.8). It is to be remarked further that the work is performed with such efficiency—in spite of the fact that there is far less distinctiveness between ore and waste than there is between galena (or blende) and gangue minerals—that the waste culled by hand at certain mines assays less in gold than the final tailings from the cyanide plant.

Hand sorting is extensively practised at zinc mines in Europe, including some which have a galena-blende-siderite mixture, such as occurs in the Slocan. It is also practised to a considerable extent in the United States, among other places at Leadville, Colo., where a large part of the zinc ore is prepared for market in this manner. In picking ore which assays about 25% zinc at the Moyer mine, one man is able to produce in a shift of nine hours about 10 tons of ore assaying 31% Zn., making the cost of the latter about 30 cents per ton. In Wisconsin and in Utah, hand sorting is an essential feature in the dressing of zinc ores. At the Square Deal mine in Wisconsin, ore passed over a ⅝-in. grizzly is delivered to a picking belt, where one boy, paid $1.50, picks out 20 tons of waste rock per shift, making the cost 7·5c. per ton, whereas the cost if it were run through the mill would be upward of 30c. per ton.

It may be safely reckoned that with proper picking facilities, and ore crushed to 1½ in. size, with wages at 37½c. per hour, an ore yielding 6% of lead mineral and 12% of zinc mineral can be culled for the two minerals at an average cost of 66 cents per ton of mineral produced.

This important subject has been discussed by Mr. Argall, in his report with whose views I am thoroughly in accord,

Separation of Minerals of Nearly Equal Specific Gravity.

The separation of minerals by ordinary milling depends fundamentally upon their difference in specific gravity. When there is a considerable difference, as for instance that between galena and blende, or between blende and quartz, a nearly complete separation can be made, depending more or less upon the fineness to which the ore must be crushed to free its component minerals. When, however, there are minerals in the ore of equal, or nearly equal specific gravity, ordinary milling fails inevitably to make a separation. Careful milling may enable a partial separation to be made between pyrite (sp. grav. 5) and blende (sp. grav. 4) but, when the difference is less than one unit plus or minus 4, practically nothing can be hoped for in this way. The following list will show the minerals commonly associated with zinc blende which fall within this range:—

Blende	3.90 to 4.10
Chalcopyrite	4.10 to 4.30
Fluorite	3.10 to 3.20
Barite	4.30 to 4.70
Siderite	3.70 to 3.90
Marcasite	4.65 to 4.90
Garnet	3.50 to 4.30
Rhodonite	3.50 to 3.70

There are occurrences of blende in various parts of the world, where separation difficulty is caused by the presence of one or another of the above mentioned minerals. The economic separation of such ores has been a problem for many years. So far as it can be practised hand-picking is a solution, because it is in no way dependent upon gravity, but hand-picking although a useful aid in many cases, can never effect more than the separation of a small proportion of the mineral constituents, being quite inapplicable on pieces smaller than 0.5 in. diameter. Mechanical separation must consequently be resorted to. It was quite obvious, from the outset that no single process would be applicable to all classes of these ores. Among those which have been tried the following may be mentioned:—

1.—*Roasting.*—To alter the physical character of one of the minerals, permitting separation then to be made by jigging, e.g., the blende-marcasite mixture of Wisconsin, which was treated successfully in this way at one mill.

2. *Sifting.*—Taking advantage of some property, such as decrepitation, which would reduce the minerals to different degrees of fineness, thus enabling a separation to be made by screening. For example, blende has been separated from pyrite, at Lintorf, Rhenish Prussia, by crushing in a Vapart mill under conditions so adjusted that the blende particles would break, while the pyrites would not. Blende and siderite have been separated at Oberlahnstein, on the Rhine, Germany, by heating to redness, and throwing into water, whereby the siderite was thoroughly disintegrated into small particles, which could be removed from the blende by sifting. The inefficiency of such processes is apparent at first glance, and it is no cause for surprise that they have failed to find any general application.

3. *Magnetic Separation.*—This was early applied to the separation of blende and siderite, calamine and limonite, blende and pyrite, and it has been used practically for many years. The modern development of this process is due, however, to the attention attracted to it by the remarkable success of the Wetherill separator, which was introduced about 10 years ago.

I think it may be fairly said, indeed, that it was the invention of the Wetherill separator, and the extraordinary separations of zinc ore which it was able to accomplish, that gave a fresh impetus to the investigation of the problem of mixed ores, and thus led to the invention of electrostatic separation, and finally of flotation. At the present time there are three special processes, broadly classified as magnetic, electrostatic and flotation separation, which are practically applied, on a large scale, to the treatmnet of mixed sulphide ores, and the tonnage of zinc ore annually produced by them is large. They are not universally applicable. Some ores, as for example certain blendes associated with barite or fluorite, defy separation by any of them, but it is to be remarked that their total history is less than 10 years, that they have already augmented the world's supply of zinc ore by an immense tonnage at a time when it was badly needed, and finally that improvements are being made every year of a character which indicates that what are still unsolved problems may cease to be so in the near future. Before entering into a discussion of these processes, it is advisable to outline the particular problems which are to the front in the zinc industry of British Columbia.

The Mixed Ores of British Columbia.

The mixed ores of British Columbia may be classified broadly under two heads, viz.:—

1. Blende-pyrite mill concentrate.
2. Blende-siderite mill concentrate.

These products are similar to those which are made in many parts of the world, the blende-pyrite mixture being a common occurrence; indeed, the commonest of all the mixed zinc ores. This separation is the problem at Leadville, Colo., in Wisconsin (where, however, the iron sulphide is marcasite, instead of pyrite), in the Joplin district, where there is considerable blende associated with pyrite, and at many other places in the United States.

The blende-siderite mixture is of less common occurrence but it is, or has been, an important problem in Germany. In the United States there is only one group of mines, this being in the Wood River district of Idaho, now worked for zinc, in which siderite is an important accessory mineral.

Really the only peculiar characteristic of the Slocan ore is its high silver content, which, however, is noteworthy only in the ores of a comparatively few mines. It appears to be carried by freibergite, which in turn is carried by galena, the blende owing its richness in silver to the galena that is associated with it. To some extent, freibergite may be directly associated with the blende, which is by no means an uncommon occurrence. In so far as the silver is concerned, the desideratum is to keep it as much as possible with the galena concentrate, and as little as possible with the blende concentrate. Beyond that it is merely a question of marketing the two ores The siderite of these ores, contrary to some belief, is free from chemically combined zinc. The fundamental problems in British Columbia are, therefore, the separation of blende-pyrite and blende-siderite.

MAGNETIC SEPARATION.

The separation of blende from other minerals, is accomplished with great success by the aid of magnetism; since the introduction of the Wetherill separator this has been done even with minerals which display no marked magnetic properties and cannot be converted into magnetic forms; for example, certain kinds of blende can be separated thus from both pyrite and galena. The mixed sulphide ore of Broken Hill, N.S.W., and of Leadville, Colo., is treated successfully in that manner.

Iron sulphide, carbonate and sesquioxide may under certain conditions be converted into the magnetic oxide, Fe_3O_4, in which form it is easily attracted by the magnet and thereby may be separated from compounds of zinc, all of which are non-magnetic, or at least only feebly magnetic. Processes depending upon that principle were first applied many years ago, perhaps as early as 1855. Similarly, iron bisulphide may be changed into a magnetic sulphide. The conversion of non-magnetic iron minerals into the magnetic forms requires considerable care and many failures have been due to ignorance of the precise conditions.

The reduction of iron bisulphide to the magnetic subsulphide is effected by moderately heating pyrite, when the change expressed by the following equation takes place:—

$$3\,FeS_2 = Fe_3S_4 + 2\,S.$$

G. M. Gouyard of Denver, Colo., has published* results of experiments showing that in roasting a mixture of pyrite, blende and galena for magnetic sulphide of iron the magnetic concentrates run lower in lead and zinc than when magnetic oxide is produced.

The production of magnetic oxide of iron, Fe_3O_4, is effected by carbon according to the following reactions:—

$$6\,Fe_2O_3 + C = 4\,Fe_3O_4 + CO_2$$

or the ferric oxide may be reduced first to ferrous oxide, and the latter converted to magnetic oxide by combination with an atom of oxygen from the air, which is known to take place when FeO is heated to redness in the air, the reactions being expressed thus:†—

$$2\,Fe_2O_3 + C = 4\,FeO + CO_2$$
$$3\,FeO + O = Fe_3O_4$$

When siderite is calcined it is decomposed according to the reaction,

$$FeCO_3 = FeO + CO_2$$

and the molecule of FeO is subsequently converted to Fe_3O_4 by taking oxygen from the air. The roasted carbonate is always strongly magnetic, but in execution of the process the temperature must be regulated carefully to avoid sintering the ore, which because of the fusibility of ferrous oxide and silica may easily happen. According to Le Chatelier ‡ the decomposition of ferrous carbonate takes place at 800°C.

* *Proceedings*, Colorado Scientific Society.

† These reactions are expressed merely as typical, the reduction not being necessarily effected by carbon, but perhaps by carbon monoxide or carbon monoxide and carbon together.

‡ *Thonind. Ztg.*, 1886, p. 429.

The bisulphide of iron may be changed into the magnetic oxide if it be roasted carefully at dull red heat. Practically, however, it is difficult to obtain all of the iron in that form and it is generally necessary after the sulphur has been removed to introduce some carbonaceous matter so as to reduce the ferric oxide to magnetic oxide. When the iron is present originally as ferric oxide, as in limonite ore, a similar reduction by carbon is necessary. Siderite on the other hand is converted to magnetic oxide by a simple heating. The last process, not requiring so much delicacy in manipulation, has found more general application.

In calcining siderite, to make it magnetic, it is necessary to expose the mineral to a bright, cherry-red heat (850°C) for only 15 to 25 minutes. The mineral particles blacken and become so strongly magnetic that comparatively large pieces can be lifted by a very small horse-shoe magnet. The ore loses weight through the expulsion of the carbon dioxide of the siderite and the second atom of sulphur of any pyrite that may be present, but the heating should not be prolonged so as to cause the blende to roast. The ore is withdrawn from the furnace red hot, and emitting a strong odor of sulphur dioxide, but the latter disappears immediately and the particles of blende are observed to be coated with an infinitesimal film of white zinc oxide. These are evidences of a properly conducted calcination. When properly conducted, the siderite can be lifted out by a magnet of extremely feeble intensity and a very clean separation can be made from the blende. A low intensity of magnet is often desirable not only because of its less consumption of electrical power, but also because a higher intensity of magnetism may lift out some of the blende, when the latter contains combined iron as is the case with the Slocan blende.

Fortunately the same roasting which converts the siderite into magnetic oxide is approximately sufficient to convert the pyrites into magnetic sulphide, wherefore if there be pyrite in the ore it is also rendered magnetic to some extent. The conditions of roasting are not precisely the same, however, and the same degree of conversion to the magnetic form cannot be attained in both minerals. Whereas siderite is converted into magnetic oxide in about 20 minutes at cherry-red heat, for the conversion of pyrite into magnetic sulphide, when the quantity present is considerable, about 35 minutes appears to be required. The mineral becomes black, indicating the formation of the magnetic sulphide. If brownish or reddish particles are manifest, it is an indication that the roasting has gone too far, oxidation having been begun, which is undesirable.

The roasting may be done in a revolving cylinder, or in a furnace of the McDougall type. I consider the latter the preferable, because of its less first cost, less space required, and less cost both of maintenance and operation. A furnace of this type costs, erected, from $5 to $6 per square foot of hearth area. Either the revolving cylinder or the McDougall type of furnace makes a rather large percentage of flue dust, and an ample dust-settling chamber should be provided. This chamber should be of such section that the velocity of the furnace gases passing through it will be reduced to 4 ft. per second, and before the gases are allowed to escape through the chimney they should remain in the chamber long enough, say 15 seconds, for the dust to settle.

In view of the probable importance in the Slocan of magnetic separation by such a roasting as has been above described, followed by passage of the ore over machines of low magnetomotive force, it will be useful to describe the process employed at Friedrichssegen, on the Lahn, Germany, although after nearly 20 years of application it was a few years ago abandoned in favour of treatment of. the raw ore on Wetherill machines.

The ore treated at Friedrichssegen is a mill product assaying from 11 to 15% Zn. (in the form of blende) and 18 to 23% Fe (in the form of siderite). This was heated to redness in a furnace of the McDougall type, which put through from 20,000 to 25,000 kg. of ore (according to the size of the particles) in 24 hours, with a coal consumption of 1,200 kg., i.e. about 5% of the weight of the ore, which is, I believe, a safe factor to use in calculating the cost of this process. The plant comprised two furnaces, each of which required the attention of one man, who also trammed the calcined ore to the cooling floor. The work of the furnaces was about 175 lbs. of ore per square foot of hearth per 24 hours. When the ore had cooled to 50°C., or lower, it was elevated to a trommel which divided it into sizes, namely, larger than 4 mm., 2 to 4 mm., and smaller than 2 mm. The stuff larger than 4 mm., which was due to fritting together of particles during the calcining, went to a set of rolls, by which it was reduced to 4 mm., whence it was raised again to the sizing screen; the stuff from 2 to 4 mm. in size and smaller than 2 mm. fell into separate bins, whence it was drawn to the primary magnetic separators. There were 12 of the latter arranged in three groups of four each, the four machines of each group being set in pairs and the pairs in series. The arrangement of the four machines constituting a group is shown in Fig. 4. Two groups took the coarser mineral and two the finer. The ore diverted to a group was divided equally between machines A and B, which made two products, one enriched in zinc and the other enriched in iron. The iron product of both machines was led to D and the zinc product to C. The two lower machines made a zinc product with 38 to 42% Zn. and 6% Fe at the most, a mixed product, and an iron product, which still contained 6 to 8% Zn. The mixed product was retreated by a group of two machines. The iron product went to another group of four machines, which yielded a final product containing 40% Fe and 3 to 4% Zn; that represented the entire loss of zinc in the process.

It will be observed that the plant comprised 18 separators, of which 12 were employed in making the primary separation and six in reworking between products. The general arrangement of the plant is shown in the accompanying engraving, Fig. 5, which will be readily understood from the foregoing description.

The calcining furnace used at Friedrichssegen requires no extended description, since it differed, from the Herreshoff furnace and others of the McDougall type only in dimensions and structural details. It had two series of five superimposed hearths, which were about 6 ft. in diameter. The vertical shaft, which carried the stirring arms, was protected from the heat inside the furnace by an enclosing tube, and was driven by a worm gear at its upper end. In each hearth-room there was a stirring arm which moved forward the ore. The latter fell on the uppermost hearth at its periphery and was plowed toward the centre, where it fell through a hole to the second hearth, on which it was plowed toward the periphery, and so on, being discharged from the lowest hearth into a car standing to receive it. The furnace was fired from a grate whence the flames passed over the hearths in the direction of the arrows, escaping through a dust chamber to the chimney. The speed of the plows was regulated according to the size of the ore particles. The admission of air into the furnace was governed so as to prevent, so far as possible, the formation of ferric oxide instead of magnetic oxide.

The magnetic separators formerly employed at Friedrichssegen are shown in Figs. 6 to 9. A wooden frame supports a stationary axis *b*, to which is fastened a casting wound with copper wire so as to form four electromagnets, the wires for the connection of the latter passing through the axis *b*, which at each end is bored for a third of its length. The magnets, which are stationary in the position shown in Fig. 7., are surrounded by a brass drum, on the exterior of which brass flanges are brazed on parallel with its central axis. The drum is rotated in the direction of the arrow by means of the pulley *g*. while a pulley *h*, on the opposite side, transmits motion to the shaking tray *e*, which receives ore from the hopper *d*, and presents it to the drum in a thin even sheet. The space between the edge of the tray and the drum is adjustable. In operation, the magnetic particles are attracted to the surface of the drum, where they are held so long as in the magnetic field, being thence carried over into a separate bin, the flanges preventing them from falling back. The non-magnetic particles fall directly from the edge of the tray into a bin. The capacity of a single machine is from 300 to 500 kg. per hour. The drum makes 36 r. p.m. and the shaking tray from 180 to 200 throws per minute. The power required is 0.125 h.p. and the electric current about 325 watts per machine. The Friedrichssegen plant comprised five dynamos, each of which delivered 20 to 25 amperes of 65 volts. One man attended to the entire plant of separators.

Since 1898, the Wetherill separator has been in use at Friedrichssegen, treating the ore raw, which evidently has been found an improvement there although the details have not been published; the operating expenses are said to have been reduced while the grade of the zinc product has been increased. The advantage of any process depends, however, largely upon local conditions. The experiments on the Slocan ores, made at Denver, Colo., by the Commission, have indicated the possible superiority of the roasting process and low intensity magnets under the particular conditions as will be discussed in detail further on.

At Meiern, in the Tyrol, a similar process, for the treatment of a similar ore, is outlined by the flow-sheet shown in Fig. 12.

In further description of the existing practice in the treatment of this particular class of ore, the method in use at Lohmannsfield, in the Siegen district, Germany, may be referred to. At this place the ore consists of galena, siderite and blende, with a gangue of quartz and quartzite. The siderite contains up to 12% of manganese. By the ordinary process of milling, the third and fourth compartments of the Harz jigs furnished a mixed product containing from 2 to 3% of gangue and 15 to 22% of blende, according to the tenor of blende in the crude ore, the remainder being spathic iron. A marketable blende could be produced by roasting and removing the magnetic oxide by ordinary magnetic separators, but the company was not able to install roasting furnaces. Experiments in separating the ore crushed to 3 mm. size by means of Wetherill machines gave such favourable results that it was determined to make an installation of them.

The crude mineral sent to the Wetherill plant is the middlings from the Harz jigs of the wet-dressing works, varying in size from 1 to 10 mm. and containing from 5 to 20% of water. The arrangement of the plant is shown in the accompanying engravings, Figs. 13 to 16. The mineral is dried while being moved forward by means of a screw conveyor, the trough of which is made double, of cast iron, exhaust steam from the engine being passed between the two sections. The speed of the screw A is such that the mineral remains in the trough 30 minutes, while in the trough B it remains 25 minutes. These conveyors discharge the dried mineral into the primary trommels, A_1 and A_2, by which it is separated into two classes. That which is finer than 3 mm. falls into the boot of the elevator C_1, which raises it to the main system of classifying trommels; that which is larger than 3 mm. passes to two sets of rolls, whereby it is crushed, falling thence into the pit B, whence the elevator C_2 raises it to the trommel A_2, to separate it into mineral smaller than 3 mm., which falls into the pit B, and mineral larger than 3 mm., which is returned to the rolls.

All the mineral crushed to pass a 3 mm. screen is collected therefore in the pit B_1, whence the elevator C_1, raises it to the train of sizing trommels, to reach which, however, it has to be conveyed by a horizontal belt D, over which it set a powerful electro-magnet to pick out strongly magnetic articles such as rivet heads, small pieces of iron and steel, etc.

The sizing trommels separate the mineral into four classes, namely, 3 to 2 mm., 2 to 1.4 mm., 1.4 to 0.75 mm., and that which is smaller than 0.75 mm. The separated classes pass thence over three series of Wetherill separators of the two-pole and three-pole types, arranged two in a series. In the first member of each series, pure spathic iron is removed by a feeble current. The tailings pass to the lower machine where the intensity of the magnetic field is greater, enabling a mixed product of blende and spathic iron to be removed, while pure blende passes on as tailings. The blende of Lohmannsfield is non-magnetic even in magnetic fields of the greatest intensity, because of the absence of iron and manganese in its composition. The width of the points of the poles of the Wetherill machines used at Lohmannsfield is 340 mm. The belt speed of the upper machine is 40 m. per minute; of the lower 25 m. The current employed is of 65 volts electromotive force; the upper machine of the two-pole type works with 12 amperes; the lower with 14 to 16 amperes. For machines of the three-pole types the corresponding figures are five amperes for the upper and eight amperes for the lower.

The plant is capable of separating 3 to 3.5 metric tons (3.3 to 3.85 tons of 2,000 lb.) of crude mineral per hour, the crew required for its operation being as follows: One foreman, five 16 to 18-year-old boys, one engineman, and one stoker. The cost of treatment (year's average) is 1.4 marks (33⅓c.) per metric of crude mineral, no amortization of the plant being reckoned. The plant cost about 100,000 marks ($23,800). Allowing 20% for amortization, 20,000 marks per annum on 8,000 tons (the capacity of the plant), the total cost of treatment is 3.9 marks (92.8c. per ton.)

Some valuable deductions may be made from the above figures. The average cost of treatment per metric ton of crude material is 33⅓c. The capacity of 8,000 metric tons per annum corresponds to 26⅔ tons per day, assuming a running time of 300 days. It may be assumed that the foreman is paid $1.50 per day, the boys 50c., and the engineman and stoker $1 each, giving a total daily labour cost of $6, or 22½c. per metric ton, leaving a little less than 11c. for power, supplies, repairs and renewals, etc. In Canada or the United States the wages for labour would be much higher, but such a plant would be run by fewer men. We may reckon a labour cost of three times as much, or 68c., and doubling the other costs we should have a total of 90 cents per metric ton, or say 82c. per ton of 2,000 lb. The allowance of 20% for amortization may be too high, but experience is not yet sufficient to furnish much data as to the life of such plants. It would be unwise, however, to reckon amortization at less than 10% which with interest at 6% comes to nearly 20%. Reckoning the cost of plant at $3 per ton of annual capacity, the total cost per short ton of crude material would be:—

Direct operation	$0.82
Amortization at 10%	.30
Interest at 6%	.18
Total	$1.30

At the old mill of the New Jersey Zinc Co., at Franklin Furnace, N.J., which was of 200 tons capacity per 24 hours, the cost of separation was 74.54c. per long ton, of which 53.35c. was for labour, 8.77c for coal and 12.45c. for miscellaneous supplies, and repairs and renewals. The labour was divided as follows:—

Crushing house, including drying and hand picking	$0.1175
Separating house, including jig runners and drying concentrate	0.1477
General, including foremen, enginemen, electricians, loaders, etc.	0.2683
Total	$0.5335

In the new mill of the company, which is of 1,000 tons capacity per 24 hours, the operating cost is about 40c. per ton, not including interest and amortization. The cost of this mill was about $600,000, or $2 per ton of annual capacity. This is stated, however, in terms of long tons, and the capacity is somewhat underrated. It would, doubtless, be more nearly correct to estimate the cost of the plant at $1.75 per 2,000 lbs. of annual capacity.

In a magnetic separating mill of 50 tons daily capacity, for the treatment of mixed sulphide ore by concentration on Wilfley tables, and magnetic separation of the zinky product by Wetherill machines, the cost per ton of crude ore, under the conditions existing in Colorado is about $1.25 @ $1.50 per ton, exclusive of interest, amortization, and general expense. The cost of plant will be about $45,000, or $3 per ton of annual capacity. Interest (at 6%) and amortization (at 10%) will amount therefore to about 50c. per ton.

I have previously expressed the opinion that the conditions existing in the Slocan, and the probabilities to be anticipated from its mines, do not warrant the erection of more than two mills, of 50 tons daily capacity each, to treat the ores of the whole district. It will be pointed out, in the summary of the ore tests conducted by the Commission, on a subsequent page, together with the detailed report of Mr. Argall on this subject, that the ores of the Slocan are quite variable in character, requiring radically different methods of treatment. This subject may be anticipated here to the extent of remarking that certain ores require a preliminary concentration on tables, a roasting of the zinky concentrate, and subsequent separation by magnetic machines of high intensity. Other ores require a preliminary roasting and subsequent separation on magnetic machines of low intensity. Still other ores are best treated raw by direct action of magnetic machines of high intensity. Moreover, some ores will be received in the form of a rough concentrate, for which rolls only are required for their further comminution; while other ores will be received in lump form, requiring means for crushing as preliminary to delivery to the rolls. A mill to treat satisfactorily this wide variety of ores must consequently be equipped with adequate means for treating all kinds.

Such a mill should comprise essentially one 9 × 15 in. Blake crusher, two sets of 15 × 26 in. rolls, one cylindrical dryer, six Wilfley tables, five Wetherill separators, or three Internationals, two 5-hearth, 9-ft. McDougall furnaces, and one conveyor to remove the ore from the last, at the same time cooling it. A long, slightly tapering iron cylinder, supported by rollers like a rotary cement kiln, and sprayed externally with water serves as a very efficient cooler and conveyor. Besides the necessary belt elevators, screens, hydraulic classifiers, etc., the mill should have a platform scales for weighing the ore upon receipt, a Vezin sampler for sampling it, ample bins for storage of the crude ore, and moreover ample bins for storage of certain supplies of ore intermediate in the process. There should be filter bins for receiving the products from the Wilfley tables, these being constructed of plank, without any attempt to make tight joints, the bins being lined on the inside with burlap through which the water can strain out, leaving the slime behind. The roasting furnaces should have a dust chamber of adequate size. The power plant should be of 100 h.p. The works should be constructed on level ground and the system of elevators and conveyors should be so arranged that the course of the ore through the mill may be altered as required by the variety of treatment that may be necessary.

In the absence of detailed plans and specifications for such a plant, no precise estimate of its probable cost can be given, but it would be approximately $50,000 to $60,000, or $3.33 to $4 per ton of annual capacity, reckoning the latter at 15,000 tons, *i.e.*, 50 tons per day for 300 working days.

The cost of treatment will range from $1.25 to $2 per ton for direct operating expense, to which must be added at least 35c. for administration and general expense, and about 65c. for amortization and interest on the capital invested, making the total expenses come to $2.25 @ $3 per ton of ore treated. These apparently high estimates are due partly to the costly character of the plant required, and partly to the high rates for labour that prevail in British Columbia. In stating milling costs, commonly no allowance for interest and amortization is commonly included, and the addition to the above estimates on this account may make them appear inordinately high, but this charge is a very real factor in the cost of milling and profits cannot properly be reckoned until it has been amply allowed for.

To aid in the detailed consideration of the ore tests reported further on, some indication of the cost of performing the various processes must be given, which for convenience will be based on a plant of capacity for the treatment of 50 tons of ore per day. In British Columbia, the skilled labour required in such a plant will command $3 wage per eight hours. The unskilled labour can perhaps be obtained for $2.50 per 10 hours. Coal at Kaslo or Roseberry will cost $6 per ton.

As between magnetic separators of high and low intensity, the former require much more power than the latter, the ratio for equal ore capacity being under certain conditions as 7.5 : 1.25, The difference in this respect is, however, partially offset by the cost of roasting, which is a necessary preliminary to the employment of low intensity machines at all. The cost of roasting the blende-siderite ores of British Columbia will be about 50c. per ton. When washing on Wilfley tables is required as part of the process, an additional cost of about 15c. per ton will be entailed.

Attention must be called emphatically to the fact that these estimates, which are admittedly conservative, are based on the probable operation of a small custom plant, designed for the treatment of a wide variety of ores, and including expenses for rehandling ore, administration, etc., of which an individual mine plant would be free. In the case of a mine developed sufficiently to justify an independent plant, both the first cost of installation and the operating expense would be considerably less than what has been herein estimated, and moreover all costs would diminish materially with increase in the size of the plant. With this explanation, the direct operating cost, under a variety of conditions, may be generalized roughly, as follows:—

Treatment.	Cost Per Ton.
1. Raw magnetic separation, h.i. machines	$1.30
2. Roasting and magnetic separation, h.i. machines	1.85
3. Tabling, roasting and magnetic separation, h.i. machines	2.00
4. Roasting and magnetic separation, l.i. machines	1.70
5. Tabling, roasting and magnetic separation, l.i. machines	1.85

In each case the above figures are to be increased by $1 to allow for general expense, interest and amortization as explained above.

SUMMARY OF MAGNETIC ORE TESTS.

The following condensed summary of the magnetic ore tests, including also the tests with the Blake electrostatic separator and tests of wet concentration, conducted by the Commission at Denver, Colo., under the direct supervision of Mr. Argall, is based on the report by Mr. Argall, which is given in full in a subsequent section of this volume. All computations of ore values are based on zinc at £23 (5c. per lb.) at London, lead at 2.5c. per lb. in British Columbia, and silver at 60c. per ounce. In the case of lead ores, the net value is reckoned on the basis of $12 per 2,000 lb. for freight and treatment.

The valuations of the various ores are presented as illustrative rather than as absolute determinations. Obviously, nothing more than this can be expected. Not only are the prices for the metals constantly fluctuating, but also the prices for ore, based fundamentally on the prices for their metal contents, which the smelters will bid, are subject to never ending and wide variations. Even during the writing of this report the market both for lead and zinc ore has experienced radical changes. In a general way, however, the following estimates reflect the conditions toward the end of 1905, except that the value of lead ore has been computed on the new basis, and such errors as there may be in them, or such differences from present conditions, will not materially affect their usefulness for purposes of comparison, or their approximate indication of the results that may be expected from the milling of these ores.

Lot 1.—Jackson mine. Blende-siderite-pyrite-quartz. Assayed 35% zinc, 1.4% lead, and 5 ozs. silver. Treated raw, one ton of ore gives 0.666 ton of zinc concentrate, assaying 49.1% zinc, saving 91.59% of the zinc in the original ore. Treated after roasting, one ton gives 0.637 ton of zinc concentrate, assaying 50.75% zinc, saving 94.37% of the zinc in the original ore. Loss of weight in roasting 13.8%.

A comparison of the above results is as follows:—

A. RAW TREATMENT—

0.666 ton zinc ore @ $12.10	$8.06
Cost of concentration	2.30
Net value	$5.76

B. ROASTING TREATMENT—

0.637 ton zinc ore @ $13.75	$8.76
Cost of concentration	2.70
Net value	$6.06

The second method is, therefore, the more profitable.

Lot 2.—Ruth mine. Blende-siderite-pyrite-quartz. Assayed 37% zinc, 1.4% lead, and 9.4 ozs. silver. Treated raw, one ton of ore gives 0.710 ton zinc concentrate, assaying 48.4% zinc, saving 94.96% of the zinc in the original ore. Complete results of roasting-treatment were not made on this ore, but judging from the partial test they would have been similar to the results shown by Lot 1.

Lot 3.—Payne mine. Blende-siderite-pyrite-quartz. Assayed 42% zinc 4.9% lead and 18 oz. silver. Treated raw one ton gives 0.88 ton of zinc concentrate, assaying 47.1% zinc, saving 98.9% of the zinc in the original ore, the silver of the ore going chiefly (96%) into the zinc concentrate. Treatment after roasting gives (1) 0.1345 ton of iron ore, assaying 21.4 oz. silver, 5% lead and 8.3% zinc; (2) 0.7831 ton of zinc concentrate, assaying 52.4% zinc, 4.2% lead, and 17.7 ozs. silver; and (3) 0.0724 ton of lead ore, assaying 30.8% zinc, 10.9% lead and 32.2 ozs. silver. By retreatment of (3) a further portion of zinc can be recovered. The loss of weight in roasting was not stated.

In this case, the crude ore would be marketable as a zinc ore, although its value would be inferior to the result after concentration. The comparative results of marketing crude ore and the two available methods of enrichment are as follows:—

A. CRUDE ORE—

1 ton @ $10.68	$10.68

B. RAW TREATMENT—

0.88 ton zinc ore @ $15.50	13.64
Cost of concentration	2.30
Net value	$11.34

C. ROASTING TREATMENT—

0.1345 ton iron ore @ $11.45	$1.54
0.7831 ton zinc ore @ $19.77	15.48
Total	$17.02
Cost of concentration	2.70
Net value	$14.32

Here there is a distinct and important advantage in magnetic separation after a preliminary roasting, which will be increased by further treatment of the zinc-lead middling, disregarded in the above computation.

Lot 4.—Hartney mine. Blende-siderite-galena-quartz. Assayed 26.1% zinc, 8.4% lead, 23 oz. silver. Crushing to 0.02 in. size and washing on table gave, from one ton of crude ore, 0.1113 ton of galena concentrate, assaying 53.2% lead, 12.2% zinc and 70.5 ozs, silver; and 0.8286 ton of blende-siderite-quartz tailing, assaying 28.2% zinc, 2.3% lead and 16.4 ozs. silver. Roasting the latter product and separating it magnetically gives from a ton of the original ore 0.5 ton of zinc concentrate, assaying 44% zinc, and a middling product, incapable of further separation, which is worthless either as a zinc ore or as a lead ore.

In this case a crude ore which is unmarketable gives products as follows:—

0.1113 ton lead ore @ $51	$5.68
0.5000 ton zinc ore @ $7	3.50
Total	9.18
Cost of concentration	2.50
Net value	$6.68

Lot 5.—Slocan Star mine. Blende-siderite-pyrite-pyrrhotite-quartz. Assayed 33.8% zinc, 1.8% lead and 38.6 ozs. silver per ton. Treated raw, one ton of ore gives 0.709 ton of zinc concentrate, assaying 44.8% zinc, saving 94.9% of the zinc in the original ore. Roasting and subsequent separation yields 0.54 ton of zinc concentrate, assaying 50.6% zinc and 0.36 ton of iron ore, assaying 35.5 ozs. silver per ton. The zinc concentrate assays 46.8 oz. silver per ton and carries about two-thirds of the amount present in the original ore.

The commercial results on this ore are as follows:—

A. RAW TREATMENT—

0.709 ton zinc ore @ $24[a]	$17.02
Cost of concentration	2.30
Net value	$14.72

[a] Approximate, the silver content of this ore not having been determined.

B. ROASTING TREATMENT—

0.54 ton zinc ore @ $28.65	$15.47
0.36 ton iron ore @ $13.00	4.68
Total	$20.15
Cost of concentration	2.70
Net value	$17.45

The commercial advantage of roasting and then separating magnetically is in this case very striking, as is also the degree to which the crude ore is enriched. The crude ore is a marketable product (because of its high silver content), but its value per ton is more than doubled by the simple treatment indicated under B. This result is particularly important inasmuch as the Slocan Star mine is among those of the West Kootenay district that has the largest capacity for production.

Lot 6.—Monitor and Ajax mine. Blende-pyrite-pyrrhotite-galena-siderite. Assayed 34% zinc, 3.60% lead and 14 oz. silver. Roasting and magnetic separation give, from one ton of ore, 0.616 ton of zinc concentrate, assaying 50% zinc, 9.8 oz. silver, and 0.13 oz. gold, saving 90.64 % of the zinc in the original ore; and 0.313 ton of iron ore, assaying 0.34 oz. gold, 24.9 oz. silver, 5.1% lead and 10.1% zinc.

In the case of this ore the benefit to be derived from magnetic enrichment is very positive. The crude ore is of too low grade in zinc and silver to be marketable. The process of treatment which is outlined above, gives products as follows:—

0.616 ton zinc ore @ $16	$9.86
0.313 ton iron ore @ $20	6.26
Total	16.12
Cost of concentration	2.85
Net value	$13.27

Lot 7.—Enterprise mine. Blende-siderite-galena-pyrite-quartz. Assayed 43.7% zinc, 4.8% lead and 115 ozs. silver. This ore is already a fair grade of zinc ore, and treatment must be aimed at putting as much as possible of the silver into lead smelting ore, rather than into zinc ore, because of its greater value in the former. The tests point to a reasonably successful method of treatment. The ore should be crushed to pass a 25-mesh screen and should then be carefully concentrated on Wilfley or similar table, yielding about 4% of galena, assaying 45% lead and 293 oz. silver. The tailings from the tables should be collected in burlap-lined bins, which while allowing all of the water to drain out will save all of the tailings. The latter should then be dried, should be roasted lightly and should be passed over a Wetherill magnetic separator. The tests do not unfortunately, show precisely the results that may be expected from this treatment, but they indicate that out of a ton of the crude ore there can be obtained 0.55 ton of zinc ore with about 57% zinc, and not more than 40 ozs. silver per ton, a further product of comparatively high grade zinc ore, also high in silver, which may be marketed either with the zinc smelter or the lead smelter, and finally an iron ore, high in silver, that would be sold to the lead smelter.

In this case the crude ore is marketable either as a zinc ore or as a silver-lead ore. As a zinc ore its value would be $46.30 per ton. As a silver-lead ore its value would be $37.80 per ton. The problem is to separate this ore, without loss of valuable minerals, so that the aggregate value of the various products will exceed that of the crude ore. The tests were not carried to their ultimate conclusion, but basing computations on Test C, (See Mr. Argalls' report) one ton of the crude ore yields the following products:—

0.550 ton zinc ore @ $32.60	$17.93
0.124 ton silver ore @ $80.89	10.03
0.241 ton iron ore @ $117.55	28.33
Total	$56.29
Cost of concentration	2.85
Net value [a]	$53.44

[a] The net value of this ore would doubtless be materially increased if a considerable tonnage of the zinc products rich in silver could be offered.

It is evident that separation adds materially to the value of this ore, and it is probable that a process conforming to the suggestions in Mr. Argall's report will lead to a greater increase in value than has been here estimated. This line of treatment is the most promising solution of the problem of the zinky ores of the Slocan that are very high in silver.

Lot 8.—Molly Gibson mine. Blende-siderite-galena-pyrite-quartz. Assayed 11.3% zinc, 4.3% lead and 71.5 ozs. silver. This is obviously an ore which can be treated most advantageously by direct lead smelting.

Lot 9.—Big Ledge. Blende-pyrites-pyrrhotite. Assayed 19.4% zinc, trace of lead, and 0.7 ozs. silver.

Lot 10.—Big Ledge. Blende-pyrites-pyrrhotite. Assayed 20.6% zinc, trace lead, 0.7 ozs. silver.

Lot 11.—Big Ledge. Blende-pyrites-pyrrhotite. Assayed 22.7% zinc, trace lead, 0.7 ozs. silver.

Lots 9, 10 and 11 are of practically identical character. Concentration on tables to remove gangue, roasting of the concentrate and treatment of the roasted ore on the Wetherill separator give, from one ton of the crude ore 0.367 ton of concentrate assaying 40.7% zinc, a saving of 79.5% of the zinc in the original ore. In a region nearer to the markets than British Columbia, this would be considered a fairly good result, both technically and commercially. The ore is a difficult one to separate. In treating it raw, only a small proportion of worthless material can be picked out as magnetic material, while after roasting the blende and pyrites all become more or less magnetic, and there is no sharp dividing line between the attractability of the various minerals.

The result of the treatment of this ore, stated above, does not offer any hope of profitably working this ore on the basis of 5c. spelter and other conditions as they exist at present, because the gross value of the product derivable from a ton of the crude ore would not amount to the cost of production. The gross value of the products would hardly be more than $1.35 per ton of crude ore, which could not be expected to cover the cost of mining and milling even if carried out on the large scale that the magnitude of the ore deposit would appear to warrant.

Lot 12.—Goodenough mine. Blende-galena-siderite-pyrite. Assayed 45% zinc, 10.8% lead and 22 ozs. silver. Crushing to pass a screen with

0.03-in. holes and washing on tables gives, from one ton of crude ore, 0.893 ton of zinc concentrate, assaying 48.7% zinc and 15 ozs. silver, comprising 97.4% of the zinc in the original ore, and 0.107 ton of lead concentrate, assaying 62.4% lead and 81.6 oz. silver. The zinc concentrate can be further raised in grade by magnetic separation, but the additional gain would not justify the expense. The treatment outlined above is simple, efficient and satisfactory.

The crude ore is marketable as a zinc ore and the question is as to the gain in value by separation of its constituents. This appears in the following comparison:—

A. CRUDE ORE—	
1 ton @ $15.10	$15.10
B. SEPARATED PRODUCTS—	
0.893 ton zinc ore @ $15.30	$13.67
0.107 ton lead ore @ $62.00	6.63
Total	$20.30
Cost of concentration	1.00
Net value	$19.30

The cost of concentration in this case is low, because it is a simple problem in milling.

Lot 13.—Hewitt mine. Blende-siderite-pyrite-quartz. Assayed 32.8% zinc, 11.5% lead, and 80.2 oz. silver. Concentration on tables to cut out a galena product, collection of the tailings, roasting and magnetic separation give, from one ton of ore, 0.482 ton of zinc concentrate, assaying 54.8% zinc, and 41 ozs. silver, comprising 83.3% of the zinc in the original ore, and 0.446 ton of lead concentrate assaying 25% lead, 14% zinc, 20% excess of iron over silica, and 131 oz. silver, comprising 100% of the lead and 75% of the silver in the original ore.

This ore, because of its high silver content, would be marketable in its crude state either as a zinc ore or as a lead ore, but its value is greatly increased by separation into two classes, the high concentration of lead and silver in the lead ore product causing the separation to be of remarkable technical efficiency. The products derivable from a ton of the crude ore are as follows:—

0.482 ton of zinc ore @ $30.76	$14.83
0.446 ton of lead ore @ $77.92	34.75
Total	$49.58
Cost of concentration	2.50
Net value	$47.08

The increase in the value of this ore by separation is very important.

Lot 14.—Emily Edith mine. Blende-siderite-galena-pyrite-quartz. Assayed 38% zinc, 8% lead and 24 ozs. silver. Concentration on tables to cut out a galena product, collection of the tailings, roasting and magnetic separation, give, from one ton of ore, 0.62 ton of zinc ore, assaying 47.29% zinc, 3.06% lead, and 18.3 ozs. silver, comprising 81% of the zinc in the original ore; and 0.078 ton of lead ore, assaying 54.6% lead, 12.5% zinc, and 58.5 ozs. silver; besides various iron products of doubtful value.

As a silver-lead ore, this ore could not be marketed crude. As a zinc ore it would be worth about $6.84 per ton. Separated into products, as stated above, its value would be as follows:

0.620 ton zinc ore @ $15	$9.30
0.078 ton lead ore @ $44.57	3.48
Total	$12.78
Cost of concentration	3.00
Net value	$9.78

Lot 15.—Lucky Jim mine. Blende-galena-pyrites-quartz. Assayed 33.2% zinc, 9.8% lead, and 11.5 ozs. silver. By washing on tables, roasting the zinc product and separating magnetically, one ton of ore yields 0.15 ton of lead ore, assaying 50.2% lead, 11.9% zinc and 49.5 ozs. silver; and 0.4124 ton of zinc ore, assaying 54% zinc, the extraction of zinc being about 70% of the content of the original ore, while the extraction of lead in the marketable lead ore is 73.8%

This ore crude would have no value either as a zinc ore or as a lead ore. Separated into two classes it acquires value as follows:—

0.4124 ton zinc ore @ $17	$7.01
0.1500 ton lead ore @ $37.80	5.67
Total	$12.68
Cost of concentration	3.00
Net value	$9.68

Lot 16.—Hewitt mine. Quartz-blende-galena-siderite-pyrite. Assayed 5.8% zinc, 5.4% lead, and 18 oz. silver. Concentrating on tables, roasting the zinc concentrate, and separating magnetically, one ton of ore yields 0.0434 ton of lead concentrate, assaying 68.4% lead, 6.87% zinc and 131 ozs. silver; and 0.0205 ton of zinc concentrate, assaying 54% zinc, and about 33 ozs. silver.

In this case, the concentration is only indifferently successful, notwithstanding the production of a high grade of zinc ore, the unsatisfactory result from the commercial standpoint being due to the low grade of the crude ore and the high loss in the process. The products are as follows:—

0.0205 ton of zinc ore @ $27.08	$0.56
0.0434 ton of lead ore @ $93.45	4.06
Total	$4.62

The above estimate is for the gross value of products per ton of crude ore without any allowance for the cost of milling.

Lot 17.—Aurora mine. Heavy zinc-lead sulphide, practically free from gangue. Assayed 33% zinc, 21.5% lead and 7.3 ozs. silver. Separating the raw ore by Wetherill machine, one ton yields 0.455 ton of lead ore assaying 42% lead and 14 ozs. silver; and 0.53 ton of zinc ore, assaying 51.4% zinc, 82.4% of the zinc in the original ore being recovered.

The products are valued as follows:—

0.530 ton zinc ore @ $14.40	$7.63
0.455 ton lead ore @ $16.48	7.50
Total	$15.13
Cost of concentration	2.30
Net value	$12.83

The crude ore is of high grade both in lead and zinc, and the separation is efficient; hence the good commercial result.

Lot 18.—Blue Bell mine. Blende-galena-pyrite-pyrrhotite-quartz-limestone. Assayed 14.6% zinc, 12.8% lead, and 3.8 ozs. silver. By a combination process, one ton of ore yields 0.2 ton of lead concentrate, assaying 55.2% lead, 5.1% zinc and 11 ozs. silver; and 0.227 ton of zinc concentrate assaying 44% zinc, the latter representing 69.4% of the zinc in the crude ore.

The products are valued as follows:—

0.227 ton zinc ore @ $ 7	$1.59
0.200 ton lead ore @ $19.11	3.82
Total	$5.41
Cost of concentration	1.60
Net value	$3.81

The combination process proposed for the treatment of this ore consists first in crushing and washing on tables, producing a lead ore as a finished product (0.2 of the original ore) and a blende-pyrites middling, amounting to 0.5315 of the original ore. This blende-pyrites product is dried and passed over Wetherill separators at low amperage, whereby 34.73% of the product is removed as worthless pyrrhotite. The remainder, amounting to 0.6527× 0.5315, or 0.347 of the original ore is roasted and then passed again over magnetic separators, the pyrites rendered magnetic by the roasting being picked out at low amperage, while the blende also made magnetic, but less so than the zinc, is removed at high amperage. A small amount of non-magnetic material (quartz, etc.), very low in zinc, goes over the tail end of the separator as waste.

The process is an ingenious solution of a difficult problem. The mine for which it is designed ranks among those of the West Kootenay that are capable of making a moderately large production, and would undoubtedly be equipped with an independent plant. On the basis of 100 tons capacity per 24 hours, the process outlined above should be performed for about $1.20 per ton, direct operating expense; and not more than $0.40 per ton, interest and amortization; total, $1.60 per ton.

Lot 19.—St. Eugene mine. Blende-pyrite-galena-garnet-quartz. Assayed 20.2% zinc, 9.7% lead, and 5.7 ozs. silver. By wet concentration, followed by magnetic separation of the zinc product, without roasting, one ton of ore yields 0.157 ton of lead concentrate, assaying 33½% lead and 16.3 ozs. silver; and 0.314 ton of zinc concentrate assaying 40.8% zinc, 56% of the zinc in the original ore being recovered; while, moreover, the grade of this zinc concentrate can be raised to 50% by roasting and further treatment. If raised to 50% grade, the product would be 0.218 of the weight of the original ore.

The value of these products would be as follows:—

0.218 ton of zinc ore @ $13	$2.83
0.157 ton of lead ore @ $15.48	2.43
Total	$5.26

The cost of performing the process can not easily be estimated for the purpose of this report, because it would be conducted as an adjunct to the present milling process, which is practiced on a large scale, and would depend upon the arrangements which would result from a careful study of the problem by the managers of the property. The process consists in regrinding the zinky product made by the present mill and carefully washing on tables, producing a galena concentrate, amounting to 0.105 of the original material, and a slightly enriched zinky product, amounting to 0.6646 of the original. This zinky product is dried and passed over Wetherill separators, yielding at high amperage 0.314 ton of zinc ore assaying 40.8% zinc, besides a small amount of low grade lead ore, which is combined with the galena concentrate from the tables. The zinc concentrate, amounting to less than one third of the weight of the original material, is then roasted and passed over a magnetic separator of low intensity. Consequently all of the material must be washed on tables. The two passes over magnetic separators are practically equivalent to the passage of all the material once over such separators, two-thirds of high intensity and one-third of low intensity, while roasting is required on only one third of the material. As previously stated (in an earlier portion of this report) the quantity of this material produced by the St. Eugene mill is about 25 tons per day. It is evident that it can be further worked, as herein suggested, for a considerable additional profit.

GENERAL CONCLUSIONS.

The possibility of enriching the zinc ores of British Columbia to a high degree by magnetic separation has been thoroughly demonstrated by the tests conducted by the Commission, which have been hereinbefore summarized. The ores tested are of wide variety and represent all of the classes that the mines of the Province are at present capable of producing. In every case it has been possible to produce a zinc concentrate assaying upward of 40% zinc; in many cases concentrates assaying about 50% zinc; and in a few cases concentrates assaying as high as 57% zinc.

These results compare very favourably with those which are attained in the concentration of similar ores in the United States, in Australia and in Europe. The zinc concentrates, produced from the mixed ores of west of the Rocky Mountains in the United States do not average better than 40% zinc; that produced at Broken Hill, N.S.W., goes only 40 to 45% zinc. The zinc concentrate that can be produced in British Columbia will average better than those figures. It is to be remarked, however, that a thoroughly good technical result is not necessarily a good commercial result. Ore assaying only 40% zinc, unless it contain also an important silver value, will not be a profitable product in British Columbia on the basis of 5c. spelter (it is not safe to make calculations on a higher basis than that), but this is because of the remoteness of the Province and the long railway carriage to markets, absorbing a large part of the value of the ore, which must inevitably be borne. This is an unfortunate condition, which can not be altered; indeed, can not be much ameliorated. This subject has been thoroughly discussed in previous sections of this report.

Of practical importance in no way inferior to the degree of concentration is the matter of percentage of mineral extracted from the ore. With the exception of a few especially difficult ores, the tests of the Commission show remarkably high percentages of extraction, these rising in one case to nearly 99%, and being upward of 90% in many cases.

The matter of relative efficiency of various types of magnetic separators will not be discussed, except broadly. It was not within the means of the Commission to make exhaustive competitive trials of different makes of machine, and anyway this matter is of comparatively little importance providing the machines conform to sound principles of mechanism and magnetism. The design of these machines will be treated to some extent further on in this report. It is a common mistake among the inexpert to look for some machine, which in itself will be the solution of all problems in ore treatment. There is no one machine which will prove such a universal panacea. Attention should be concentrated not on the machine. but on the process. The machine is simply a means to carry out some part of the process. The process being successfully devised, it will frequently be found that several different kinds of machine will do equally well the particular work that is required of them. It will be observed from Mr. Argall's report that the principal experimental work of the Commission was in the determination of processes, rather than the competitive trial of various machines. In so far as such trials were made, readers of this report may draw their own conclusions.

It is proper, however, to point out the broad difference between the two grand types of magnetic separators, viz., the high intensity and the low intensity. The former require much more power than the latter, but they will make separations of minerals of low magnetic attractability, which remain unaffected by the machines of low intensity. In order to enable the latter to operate at all, the ore must be converted by roasting into a more highly magnetic form, and the cost of roasting is more than an offset of their economy in power and lower first cost. In the case of the blende-siderite ores, however, roasting and separation by low intensity machines gives a higher grade of product than does raw separation by high intensity machines, and the commercial result figures out better. Consequently, if that class of ore were alone to be treated, the choice would by all means be for a roaster and low intensity machines. But when it is a case of treating a variety of ores, some of which can be separated only by a high intensity machine, the choice is determined by different considerations. The high intensity machine will perform all of the separations of strongly magnetic material that the low intensity machine will make, and is capable of far more delicate adjustment and of effecting separations between minerals of several different degrees of magnetic permeability. A plant for the treatment of the miscellaneous ores of the West Kootenay should, therefore, be provided with high intensity separators, rather than low intensity. Means for roasting the ore are required in any case.

A disappointing feature of the experimental work at Denver was the failure of the Blake electrostatic separator to prove of service in the treatment of the ores of British Columbia. In no case did this appear to be of useful application. In the treatment of some ores in Colorado and elsewhere, this separator has given excellent results on a commercial scale, but its action depends upon the relative conductivity for static electricity of the component minerals of an ore, and in many cases the difference is not sufficient to enable a separation to be made. This appeared to be the case of the ores of British Columbia in so far as they were tested by this process. In fairness it is to be remarked that the tests of the Blake separator were by no means exhaustive, and it is possible that by modifications in the ore treatment satisfactory commercial results may be obtained with it in certain cases.

In Australia, the ores of Broken Hill were first successfully treated by magnetic separation. During the last two or three years a class of processes, known as the flotation processes, has been introduced, and these have now clearly beaten the magnetic separation processes in the treatment of the ores of Broken Hill. So far, the flotation processes have not been tried except at Broken Hill, and comparatively little is yet known as to their theory or the precise conditions which determine their application.

I caused some experimental work to be done, on a laboratory scale, at New York, under my own direction, to determine the applicability of these processes to the ores of British Columbia. I shall refer to these experiments at more length in a subsequent section of this report. It is sufficient to state here that some of the ores of British Columbia appear to be amenable to separation by this process, but the results do not indicate any superiority over magnetic separation in point of enrichment of the grade of the ore, while for the treatment of the miscellaneous ores of a district flotation would probably be less elastic and less efficient than magnetic separation. Flotation is especially adapted to the treatment of the ore of a single mine on a large scale. This condition does not obtain at the present time in British Columbia; certainly not in the Slocan. The conclusion may, consequently, be accepted that for the enrichment of these ores, the best method is a combination process in which washing on tables and magnetic separation are the essential features. The experimental work of the Commission was largely confined to this branch of the subject, as being the most logical development, which will explain the detailed attention, to the exclusion of nearly everything else, that is given to it in Mr. Argall's report.

MAGNETIC SEPARATORS.

The forms of magnetic separators which have been designed and put into practical use, especially for the separation of magnetite, are so numerous, that a description of them and their principles would constitute the subject for a voluminous treatise. Even for the separation of zinc ore, a large variety of separators has been practically employed. In this report it is unnecessary to go further than refer to a few of the more important.

The principle of all magnetic separators is simple. An electro-magnet, when energized by a current passing through its coil, attracts magnetic substances to one or both of its poles. This is the first step in the separation. The second is to cause the magnetic material to drop off from the pole, separate from the non-magnetic material from which it was removed. The simplest, but most impracticable, method of doing this is to lift away the magnet, cut off the current and allow the attracted material to fall into a separate receptacle. Magnetic separation has been, indeed, effected in practice, on a small scale, in precisely that manner, and it is a useful method for laboratory tests, but for commercial work it is too cumbersome and slow.

In modern magnetic separators the dropping of the magnetic material is generally effected by mechanically causing it to pass outside of the magnetic field, which may be done in various ways. (1). The magnetic material may be prevented from coming in direct contact with the magnet by means of a travelling belt, or a revolving cylinder of non-magnetic substance, to which the magnetic material will cling so long as in the field, but from which it will drop as soon as removed from the field. (2). The magnetic material may be attracted directly to the pole, which by revolution or change in electrical connection may suffer a change in polarity, or become non-magnetic, thus dropping the attracted particles after removing them from the stream of non-magnetic. (3). The magnetic material may be attracted directly to the pole, and be removed therefrom by means of a brush, or scraper.

Magnetic separators may be classified in many other ways, e.g. those which operate on wet material and those which operate only on dry; those with fixed magnets and those with movable magnets; those which develop high magnetic intensity, and those which develop only a low intensity. Different types of machine are suited for different purposes.

The electro-magnet consist essentially of a core of soft wrought iron surrounded by a coil of insulated copper wire, through which electric current is passed. So long as the current flows the iron is a magnet, with north and south poles, and if suspended freely it would align itself with the needle of the compass. By reversing the direction of the current the poles are reversed. The straight electro-magnet is an uneconomical form, because of the great dispersion of the lines of force. In order to concentrate the latter the core is bent into U-form and the poles are caused to approach closely together. Further concentration is effected by tapering the pole pieces. The space between the poles, through which the lines of force pass, is called the magnetic field. The intensity of the magnetic field depends upon the size of the magnet, the form of the magnet, and the number of ampere turns in the coil, i.e. the product of the amperes of current flowing in the coil times the number of turns around the core. The attraction of any magnetic substance varies with the intensity of the magnet and its distance from the magnet.

Magnetic lines of force are analogous to electric currents, and like the latter form closed circuits. The magnetomotive force in a magnetic circuit is directly proportional to the number of ampere-turns. The reluctance is directly proportional to the length of the circuit and inversely proportional to the sectional area, and also to the permeability of the substances in the circuit. Magnetomotive force $\div$ reluctance $=$ magnetic lines of force. By the term permeability is meant a numerical coefficient which expresses how much greater the number of lines generated in a substance by a given magnetomotive force is than those which would be generated in air by the same force. It is not possible to obtain much more than 20,000 magnetic lines per square centimeter in soft, annealed wrought iron, without using enormous magnetomotive force, and in designing electro-magnets it is not generally good economy to go above 16,000. Leakage is the number of extra lines which must be produced in order to attain a desired strength of field. This will depend upon the shape of the magnet and the length of the air gap. In a magnet of which the poles are bent around to face each other, with an air gap of only 0.25 in., the leakage may be about 0.3 of the useful lines; for larger air gaps it will be greater; in a poorly designed magnet it may be much greater. Where a very strong field is desired, the lines of force may be condensed by beveling off the poles so that their sectional area is less than that of the core, but this is done at some loss of power, inasmuch as halving the area does not by any means double the strength of the field.

It will be manifest that the intensity of the magnetic field is the greater, the closer the particles to be attracted can be presented to the magnet. The interposition of a belt, or drum, or similar device, which may be necessary to effect the removal of the attracted material from the magnetic field, inevitably reduces to some extent the intensity of the latter.

Besides the design and arrangement of the magnets, the method of presenting to them the material to be separated is of great importance. This is done usually by distributing the material in a thin sheet by means of a shaking tray, a travelling belt, or by spreading it over a magnetic drum. The rapidity with which it is passed into the magnetic field is a highly important consideration. For example, in testing a mixture of magnetite (strongly magnetic), rhodonite (feebly magnetic), and blende (very feebly magnetic) on a machine of high intensity, it was found that at a certain speed of the belt only the magnetite was attracted. At a reduced speed, the rhodonite was partially attracted. At further reduction the rhodonite was completely attracted and the blende was still uninfluenced. At still further reduction the blende was completely attracted. The size of the particles is also of importance. Theoretically, there should be no difference on this score, save that a closer adjustment of the distance between the magnets and the sheet of ore is possible with evenly sized particles, but practically it is found that reasonably close sizing gives the best results in magnetic separation, and in well designed mills this is arranged for. The treatment of the very fine ore is a serious difficulty in magnetic separation, just as it is in ordinary wet concentration, though to a less extent.

Important principles in magnetic separation are the production of a magnetic current of the minimum intensity and maximum density; the production of a homogeneous magnetic field; passage of the material to be separated through the magnetic field as near as possible to the attracting pole of the magnet, and in an even, regular sheet, at the proper speed.

As has been previously remarked, the variety of magnetic separators that have been employed, and even of those employed at the present time, is so large that a general description and discussion of them would require the space of an elaborate treatise. The following brief summary of a few types of commercial machines will sufficiently indicate the general principles.

Monarch. (Ball-Norton).—Stationary electro-magnets, inside of a revolving drum. Used for separating magnetite.

Sautter.—Similar to the Monarch. Used at Pierrefittes and Friedrichssegen for separating blende from artificial magnetite (calcined siderite).

Siemens & Halske.—Similar to the above. Used at Meiern (Tyrol) for separating blende from artificial magnetite.

Heberli.—Similar to the above. Used at Friedrichssegen for the same purpose.

Ferraris No. 1.—A revolving cylinder, consisting of a hub and series of radial magnets, which come into and pass out of circuit by connection through a commutator, revolving with the drum. Two diametrically opposite magnets are thus caused to be continually out of circuit, forming a neutral plane, which is usually set at an angle of 45° with the horizontal. Non-magnetic particles fall off the drum by gravity; magnetic particles are attracted to the drum until they pass into the neutral plane, when they too fall off. Used at Monteponi for separating roasted iron ore from zinc ore. The machine is somewhat complicated and hence was replaced by a different type (Ferraris No. 2).

Payne.—Stationary magnet, inside of revolving drum. Magnet especially designed to insure efficiency and distribution of lines of force approximately normal to surface of drum. Used at Austinville, Va., for separation of roasted limonite from zinc ore.

Wenstrom.—Stationary magnet, inside of revolving drum, the latter being made up of alternate magnetic and non-magnetic bars, parallel to the axis. The magnetic bars are practically prolongations of the pole pieces. Non-magnetic material falls off the drum, while magnetic particles are carried around underneath until the attracting force becomes so weak that they too fall off. This machine, which is adapted to the treatment of coarse stuff, not necessarily dry, is used extensively in Sweden and the United States for separation of magnetite.

Buchanan.—Two hollow rolls of iron, which revolve toward each other, being supported on journals which rest on the ends of two horse shoe electromagnets, the latter wound so that one roll is supported by two north poles, the other by two south poles. The ore is fed on top of each roll. The non-magnetic falls into a chute below and between them; the magnetic adheres to the rolls until they have passed out of the magnetic field. Used for separating magnetite. This machine is the prototype of the Mechernich separator, which is one of the modern and successful machines for the separation of zinc ore.

Mechernich.—This is a modern development of the Buchanan separator, described above. The principle is illustrated in the accompanying engravings, Figs. 17–21. When the poles of the magnet are two parallel cylinders, the lines of force are highly condensed on the line where the cylinders are nearest together, diminishing in density away from that line. Consequently when the cylinders, or only one of them are revolved, magnetic material is attracted to the poles, and as the latter carry it outside of the magnetic field the attracted material drops off. In the Mechernich separator, the hollow cylinders of Buchanan are replaced by solid cylinders, which serve both as cores and as poles, the exciting coil being wound on their middle portion. The revolution of the poles, instead of being opposite to the direction of the ore feed, is caused to coincide with the latter. The two poles, instead of being on the same horizontal line, are set with their axes on a diagonal line, the north pole uppermost and a little ahead of the south pole. In the latest constructions, the south pole is stationary and either of semicylindrical or pear-shape cross-section. This pole is encased with a non-magnetic covering, so that the non-magnetic material may slide directly off from it, while the magnetic is attracted to the north pole. Behind the latter a brush is arranged to remove any material which may fail to be dislodged by gravity and centrifugal force. The whole apparatus is enclosed in a dust-proof housing of sheet zinc. The machine develops a field of high intensity and is capable of separating feebly magnetic minerals from non-magnetic and from strongly magnetic. Its mechanical and magnetic efficiencies both are high. It is used at several places in Europe and has been used at Broken Hill, New South Wales. In the United States it is employed by the United States Zinc Co., at Pueblo, Colo., for the separation of Leadville, and other mixed sulphides, which, however, are subjected to a magnetic roast before passing to the separator. The Mechernich separator is controlled by the Electro-Magnetische Gesellschaft, of Frankfort am Main, Germany. No general attempt has been made to introduce this separator in America, the patent rights having been under option to the United States Zinc Company, I understand, and the Commission had no opportunity to test it. I visited Pueblo and saw it in operation, but was not accorded permission to report its results.

Siemens.—A hollow, revolving drum, made up of annular wrought iron discs, separated by brass annular discs. The exterior of the drum is wound with insulated copper wire, making the whole a hollow magnet, of which each pair of annular iron discs are pole pieces, pointing inward. The ore is fed into the drum as in a trommel. The non-magnetic passes straight through. The magnetic is carried to the top, where it is scraped off and removed by a conveyor.

Dellvik-Gröndal.—A cylindrical magnet, revolving around a vertical axis. Parallel with it is a wooden cylinder, studded with iron pegs. Material is fed against the first cylinder, the magnetic being carried around until adjacent to the wooden cylinder, when the induced magnetism on the pegs causes the particles to hop over, after which they fall off or are washed off as the revolution of the wooden cylinder takes them out of the magnetic field. Works on wet material. Used at Pitkäranta for separating magnetite from blende and pyrites.

Conkling.—A continuous, horizontal, travelling belt, over which are set a series of magnets, a cross-belt (at right angles to the conveying belt) passing under the pole piece of each magnet. The ore is fed on the conveying belt. As it passes under the first magnet, the magnetic particles are lifted up against the cross-belt, and are by the latter drawn to one side; similarly with the second and third magnets. The non-magnetic material continues on the conveying belt and is delivered over its end. This is the prototype of the class of cross-belt magnetic separators.

Ferraris No. 2.—Similar to the Conkling, but the poles of the magnets are tapered and bent around so as to come quite close together. Used at Monteponi for separating roasted limonite from calamine. This machine displaced the Ferraris No. 1.

Rowand.—Commonly known as the Wetherill. Similar to the Conkling, but the magnets (Wetherill type) are placed both above and below the conveying belt, which gives a stronger field, as does also the form of the pole pieces

Wetherill.—The chief feature of these machines is the production of a very strong magnetic field by concentrating the lines of force, which is done by placing the two poles of the magnet as near together as possible and beveling off the ends of the pole pieces. This idea was originally employed in several types of machine, known as the Wetherill, all of which have now been replaced by the Rowand-Wetherill (known commonly as the Wetherill) referred to above. The invention of the Wetherill separator and its success in treating feebly magnetic material certainly gave the impetus to the remarkable development which the art of magnetic separation has received during the last ten years. The status of the Wetherill patents is discussed elsewhere in this report.

Cleveland-Knowles.—Two cylindrical magnets, with concentric poles (the coil being wound between them) are set with their axes vertical over a conveying belt. The magnets are set so as to off-set the belt on one side. Having picked up the magnetic material, passing under them on the belt, their revolution carries it outside the line of the belt, where it is removed by a scraper. This machine is extensively used in the United States for the separation of zinc ore, and has been employed in the Payne mill at Sandon, B.C.

Finney.—A revolving magnetic drum, the core and coil being inside the drum and the pole pieces dove-tailing around the periphery. A belt passes over this drum and over a parallel roller, so that its upper surface is horizontal. The ore is fed on the belt. In passing over the magnetic drum the non-magnetic material falls off, while the magnetic sticks to the belt until the latter begins to leave the drum, when it also falls.

Ferraris No. 3.—This is similar to the Finney. It is now used at Monteponi for the separation of roasted limonite from zinc ore.

Courtney.—Similar to the above, but without the belt, and designed for wet work.

Knowles.—Known also as the "New Century." This is a separator of the induction type. There is a magnet with chamfered pole pieces, between which passes an endless belt, which is caused by the supporting and guiding rollers to travel in a trapezoidal path. The belt is thickly studded with soft steel rivets, with cup-shaped washers, the latter having serrated edges. The ore is delivered by a shaking tray under the inclined portion of the belt. When the belt passes between the pole pieces of the magnet, its studs become magnetized and attract the magnetic particles of the ore. As the belt moves away from the pole pieces, in its horizontal path, the studs gradually drop the particles which they have attracted, these falling into appropriate chutes. The construction and operation of the separator will be clearly understood from Fig. 22. The Knowles separator is extensively employed in the United States for the separation of roasted zinc ore, and has been used in British Columbia at the Payne mill, at Sandon.

Other Separators.—The separators which were used in the experimental work of the Commission are described in the next chapter.

SEPARATORS USED IN EXPERIMENTAL WORK.

It was contemplated originally by the Commission to test only the general principles of electro-magnetic and electrostatic separation as of possible application to the ores of British Columbia, but later it was considered advisable to try a variety of commercial machines on a competitive basis. The smallness of the appropriation limited the scope of such tests, which under the circumstances could not, obviously, be conducted with the idea of determining the best form of magnetic separator among those on the market, but had in view rather the determination of the difference in results that might develop from the use of different forms of machines.

Invitations were extended to the following concerns to submit machines for trial, preference being expressed for a small machine, of the respective makes, as being most easily managed under the conditions that had to be complied with:—

1. Wetherill Separating Co., New York; Magnetic Separator.
2. International Ore Separator Co., Chicago; Magnetic Separator.
3. United Iron Works Co., Springfield, Mo.; Magnetic Separator.
4. American Concentrator Co., Joplin, Mo.; Magnetic Separator.
5. Dings Electromagnetic Separator Co., Milwaukee, Wis.; Magnetic Separator.
6. C. G. Buchanan, New York; Magnetic Separator.
7. Imperial Ore Separator Co., New York; Magnetic Separator.
8. Blake Mining and Milling Co., Denver; Electrostatic Separator.
9. Huff Separator Co., Boston; Electrostatic Separator.
10. Sutton-Steele Co., Dallas, Tex.; Electrostatic Separator.

This invitation was accepted by Nos. 1, 2, 5 and 8, of whom Nos. 1, 2, and 8 had machines available in Denver, while No. 5 prepared a special machine (of standard type, special only in size) for the purpose. No. 9 declined the invitation, on the ground that its machine was not yet ready for general introduction. No. 10 offered to prepare a special machine for the purpose, but it was considered by the Commission that this would occasion undesirable delay in the experiments, and in further view of the newness of this type of separator, which has not yet come into general use, the offer was declined. No. 3 offered the use of a machine belonging in Denver, but temporarily loaned to the U. S. Geological Survey's testing plant at Portland, Oregon; however, it was not returned to Denver in time for our experiments. No. 3 offered the use of a machine installed near Denver, but this was too inconvenient for our work, and the offer was consequently declined. No reply was received from No. 7.

After preliminary experiments to determine the general lines of treatment, competitive trials were made with several of the machines mentioned above. In these competitive trials, the makers of the various machines were represented by agents, and the methods of performing the tests received their approval. Indeed, all possible effort was made to insure absolute fairness. The details of these experiments and their results are stated in the report of Mr. Argall.

The machines employed in these experiments are described as follows:—

Wetherill Separator.

The Wetherill separator commonly in use is described as the "Cross-belt" form, or "Type E." The principle of its operation is shown in Fig. 23. The material flows from the holes of the hopper to the feed roller, which discharges it in a uniform layer over the whole width of the conveyor-belt "B," passing between the poles of the magnetic system. The latter consists of two horse-shoe electro-magnets, the poles of which are arranged one above the other. The poles of the upper magnet have the shape of a sharp wedge, while the lower ones are flattened. With this arrangement of the magnets, the paramagnetic minerals, when brought into the magnetic field are influenced in such a manner that at a comparatively small distance from the lower pole, the magnetic force of the upper poles supercedes that of the lower, this distance being given by the thickness of the conveyor-belt, which passes between the poles. The magnetic particles jump toward the upper poles as soon as they are carried by the conveyor-belt into the magnetic field. The cross belts "B" prevent the magnetic particles from adhering to the upper poles and carry them out of the magnetic field. The removal of the magnetic particles is further facilitated by a sharp piece of iron mounted on the front part of the upper poles, by which the intensity of the magnetic field is gradually decreased.

The capacity of the machine depends on the thickness of the layer of material on the conveyor-belt and on the speed of this belt, together with the speed of the take-off belts. These factors are regulated according to the nature of the ores.

This type of separator is constructed with one, two or three double magnets, giving two, four, or six poles. The principal dimensions are given in the following table, communicated by the Wetherill Separating Company.

No. of Machine	No. of Poles	Width of Poles	MAGNET WOUND FOR AMPERE TURNS			Max. Amp. at 110 Volts Direct Current	Floor Space	Height to top of Machine	Shipping Weight, Lbs.
			1st	2nd	3rd				
E No. 1a	2	18″	30,000			6	5′ 0″ x 11′ 10″	8′ 6″	14,000
E No. 1b	2	18″	60,000			14	5′ 0″ x 12′ 7″	8′ 6″	15,000
E No. 1c	2	18″	100,000			30	5′ 0″ x 13′ 4″	8′ 6″	16,000
E No. 2a	4	18″	30,000	60,000		20	5′ 0″ x 17′ 3″	8′ 6″	22,000
E No. 2b	4	18″	30,000	100,000		36	5′ 0″ x 18′ 0″	8′ 6″	23,000
E No. 2c	4	18″	60,000	100,000		44	5′ 0″ x 18′ 9″	8′ 6″	24,000
E No. 3	6	18″	30,000	60,000	100,000	50	5′ 0″ x 23′ 4″	8′ 6″	30,000

A large number of these machines are in operation in the United States and Mexico. They were formerly put out only on a royalty basis, but now they may be obtained on different terms. The machine is rather costly, but it is highly efficient, and is capable of very delicate adjustment. With respect to the Wetherill machine used in the experimental work of the Commission, and the capacity of the full-sized machine in practice, Mr. Argall reported as follows:—

"The Wetherill separator used in these tests was built expressly for experimental work. It has only one double pole magnet, against three in the standard machine. This laboratory machine was much better adapted for our work than either of the other machines, (International and Dings) and therefore was more often used, particularly in the experiments on roasted ore, where the Dings low intensity machine gave just as good results up to its limit of 0.5 ampere, but the Wetherill allowed us to go all the way up to four amperes, which in many of the tests was absolutely necessary; hence it was found preferable to conduct the experiments on the machine that covered the full field.

"In order to obtain data as to the capacity of the Wetherill separator, I visited the works of the Colorado Zinc Company, where four standard Wetherill machines are in operation. Each machine has three double pole magnets, giving six points at which magnetic material can be lifted off the main belt and separated from the non-magnetic material. The ore discharged over the tail end of the Wetherill machine, is immediately treated by a Blake electrostatic machine. The capacity of the standard Wetherill separator on Leadville ores varies from 700 to 1000 lbs. per hour, using 7.5 horse power to excite the magnets and operate the machine.

"The capacity of the Colorado Zinc Company's plant is 300 tons per week, the process being wet crushing and concentration to remove the lead (and silica when present), settling the slimes for zinc product, drying the mixed zinc-iron concentrate and treating it on four Wetherill machines, followed by four Blake machines. Consequently if all the crude ore passed over the Wetherills, it would be less than 11 tons per day. Making allowance for the lead concentrate and for the slime, it would appear that each of the Wetherill machines, under the above conditions, averages in steady work, fully 9 tons per day of Leadville mixed sulphide ores, at the plant of the Colorado Zinc Company, The Leadville sulphide ores vary from a mixture of zinc and iron sulphides, pyrites and blende (ferruginous), to varying proportions of blende, pyrites, pyrrhotite, galena, silica and limestone. Some of these ores can be separated on crushing to pass a screen aperture of 0.043 in., but the majority perhaps must be reduced to pass 0.025 in., screen apertures, in order to obtain an effective separation of the minerals by wet concentration and magnetic treatment.

"The Wetherill separator is an excellent machine, covering a wider field of usefulness and being capable of more delicate adjustment than any other magnetic separator I know of."

The International Separator.

This machine, which is manufactured by the International Separator Co., of Chicago, Ill., consists of a cylindrical armature made up of thin laminated discs of a special annealed wrought iron mounted upon a steel shaft and revolving horizontally between the pole pieces of a large inverted horseshoe field magnet. The discs of the armature are pressed tightly together by heavy cheek-plates at each end. The edge of each disc is toothed. In assembling the discs, the teeth are so staggered on adjacent discs that the surface of the finished armature presents a great number of small steel points. The pole pieces are at the extremities of a horizontal diameter of the armature. By induction the magnetism of the pole pieces causes magnetic poles to appear on each side of the surface of the armature. The exciting coils enclose magnet cores which are located just below the pole pieces, there being no direct electrical connection with the armature itself.

In operation, the armature is revolved by a belt and pulley. Material to be separated is fed, from the hopper on which is an adjustable gate for regulating the feed, upon the top of the revolving armature, and is carried by its movement around to one side. Here the more attractable particles are held by the magnetic pole on that side, and are carried around under the armature. The less attractable material, on reaching the side, slides off. The magnetic pole on one side is a north pole; and on the other, a south pole. There is, therefore, on the bottom of the armature a place where the polarity changes from north to south. As this place of reversal is approached the magnetic attraction of the armature becomes weaker, until at the point of reversal there is no attraction whatever. Here even the most highly attractable material drops off.

Long, narrow, adjustable hoppers are supported under the armature by brass stirrups from the shaft bearings. These hoppers can be moved to take different products as desired from the under surface of the armature, as the products successively drop off under the gradually weakening magnetic attraction. In this way several diderent minerals can be separated in one operation. The position of each hopper is controlled by links mounted on brass shafts extending across the separator. These shafts are provided with indicators for showing the position of the hoppers at any time; and with set-screws, for locking the hopper in any desired position.

In a magnetic separator, the heavier the field magnet the less electricity is required to give the armature the required attracting strength. This weight can be put into either the copper wire or into the soft-steel field-magnet frame; preferably it is put into both. The field magnet of this separator weighs 9,000 lb. As it takes a great deal of electricity to make the magnetic lines of force pass through the air, the pole-pieces are brought up as close to the armature as possible. Just sufficient room is left to allow the material to be separated to pass between armature and pole-piece. The points on the armature prevent the easily attracted material from sticking to the primary pole-piece. The magnetism is much more concentrated on the points than on the face of the pole-pieces. An attractable particle, therefore, even when put on the bare steel face of the pole-pieces, jumps across to one of the points on the armature. Being able in this way to make the air gap between pole-pieces and armature small without having particles accumulate on the pole-pieces, only a small current is required to give the armature a high magnetic attractability.

Any tendency that non-magnetic material may have to adhere to, and to follow the armature, is overcome by the centrifugal force of the armature in its revolution. By adjusting the speed of rotation this centrifugal force can be given any desired value, and can be balanced against the attractiveness of any one of a series of magnetic materials which it is desired to separate. At 135 r.p.m. the centrifugal force is three times the weight of a particle.

At present 18 of these separators are running in the Yak mill at Leadville, making separation of pyrite, blende and pyrrhotite, treated raw. Others are in use, cleaning manganese ore and concentrating monazite sand. This separator is illustrated in Figs. 25–27.

A standard machine weighs 10,000 pounds. It is 3 ft. 2 in. wide by 3 ft. 9 in. long and 5 ft. high. According to the makers, in practice one horse power is used for excitation and one horse power for mechanical operation. They state also that the capacity of a single machine is from two to four tons per hour, depending on the character of the ore and the thoroughness of its preparation for separation. As more than one machine is sometimes necessary to effect a commercial separation the capacity per machine in operation would be somewhat less than this. The cost of the machine, f.o.b. works, is $4000. No royalty is required in Canada.

In order to obtain information as to the capacity of the International separator, Mr. Argall visited the Yak mill at Leadville, Colo., where a large number of them are in use, as mentioned above. Mr. Argall reported as follows:—

"The mill superintendent informed me that on 20% zinc ore the capacity of a machine is 2.5 tons per hour, and 3 tons per hour on 30% zinc ore. There are 18 International machines in this mill, the daily capacity of which is 200 to 250 tons. Four machines receive the initial ore, which at the lower tonnage corresponds to 50 tons per day for each machine. The other machines are used in re-passing the products made from the primary machines and of course cannot be grouped so that each will be supplied with full feed.

"In small plants one machine could be used for the entire process, and assuming all the products required one or more passes, then a machine of capacity of 5,000 lbs. per hour, with one pass, has a capacity of 2,500 lbs. per hour with two passes, or 1,666 lbs. per hour with three passes, or 20 tons per day, while a product requiring four passes would give a daily capacity of about 15 tons, which is a little more than the 18 machines give in the Yak mill, where $250 \div 18 = 13.88$ tons per machine, at 250 tons total daily capacity.

"The ore fed to the separators in the Yak mill was reduced, dry, to pass a 0.043 in. screen aperture. The 18 machines, with a 250 volt current, used 64 amperes for exciting the coils, or less than a kilowatt for each machine. One horse power is ample to rotate the armature of the machine.

"The International separator has large capacity, is simple in operation, cheap in maintenance and gives good separation of the minerals, as will be seen from the tests, which are reported elsewhere."

The International machine, on which some tests were made by the Commission was a full-sized machine, and therefore not well adapted for testing small lots of ore.

The Dings Separator.

The Dings separator, made by the Dings Electro-magnetic Separator Company, of Milwaukee, Wis., is a low intensity machine of the induction type. The ore is presented to the magnets by means of an inclined, shaking tray, which is fed mechanically from a hopper at the upper end. In the standard machine the tray is 16 in. wide. As in all magnetic separators the feed must vary according to the character of the ore. The feed to the tray is regulated by means of a hand wheel on the hopper, and means are also provided for adjustment of the inclination of the tray.

The primary magnet consists of a core and coil, fixed above the tray, with pole-pieces projecting toward the tray, the two pole-pieces having each a thin edge, shaped into arcs of a circle, the chords of which are about as long as the tray is wide. The secondary magnets form essentially a wheel, which is set to revolve in a plane parallel with that of the tray. The upper side of this wheel has a circle of U-shaped grooves, which embrace the curved edges of the pole pieces of the primary magnet, without touching them. In other words, the edges of the pole-pieces of the primary magnet are encircled by the grooves of the secondary magnets. On the lower side of the wheel the secondary magnets have the form of a series of cylindrical studs, about 1 in. in diameter, projecting an inch or an inch and a half from the wheel. The diameter of the wheel is greater than the width of the tray.

The primary magnet being energized by the current, the wheel of secondary magnets and the shaking tray being set in motion, the secondary magnets being energized by induction while their U-shaped grooves embrace the pole-pieces of the primary magnet, material that is permeable in the magnetic field is attracted by the secondary magnets, and by the revolution of the latter is carried beyond the edge of the tray, where it is dropped into a chute, the secondary magnets having then passed outside the magnetic field, and consequently being no longer magnetized. In the meanwhile the non-magnetic material passes down the shaking tray and is discharged over the lower end of the latter. It will be observed that material passing down the tray is twice exposed to magnetic attraction, but with very magnetic mineral by far the largest portion is removed at the first pass. The accompanying illustration, Fig. 28, shows a machine with two magnets in series, known as the double machine, which gives four magnetic zones. This machine has been installed at the works of the Mineral Point Zinc Company at Mineral Point, Wisconsin.

The tray is supported on a heavy steel plate, which acts as a magnetic armature with respect to the pole-pieces, so that the path of the magnetic lines of force is from pole to pole through the secondary magnets, then through the ore being treated, and then through the steel armature under the tray. The tray is supported by roller bearings resting upon the armature, or steel plate.

The wheel of the secondary magnets is made of heavy aluminum castings, and the secondary magnets themselves of laminated armature steel. Attention is given in the design to the reduction of the magnetic resistance between the primary and secondary magnets to the minimum.

The following data were communicated by the manufacturer:—The capacity of the single magnet and double magnet machine is the same. On the basis of crude ore containing 25% zinc, it is about 2,000 lb. per hour, but it is advisable not to overcrowd the machine, and in a plant roasting 40 tons of ore per 24 hours the installation of two machines is recommended. The principle data for the two forms of machine are given in the subjoined table:—

Items	Single Magnet	Double Magnet
Weight net.	1400 lb.	3200 lb.
Weight, boxed for shipment	1700 lb.	3600 lb.
Dimensions, width	36 in.	47 in.
Dimensions, length	72 in.	120 in.
Dimensions, height (to top of hopper)	44 in.	66 in.
Mechanical power required	0.5 h.p.	1 h.p.
Electrical power required, maximum.	450 watts	1400 watts
Price, f.o.b. works.	$600	$1200

These machines are sold outright, no royalty being required. The machine used in the experimental work at Denver was of the laboratory type, and was very well suited for the purposes. It gave excellent results on roasted ore.

MAGNETIC SEPARATING PLANTS IN THE SLOCAN.

Magnetic separating plants have already been erected in the Slocan, but because of failure properly to recognize the conditions, and to undertake the problem broadly and energetically, they have not yet proved thoroughly successful. Of the two plants which have been in operation, that at the Payne mine is referred to in Mr. Argall's report, it being installed in the Payne concentrating mill for the treatment of the product of that particular mine. The other plant, installed for custom work, is at Kaslo, on Kootenay Lake, and is owned by the Kootenay Ore Company, Limited, of which Mr. George Alexander is the manager. A view of the plant is shown in Fig. 29. It was originally constructed as a sampling mill, and is still used for that purpose, one end of the works having been remodelled for the magnetic separating plant.

The latter comprises a set of ore bins for the receipt of ore, a White-Howell cylindrical roasting furnace, a cooling cylinder through which the hot ore delivered from the furnace passes, rolls for crushing the cooled product, and an elevator for raising the crushed ore to storage bins. From the storage bins, the ore is drawn to two Dings magnetic separators. These stand on an elevated platform, permitting the products to pass by gravity to bins beneath it.

This plant was under construction in 1905 and at the time it was visited by the Commission, about the end of October, the machinery was put in operation for the first time. An experimental test was made with a small quantity of blende-siderite concentrate from the Ruth mine, which gave results approximately like those subsequently obtained at Denver, viz., the production of a blende concentrate containing 50% zinc, and the extraction of 87% of the zinc in the ore. The works were, however, far from complete and did not go into regular operation. Additional Dings separators were purchased and installed, but about the end of 1905, a readjustment of railway freight rates made it impossible to operate the works at a profit and operations were suspended awaiting settlement of the dispute.

The general situation of the zinc ore producing industry in British Columbia, the value of the ores of various grades, the cost of production and the cost of magnetic separation have been fully set forth in previous sections of this report, and I have also expressed my views as to the proper policy to be adopted by the railways. It is, therefore, unnecessary to discuss this particular case.

The plant of the Kootenay Ore Company will undoubtedly be able to treat the blende-siderite ores, for which it was designed, but it is quite incapable, with its present equipment, of treating the miscellaneous ores of the Slocan, as will be readily understood from the previous discussion of this subject.

It was reported during the visit of the Commission that a magnetic separating plant would also be installed in connection with the Roseberry mill at Brockman, on Slocan Lake, with a view toward the treatment of custom ores.

Kaslo and Brockman are both good locations for custom mills, the former to receive ores over the Kaslo & Slocan Railway, and the latter over the Sandon branch of the C.P.R.

PATENT RIGHTS.

Almost every magnetic separator now on the market is patented. The art is, however, an old one, and the broad practice is incapable of monopoly. In most cases the patents are only on particular types of machine, or portions of machines, and in view of the long history of this method of separation, which dates back to 1847, possibly earlier, and the great number of separators devised since that time, it is likely that many forms manufactured under patents at the present time would prove upon investigation to have been anticipated.

The only broad magnetic patent, now alive, is that of J. P. Wetherill, who on March 3, 1896, obtained American patent No. 555,792 (application filed Feb. 10, 1896.) The uncommonly short time between the dates of filing and issue of this patent show that it was distinctly considered a novelty by the examiners at Washington and probably was allowed without citation of references to previous inventions. The Canadian patents are Nos. 55,201 and 55,573, the former being dated March 9, 1897, and the latter April 9, 1897.

The leading claim of Canadian patent No. 55,201 is as follows:—

1. In a magnetic separator, an electro-magnet having a pole-piece tapering toward its free end, said free end being of less sectional area than the magnet core so as to highly condense the lines of magnetic force, an ore conveyor below and in close proximity to said tapering end and conveying the ore through a portion of the highly concentrated field, the tapering end of the pole-piece being arranged transversely to the direction of travel of the conveyor, and a second conveyor for withdrawing the attracted particles.

The leading claim of Canadian patent No. 55,573 is as follows:—

1. The herein described method of separating ferruginous materials of inferior magnetic susceptibility or permeability from non-ferruginous materials consisting in bringing the mingled materials into a condensed or concentrated magnetic field and deflecting the ferruginous material while under the influence of said condensed field into a path of movement different from that of the non-ferruginous material.

These patents claim broadly the art of separating feebly magnetic material from non-magnetic material, together with the particular form of separator described, the essential feature of which was the electro-magnet with tapering pole-pieces.

The validity of these patents has not been tested in either the Canadian or American courts. The only judicial decision, which has been rendered in respect to them, was in a suit before the Imperial German Patent Office, brought by the Mechernicher Bergwerks-Aktien-Verein (owner of the Mechernich separator) against the Metallurgische Gesellschaft (owner of the Wetherill separator) to test the validity of the Wetherill patents. The decision of the German patent office was summarized as follows in the *Engineering and Mining Journal* of Nov. 17, 1900:—

Claim 1 of the patent No. 92,212 covering the process of separating the so-called weakly magnetic substances by direct magnetization, is upheld; the wording of claims 2 and 3 it was found necessary to change so as to preclude a possible misinterpretation, the new context to express without ambiguity that the constructions covered by either of the two claims are patented only in so far as they serve for carrying out the process as defined by claim 1. The remaining claims were not assailed by the complaint. Costs are divided among the parties, viz.: Complainant to bear four-fifths, defendant one-fifth.

The arguments in support of the decision are stated as follows: "For the consideration of the question, whether the process characterized by claim 1 was new and patentable at the time of the application, it is immaterial that long before the date of application the existence of weakly magnetic and non-magnetic minerals was known, besides the strongly magnetic ores; likewise the classification given by M. Faraday 60 years ago dividing all substances into two classes, paramagnetic and diamagnetic, since constituting a common acquisition of scientific and technical knowledge, is by no means prejudicial to the patent.

"The Application Department knowing these premises has allowed the first claim on the well-justified ground that Wetherill was the first to recognize the applicability of these known facts to the magnetic separation of weakly magnetic substances, without first transforming them into the strongly magnetic state and to demonstrate their separation from non-magnetic substances upon a commercial scale.

"Magnetic concentration up to the time of Wetherill was developed only to a very small extent and still more limited in its practical application. It was principally applied to the dry concentration of iron ore which by roasting was convertible into ferroso-ferric oxide—FeO, Fe_2O_3—this being considered the only iron compound corresponding in its magnetic properties to the native magnetite and separable by means of the magnet.

"In Germany only a few works have attempted to concentrate by this method the spathic iron—$FeCO_3$—interspersed with zinc blende.

"This process involved the difficult task of converting the spathic iron by roasting into ferroso-ferric oxide (artificial magnetite).

"It would be of no consequence to investigate here the difficulties connected with this operation and their chemical causes; it may suffice to mention that in many cases it was a complete failure, in most other instances it proved unsatisfactory in spite of great expense. The news that Wetherill's process required nothing more than the ordinary crushing before separation, avoiding the extremely difficult roasting process, created great satisfaction among experts.

"Before Wetherill the efforts of specialists in this branch were guided on the one hand by the sole aim to improve the construction of the magnetic apparatus utilizing the achievements of physical and electrical sciences; the progress made in metallurgy relating to the chemical nature of the roasting process on the other hand stimulating efforts toward converting iron compounds, thought to be practically non-magnetic, into artificial magnetite.

"This problem was simplified to a remarkable extent also from an economic point of view when Wetherill demonstrated the possibility of direct separation of the weakly magnetic substances.

"The statement of the complainant relating to the alleged prior use of methods similar to the patented process producing essentially the same effects, is incorrect. Complainant has not given evidence in support of such statement. Referring especially to the old Buchanan separator the argument of the defendant has not been invalidated that the Buchanan separator before Wetherill's application for the patent had never been applied to the separation of weakly magnetic substances from non-magnetic admixtures.

"Complainant moreover adduced a laboratory method for separating feebly magnetic substances from non-magnetic ones, which in some respects bears slight resemblance to the context of claim 1 of the disputed patent. This method, described by Mann in 'Neues Jahrbuch für Mineralogie, Geologie and Paleontologie,' 1884, Volume 2, page 161, in its entirety, however, would not justify the annulment of claim 1 of the defendant's patent.

"In the first place the publication of Mann has no reference to concentrating complex ores for technical or industrial purposes, but serves merely for solving an interesting mineralogical problem by analyzing the constituents of certain minerals, it being understood that no further utilization or technical application of the products was contemplated. A conclusion as to the practicability of Mann's method for dry magnetic concentration of ores was by no means evident.

"Furthermore magnetic concentration has to deal with entirely different substances, with other classes of minerals than those which Mann experimented upon in connection with his research work.

"Finally the method itself is in a number of points essentially different from the Wetherill process."

ELECTROSTATIC SEPARATION.

A new class of ore separator has been developed and put in practical application during the last five years, in which the principle of electrostatic repulsion is utilized. It has been found that many metallic sulphides and other compounds possess a high electrical conductivity, while the gangue minerals generally, and certain sulphides, e.g., zinc sulphides, are non-conductors. Now, when a conductive particle is brought in contact with an electrified body, it immediately assumes a similar charge with the result that it is repelled, while non-conductive particles are unaffected when brought in contact with an electrified body and are not repelled. Upon this principle a separation can be made. Particles of high conductivity are instantly repelled, while those of low conductivity are not repelled until sufficient time has elapsed to allow them to be pulled out of their original path and caught in a separate receptacle. The separation of two minerals does not necessarily require that one of them be a good conductor, but simply that there be a sufficient difference in their conductivity. Absolute dryness is, of course, an essential to good work.

It has been found by experiment that galena, pyrite, marcasite, chalcopyrite, pyrrhotite and magnetite among other minerals are conductors, while blende, either "rosin jack" or "black jack," is a non-conductor. This enables a separation of blende to be made from other sulphides, but does not permit a separation to be made between blende and calamine, or blende and the common gangue minerals, unless some artificial difference in the electrical properties of the minerals can be effected, which sometimes can be done. The iron that occurs in many blendes as monosulphide does not appear to affect its conductivity, excellent separation being made of Leadville ore, in which the blende contains 7 or 8 per cent. iron as monosulphide.

Irrespective of to whom the application of this idea may be due, its practical development must be credited to Prof. Lucien I. Blake and Mr. Lawrence N. Morscher, * while Mr. W. G. Swart † has been very instrumental in putting the Blake-Morscher process on a commercial basis.

The Blake-Morscher separator is constructed of wood in substantially the form shown in transverse vertical section in Fig. 30 and in perspective in Fig. 31. The ore distributed in the double hopper (by means of a small screw conveyor) is discharged over a suitable feed plate upon a metal drum or electrode (which is revolved). The conductors, non-conductors and middlings, which are repelled or non-repelled as the case may be, are separated by the adjustable dividers and fall into separate chutes, which convey them to another set of electrodes over which they pass, being separated thereby into final products, save the middlings, which are returned by a belt and bucket elevator to the top of the machine.

The capacity of the machine on any given ore is obviously a function of the length of the electrodes. The double machine, with 20-ft. electrodes, used for separating Wetherill middlings (Leadville ore) at the works of the Colorado Zinc Company at Denver, Colo., treats about 10 tons of material per 24 hours, with a total consumption of about 1.5 h.p. The cost of such a machine is only about $800 to $1000. The machines are not sold outright, however, but are placed on a royalty or rental basis.

Many difficulties had to be solved in developing a practical apparatus, so little being known as to the application of static electricity. The proper distribution of the electric charge and the successful insulation of the high voltage currents in the face of the dust, moisture, and working conditions generally were among the problems solved. A simple means of controlling the effect of the violent fluctuations of the static charge was devised and the voltage was reduced from 250,000 to 10,000 or 20,000, corresponding to a spark of $\frac{1}{8}$ to $\frac{1}{4}$ in., though the machines at Denver are operated with about $\frac{3}{4}$ in., spark. A static generator has been devised, which is constructed almost wholly of metal, with no glass whatever, self-exciting and able to run continuously in the open air without regard to atmospheric conditions and with no more attention than an ordinary dynamo receives.

However, the generator usually furnished with the separator is a mica-plate machine of good mechanical and electrical design, and able to run twenty-four hours daily for long periods with little more than the care which should be given a good dynamo. There are other sources of static electricity available for extreme conditions, which are entirely free from any atmos-

* United States Patents, Nos. 668,791, and 668,792, both of Feb. 26, 1901.

† *Engineering and Mining Journal,* Jan. 24, 1903

pheric interference, such as series-parallel and rotary condensers, Rumkorff coils, etc.

The following information is communicated by the Blake Mining and Milling Company, of Denver, Colo., which controls this separator:—

"There exists, naturally, or can easily be produced artificially, sufficient difference in electrical conductivity to allow of separating almost any one mineral from another by this method. This statement is true on a laboratory scale, but it may not be always commercially possible.

"No definite statement can be made as to the conductivity of any given mineral, because it is not always the same on specimens from different localities. For example, zinc blende from Broken Hill, Australia, can readily be separated from zinc blende from Joplin, Missouri, because of its superior conductivity.

"It is however possible to classify the minerals roughly, and the following table is given as a guide to their usual behaviour. It should be remembered that this behaviour can only be confirmed by actual trial.

LIST OF MINERALS.

CONDUCTORS.	NON-CONDUCTORS.
Native Metals.	Quartz.
Pyrite.	Quartzite
Pyrrhotite.	Calcite.
Chalcopyrite.	Limestone.
Galena.	Porphyries.
Graphite.	Slates.
Molybdenite.	Sandstones.
Copper Glance.	Garnet.
Silver Glance.	Spinel.
Gray Copper.	Zinc Blende.
Most Sulphides.	Zinc Carbonate.
Most Copper Minerals.	Barite (Heavy Spar).
Most Iron Minerals.	Gypsum.
Most Silver Minerals.	Granite.
Most Manganese Minerals.	Fluor-Spar.
Tellurides.	Most Silicates.
Black Sands	Most Gangue Rocks.

"Usually it is difficult, commercially, to separate one conductive mineral from another on account of the extreme rapidity of action, but a separation is ordinarily possible between two non-conductive minerals. For example, zinc blende can generally be completely separated from quartz, garnet or heavy spar, all being non-conductors. The great flexibility of machine construction and the wide choice of methods of applying the electric charge place almost all such separations within easy reach of the process.

"Oftentimes where two minerals are of close conductivity, and do not readily separate on any of the usual types of machine, one or the other may be rendered conductive by artificial means, such as heating, roasting or chemical treatment. Since the conductivity toward static electricity depends on the surface of the particles, and not on their interior, it is only necessary in such cases to change in some way the surface of the desired particle. More than 200 different methods of doing this have already been developed, varying according to the ore and the object to be attained."

The Blake-Morscher separator has handled, commercially, material as coarse as six mesh. Fines have been in certain cases successfully treated, but in general the handling of ore-dust is as serious a problem in connection with this machine as it is with the electro-magnetic and almost all other forms of ore separators.

Coarse crushing is preferable, but it must be fine enough to liberate the crystals, one from another. Close sizing is not essential, but as in other concentrating methods is apt to improve results. A good range of sizes is as follows: From 6 to 12 mesh; from 12 to 20 mesh; twenty mesh and all finer. This must, however, be determined for each individual ore. Sizing is more important on coarse material than on fine.

The Blake-Morscher separator has given excellent results at the works of the Colorado Zinc Company, at Denver, where it is used for the treatment of Leadville ore, and at many other mills, a total of about 60 of these separators being now in use. Leadville ore is separated into a zinc product assaying 50 per cent. zinc and a lead-iron product assaying only 7 per cent zinc. Wisconsin blende-marcasite concentrates, assaying 18 per cent. zinc, have been made to yield about 25.5 per cent. of blende assaying 59 per cent. zinc, and 74.5 per cent. of marcasite assaying only 4 per cent. zinc. The experiments conducted by the Commission on the ores of British Columbia, in connection with which the Blake-Morscher separator was tried, did not demonstrate that it would be useful in the commercial treatment of those ores, as will be observed from the results of the tests reported by Mr.Argall. It is to be remarked, however, that these tests are in no way condemnatory of the Blake-Morscher separator, because the latter was not tested in competition with the electromagnetic separators, but was simply tried in cases where it was thought that it might offer a solution of a difficult problem. When it was found that an ore could be satisfactorily treated by magnetic separation, electrostatic separation was not tried. It is possible that in some cases it might have given results as good, or nearly as good as magnetic separation, but superior results were hardly to be expected. The electrostatic separator is a highly efficient machine on certain ores, but its range of usefulness, in so far as zinc ores are concerned, is not so wide as is that of the magnetic separators.

The Huff electrostatic separator, operating on the same principle as the Blake-Morscher, has been tested experimentally on a large scale at Boston, Mass., during several years, but its owners are not yet ready to introduce it in practice. An invitation was extended to its owners to submit it for trial at Denver, Colo., but the invitation was declined. For this reason, no further investigation of this separator was made.

FLOTATION PROCESSES.

During the last two or three years there has been developed, chiefly at Broken Hill, New South Wales, a class of processes for the separation of the mineral constituents of ores, which have been aptly termed "flotation processes." The ore, finely crushed, is charged into an acidulated bath of water in a vessel similar to the ordinary spitzkasten employed in dressing works. An action takes place, however, which is precisely the reverse of what happens in the ordinary spitzkasten. Instead of the heavier minerals settling to the bottom, and the lighter minerals passing off, unsettled, with the overflow, in this new class of processes the heavier minerals are floated to the surface, while the lighter minerals sink to the bottom of the spitzkasten, whence they are drawn off.

This extraordinary result is effected by the action of the acid in the water upon certain minerals in the ore, leading to the evolution of gas, chiefly carbon dioxide, the bubbles of which selectively attach themselves to certain minerals of the ore, giving them a buoyancy which causes them to float to the surface, where they accumulate as a scum, readily removable, and in this way enable a separation to be made.

The nature of the physical reactions which take place in these processes has not been understood until recently; indeed, it may be said that even at the present time they are by no means clearly and fully understood. Practically the only useful literature on the subject is a paper entitled "The Physics of Ore Flotation" by J. Swinburne and G. Rudorf, and a paper entitled "Flotation Processes" by A. K. Huntington, these papers having been read before the Faraday Society, at London, December 12, 1905, and republished, in abstract, in the *Engineering and Mining Journal* of February 10, and 17, 1906. Even these papers do not, however, make it entirely clear why the bubbles of gas attach themselves to a certain kind of mineral particles, zinc blende for example, and do not attach themselves to other kinds of mineral particles, quartz for example. For the present discussion it is sufficient, to recognize that such a separation does actually take place and has proved to be the basis of processes which are now commercially applied upon a large scale.

The flotation processes consist of feeding the finely crushed ore into a solution of dilute sulphuric acid, commonly 2 per cent. acid, as in the Potter process; or into a solution of acid and sulphate of soda, as in the Delprat process;* the temperature of the solution in either case being maintained at about 80° C. Under these conditions, certain of the metallic sulphides (especially blende and galena) are floated up by attached bubbles of gas and form a coherent scum, which can be removed, leaving behind the earthy and silicious matter. When carefully carried out, the separation is, practically a quantitative one in the case of certain ores.

* The Delprat process was originally known as the "salt cake" process, because of the use of sulphate of soda in the bath, but in its latest application this has been abandoned, and common salt (sodium chloride) is now used instead. The function of these salts is supposed to be to densify the bath, but this appears to be of doubtful, if any, advantage. In the Potter process the bath naturally becomes densified to some extent by the iron and other impurities which are dissolved from the ore. There has been litigation between the owners of these patents, which is not yet settled, but there is apparently no question that Potter was the real inventor of the flotation process, and his patent will probably be upheld, covering broadly the flotation process in which an acid bath is employed.

It has been shown beyond question that the gas which is principally concerned in the flotation is carbon dioxide. It was at first assumed by some that the carbon dioxide was derived from calcite, and by others that it came from carbonates produced on the surfaces of the sulphides by weathering. On going into the matter, Prof. Huntington was able to prove that the carbon dioxide which gives rise to the flotation is liberated mainly from the rhodochrosite and siderite of the ore, and certainly not from the calcite.

Besides the Potter and Delprat processes, the de Bavay process has recently attracted a great deal of attention, having been tried on a large experimental scale with highly promising results. In this process no acid is used, the ore simply being gasified with carbon dioxide, which may be derived from chimney gases, and then fed suitably into a bath of water, wherein the zinc blende floats on the surface, while the quartz sinks to the bottom. The phenomena of this process are even less understood than are those of the Potter and Delprat processes. However, the de Bavay process is much slower in its action than either of the others, and is more costly both in plant and in operation.

Considering these processes solely from the technical point of view, it appears that in the treatment of the zinky tailings of Broken Hill, they have clearly beaten the process of magnetic separation, producing a higher grade of concentrate, and at less cost per ton. With the Potter and the Delprat processes, the concentrate contains 42 to 45 per cent. zinc, and contains about 72 to 75 per cent. of the zinc in the original ore. During a five-weeks run by the Block 14 company in the autumn of 1904, 2,450 tons of tailings were treated, assaying 17.6 per cent. zinc, 6.3 per cent. lead and 7.6 oz. silver per ton. The first class concentrate amounted to 739 tons, assaying 42 per cent. zinc, 5.5 per cent. lead, and 11 oz. silver per ton. There was also produced a middling, amounting to 95 tons, and assaying 22.5 per cent. zinc, 13 per cent. lead, and 13.5 oz. silver per ton. These figures show a recovery in the first class concentrate of 72 per cent. of the zinc and 26 per cent. of the lead, the concentrate amounting in weight to 30 per cent. of the crude ore. The figures show, moreover, that the flotation of the blende in the acid bath is more active than the flotation of the galena. The latter is fairly easy to float if finely ground, but owing to its high specific gravity, finer pulverization is necessary to float it successfully. The chief separation that is made in the Broken Hill ores is between the blende and the gangue of the ore. As between the blende and the galena, the separation is less sharp, both being capable of flotation, but the galena to less extent than the blende, because of its higher specific gravity. However, it is a well-known fact that the concentrate produced by flotation at Broken Hill assays rather high in lead. Because of the different behaviour of lead in the non-acid bath, the de Bavay process appears to be able to make a concentrate of considerably higher tenor in zinc than either of the other processes, the de Bavay concentrate rising to 50 p.c. zinc. My own experiments, which will be referred to later, show that blende, galena and pyrite, under certain conditions may be floated nearly equally well.

The plant required for either the Potter or the Delprat process is of simple design, the essential apparatus resembling an ordinary spitzkasten It is built of 3 x 3 in. timber, lined with 1 in. boards (dressed smooth) and 6 lb. sheet lead. The ore, slightly moist, is delivered by a shaking feeder at one side of the spitzkasten and passes down into the bath. The heavy sulphides, which under ordinary conditions would sink to the bottom, in the flotation process, because of the buoyancy of the bubbles of gas which become attached to them, rise to the top and flow off for collection, the spitzkasten being arranged for overflow both on the right and left hand sides. The supply of acidulated water is derived from a main passing over the line of spitzkasten, from which branches project down into the latter. The main leads from the storage tanks, into which the clear solution from the settling tanks is pumped for further use. In the storage tanks it is reheated to 80° C by blowing steam into it. The gangue escapes continuously from the bottom of the spitzkasten, the flow being regulated by an adjustable valve, and is received on a belt conveyor, which transports it to the dump.

The manipulation of the process is a delicate operation, because the adhesion of the bubbles of gas to the sulphide particles is not by any means a strong one, and in fact is maintained only so long as the bath is free from vibration; this necessitates an arrangement of the spitzkasten that will be absolutely steady; the slightest shock is sufficient to detach the gas and sink the sulphide. This fact is taken advantage of in the collection of the valuable minerals, the overflowing concentrate being dropped suddenly into a collecting tank. The gas bubble is thus disengaged, and the sulphide particles, being no longer buoyant, sink immediately and are caught for further treatment. The clear supernatant solution, drawn off from the concentrate, is returned to the storage tanks for use in the treatment of further quantities of crude ore. The output of each spitzkasten is, comparatively speaking, very large, as the size ordinarily used at Broken Hill is about 4 ft. 6 in. square and 5 ft. deep, and the capacity is approximately 6 tons per hour.

The flotation process is comparatively cheap. According to my private advices from Broken Hill, one man attends to six spitzkasten, his duty being merely to see that the scum is floating off regularly and that the tailings-discharge does not clog up. The rise of the scum is very rapid. It should accumulate in a dense mass about 1½ in. thick on the surface of the bath. Holes in the scum are evidence of irregular working. Mechanical skimming, or indeed any kind of skimming, is disadvantageous, the best result being obtained when the scum quietly floats off. The consumption of sulphuric acid in the Potter process is 30 to 35 lbs. (computed as 100% H_2SO_4) per 2240 lbs. of ore. The acid solution loses about 30°F. in a circuit. The pumping and reheating of this solution, together with acid and labour, constitute the chief items of expense in the process. The total cost at Broken Hill, including loading the ore at the tailings pile (tailings being the material treated) and transporting it to the flotation plant is only about 50c. per 2240 lb.

As in the cases of magnetic separation and electrostatic separation, in flotation the treatment of very fine ore is also a difficult problem. Neither the Potter nor the Delprat process will treat slime; similar difficulty has probably been experienced in the de Bavay process, though it has not been definitely reported. This is because of the entanglement of the particles in rising in a muddy bath, the finest gangue failing to settle rapidly enough, the result being a dirty, low grade scum. On the other hand, it has been found at Broken Hill that these processes will not give good results on particles larger than 0.5 millimeter, or such as will pass a 28-mesh screen. In these respects, the de Bavay process appears to be similar to the Potter and the Delprat. De Bavay recommends material of 40 to 80 mesh size. Experiments and practice have, moreover, shown that in order to secure the best results there must be a certain ratio between the floatable and the non-floatable material. In some cases, blende which alone would not float, became buoyant after a certain proportion of quartz had been mixed with it; in other cases, an ore which alone gave a very good flotation, upon the mixture with it of a large quantity of sand gave a very bad flotation.

The possible application of the flotation processes to the treatment of the ores of British Columbia was not thoroughly investigated by the Commission, because of several reasons, these being principally the opinion that flotation would offer less promise than magnetic separation, together with the desire to issue promptly the report, and the desire to perform the work inside of the limit of the appropriation. However, a series of experiments was performed at New York, under my personal direction, which throw much light upon the possible applicability of the flotation processes to the treatment of certain of the ores of British Columbia. It was previously a matter of doubt as to whether the flotation processes were limited in their application to the ores of a particular district, such as those of Broken Hill, which might possess certain physical properties making them especially adaptable to separation in this way, whereas apparently similar ores from other districts could not be so separated. The common presence of siderite in the zinc ores of the Slocan district of British Columbia indicated, however, that these ores might also be capable of separation by the flotation process.

This, indeed, proved to be the case. In an experiment on the ore from the Ruth mine, it was found possible to produce in the first trial, a concentrate assaying as high as 50 per cent. zinc, and containing 87 per cent. of the zinc in the original ore. The results of other experiments are given in the following paragraphs:—

In making these experiments, four ores were selected as representatives of widely different mineralogical combinations, viz.: ores from the Ruth, Blue Bell, St. Eugene and Big Ledge mines. Without going into the details of all the experiments, the following results may be reported as representative:

1. RUTH.—Sample assayed 35.95% zinc. Ore crushed to pass 40-mesh screen was floated in bath of water acidulated with 2% H_2SO_4, and

also in bath of water acidulated with $H_2 SO_4$ and densified with zinc sulphate to 1.35 sp. grav., the baths in each case being maintained at 65° C. The results were as follows:

A.—Undensified bath.

	Grams	% Zinc	Grams, Zinc	% Extraction
Raw Ore............	40.000	35.95	14.38	......
Concentrate	25.452	48.75	12.41	86.3
Tailing...............	4.710	20.05	.94	6.5

B.—Densified bath.

	Grams	% Zinc	Grams, Zinc	% Extraction
Raw Ore.............	40.000	35.95	14.38	......
Concentrate..........	13.530	46.30	6.27	43.6
Tailing...............	19.850	38.70	7.68	53.4

These tests showed a large shrinkage in weight of the products as compared with the original ore, attributable partly to the evolution of carbon dioxide gas and the passing of iron into solution as sulphate, which of course implies consumption of sulphuric acid, and partly to loss in manipulation. In this respect laboratory experiments are much inferior to large scale work, wherein the floatation is very rapid, whereas the decantation from beakers was slow and tedious and the ore was too long exposed to the action of acid. Incidentally it may be remarked that this process is one whereof small experiments should be regularly beaten in large scale work. Test B shows so large a falling off in percentage of extraction as compared with test A that it is open to suspicion of not having been properly performed. However, it is in line with previous experiments of this series in indicating that there is no advantage to be gained from a solution densified to 1.25—1.35 sp. grav., an opinion which is, I believe, confirmed by the experience at Broken Hill.

C.—Undensified bath. Ore, 40-mesh.

	Grams	% Zinc	Grams, Zinc	% Extraction
Raw Ore.............	50.00	38.55	19.275	......
Concentrate	27.15	50.40	13.683	71.0
Tailing...............	16.87	26.55	4.479	23.2

D.—Undensified bath. Ore, 60-mesh.

	Grams	% Zinc	Grams, Zinc	% Extraction
Raw Ore.............	50.00	38.55	19.275	......
Concentrate	33.95	42.48	14.422	74.8
Tailing...............	9.44	19.32	1.824	9.5

Experiments C and D and all subsequent ones were performed at 80°C.

The ore used in experiment C assayed 38.55% zinc, 11.08% iron, 1.87% lead, and 7.62 ozs. silver per ton. The concentrate assayed 50.40 % zinc, 8.4% iron, 1.68% lead and 11.66 ozs. silver per ton. Thus, while the concentrate accounts for 71% of the zinc, it contained only 50% of the lead, but on the other hand contained 83% of the silver. This is not a surprising result, the silver of the ore being in the form of freibergite, which, being very brittle, naturally goes into the finest slime and floats easily; in fact, as every mill-man knows, it floats as a greasy scum in non-acidulated water, which is a troublesome source of loss of mineral in ordinary dressing.

2. BLUE BELL.—Sample assayed 14.74% zinc. All experiments were made with bath of water containing 2% of H_2SO_4, heated to 80°C.

E.—Ore crushed to 40-mesh size.

	Grams	% Zinc	Grams, Zinc	% Extraction
Raw Ore	50.00	14.74	7.370	......
Concentrate	36.65	19.17	7.026	95.4
Tailing	10.49	1.99	0.209	2.8

F.—Ore crushed to 60-mesh size.

	Grams	% Zinc	Grams, Zinc	% Extraction
Raw Ore	50.00	14.74	7.370	......
Concentrate	35.52	19.47	6.916	93.8
Tailing	10.82	2.81	0.304	4.1

The raw ore assayed 14.74% zinc, 14.67% iron, 15.30% lead and 2.37 ozs. silver per ton. The concentrate produced in experiment E assayed 19.17% zinc, 15.45% iron, 21.40% lead and 4.46 ozs. silver per ton. This shows an extraction of 100% of both silver and lead.

3. ST. EUGENE.—This ore behaved badly in the experiments. One which was carried to completion, using ore of 40-mesh size, 2% H_2SO_4 and 80° C, resulted in a concentrate weighing 34.24% of the ore used, and assaying 33.02% zinc Another, using ore of 60-mesh size, 2% H_2SO_4, and 80°C, resulted in a concentrate weighing 38.82% of the ore used, and assaying 34.87% zinc. In the case of this ore both the degree of enrichment and the percentage of extraction are too low to indicate any possible advantage to be derived from the flotation process.

4. BIG LEDGE.—This ore also gave unfavourable results. Only a small percentage of the ore could be floated, and the concentrate showed only trifling enrichment.

REMARKS.—The ores, which contained siderite, viz.: the Ruth and Blue Bell, gave good flotation; the ores which lacked siderite, viz.: the St. Eugene and Big Ledge, behaved very unsatisfactorily, as was to be expected. In the cases of the ores which could be floated well, there was clearly no advantage in crushing it to finer size than would pass a 40-mesh screen. Indeed, on the whole the ore of 60-mesh size gave results inferior to those of the 40-mesh ore. The first experiments were made at a temperature of 65° C. Later ones were made at 80° C. Increase in temperature improved the flotation. As previously mentioned, densification of the bath up to 1.35 specific gravity was of no advantage.

Especially in the case of the Blue Bell ore, the separation of metallic minerals from the non-metallic was excellent, the tailing being a pure, white granular quartz, with only a few specks of blende and pyrites intermixed. The concentrate also was very clean, showing scarcely any particles of gangue. The high degree of perfection attained in the separation of this ore suggests a promising line of treatment, namely flotation of the metallic minerals, which then should be subjected to washing on Wilfley tables to separate the galena, both the concentrate and the tailings from these tables being caught in filters to avoid slime losses. The tailing from the Wilfleys, consisting of blende and pyrite, might then be roasted and submitted to magnetic separation for enrichment in zinc. Experiments on this line would necessarily be carried out on a somewhat large laboratory scale, which the means at the disposal of the Commission did not permit.

Flotation experiments may be performed in ordinary beakers, but it is found that the depth of the bath is an important factor in the process, and the experiments are best performed in a glass tube, standing erect, about 3 in. in diameter inside and 3 ft. in depth. This tube should be supplied with acidulated water from a vessel at a higher level, a glass tube passing down from the latter into the separating tube, and extending to about 10 in. above the bottom of the latter. The separating tube should stand in a basin to catch the overflow. The acid water is heated to about 80° C in the supply vessel. The separating tube being about half filled with the acid water, the latter is allowed to run in slowly, and at the same time the ore to be separated is fed, slightly moist, into the separating tube. The scum immediately rises to the surface, and when the level in the separating tube has become flush with the top of the latter, the scum begins to overflow, running down the outside of the separating tube and being caught in the basin in which the latter stands. This method closely approximates the conditions of practice.

RAILWAY TRANSPORTATION.

The zinc ore producing districts of British Columbia are entered by two railways, namely the Canadian Pacific and the Great Northern. The Canadian Pacific is purely a Canadian line. The Great Northern is essentially an American line, only its branches entering into Canada. It has been reliably reported, however, that the Great Northern plans to construct a new trans-continental line through western Canada, which will make it a Canadian line to a much greater extent than at present, and will also go to increase its usefulness toward furnishing transportation for the mining interests of the East and West Kootenays, through which the new line will pass.

The zinc ore producing districts are fairly well served with transportation facilities by the existing lines. The lines of the Canadian Pacific and the Great Northern railways, combined with the splendid system of water transportation provided by nature in the Arrow, Slocan and Kootenay lakes, afford good communication through all the districts. The most serious feature in the transportation problem is the cost of carriage from mines to railway or lake, because of the location of many of the mines on the steep mountain sides, in many cases remote. This is an expensive feature that is a bar to the working of many properties, except those that are well situated with respect to the railway lines, or are so situated that they could install aerial or gravity tramways. The waggon roads of the district are generally good. There is, consequently, little or no remedy that can be offered in this particular.

The Canadian Pacific taps the larger part of the Slocan district. The rates granted are low, considering the comparatively small tonnage moving, and the distance to which it has to be carried. The Great Northern railway, (Kaslo & Slocan) taps only a small portion of the district, but this portion is promising in future possibilities.

The freight rate on zinc ore from points in the Slocan to Frank, Alberta, is $2.25 per ton; to Pueblo, Colo., it is $9 per ton; to smelting points in Kansas, it is $10 per ton; to Antwerp, and other European ports, it is $13 per ton; the rate of $13 is for shipments via Montreal, no rate for shipments via Pacific coast ports having been published.

Considering the distances from the Slocan to Colorado, Kansas, and Europe, the above freight rates are certainly very reasonable; if reduced to a ton-mile basis, they would doubtless figure lower than the average commodity freight rates in the United States.

ELECTROTHERMIC ZINC SMELTING.

Attention must be directed to electrothermic smelting of zinc ore, which in view of the progress already made in such application of electricity in other arts, naturally received the consideration of zinc metallurgists. Zinc ore has indeed been smelted electrothermically for several years in Scandinavia by the process of C. P. G. De Laval, and at the present time there are plants in operation at Trollhattan (Sweden) where 3,000 h.p. are employed in the reduction of crude ore and zinc ashes (galvanizers' waste); 4,000 h.p. at Sarpsborg (Norway) in the reduction of zinc ashes; and 1,800 h.p. at Hallstahammar in the smelting of ore. This represents already a considerable development of the industry.

Experiments in electrothermic smelting have been made by several other inventors, but none has made so much progress as De Laval. Those who are interested in the subject may look up the patents of Cowles, Casaretti & Bertani, Ferraris, Salguès, Dorsemagen, Brown & Oesterle, Contardo and Snyder. The Canada Metal Co. is now conducting experiments at Vancouver with the process of F. T. Snyder. I hoped to witness one of these experiments, but at the time of my visit at Vancouver the furnace had been torn down for rebuilding.

The successful smelting of zinc ore in the electric furnace is a matter which depends chiefly, indeed I may say solely, upon the cost of the process. De Laval has shown that it can be done. The doing of it profitably is quite another question. This is dependent chiefly upon the cost of the power, the electrical efficiency of the furnace, and finally upon the purely metallurgical difficulties that pertain to the reduction of zinc oxide and the condensation of the metallic vapour.

With respect to power consumption there are comparatively few data available, and such as there are available are of doubtful value. Estimates of the probable results may be formulated in two ways, viz. upon the reported results of experimental work and by analogy with the ordinary method of smelting.

In the smelting of one kilogram of ordinary roasted ore, containing 50 per cent. of zinc, there is theoretically required about 1,000 calories of heat, this covering the reduction of the zinc oxide and other metallic oxides, the accessory reactions, the heat carried away in the residues, etc. In the best practice the reduction of one kilogram of such ore is effected by one kilogram of coal of 7,250 calories, but 1.1 kg., corresponding to about 8,000 calories may be assumed as more representative. This shows an actual utilization of 12.5% of the calorific power of the coal by its direct combustion.

Now if the ore is to be smelted in an electric furnace, the heat requirement will at least be the same as in the previous case, viz. 1,000 calories per kilogram of ore. If the efficiency of the furnace be assumed at 50% which is probably as high as should be reckoned at the present time, 2,000, calories, or their equivalent in watts, must be delivered to the electrodes. Estimating the efficiency of the dynamo at 90%, then 2,222 calories, or their equivalent in horse-power, must be applied to the driving pulley.

Without carrying this computation to the coal, step by step, it may be assumed that the highest type of compound condensing steam engine develops in mechanical energy about 15% of the calorific power of the coal. Consequently the reduction of one kilogram of ore would require $2{,}222 \div 0.15 = 14{,}814$ calories, and the utilization of energy would be only 6.75%. Obviously, electrothermic smelting could not compete with direct smelting under the above conditions. Leaving aside the questions of cost of plant, consumption of electrodes, labour and other supplies, electric smelting would be only on equal terms if the efficiency of the furnace were increased to nearly 100%. The use of the gas engine instead of the steam engine, affording a conversion of 30% of the calories of the coal into mechanical power, together with an electric furnace of nearly 100% efficiency, might show a little advantage but at present the gas engine, in spite of its much higher efficiency, is only a little, if any, ahead of the steam engine in commercial economy, after the amortization and interest charges on the much more costly plant involved by it have been written off.

Making comparison with water-power, it may be reckoned that with coal of 7,250 calories heat value, available at $1 per ton, a horse-power can be developed for $25 per annum. A smelter who enjoyed those conditions would, consequently, have to obtain water-power at better than $12 per horse-power-year to make it worth his while to remove to an equally desirable commercial location to undertake electric smelting, other elements of cost being considered equal. These estimates are necessarily vague, hypothetical and inexact, because of the lack of precise data, but they will at least serve to give an idea of the problem.

Looking now from the other view point, the data are very scanty, because the results reported of experiments are so few and so inconclusive. At Crampagna (Ariège), France, Salguès experimented on the smelting of ore containing 40 to 45% zinc in a 100-kw. furnace (*Bull. Soc. Ing. Civ.*, 1903, p. 174). He claims to have obtained a production of 5 kg. of metal per kilowatt per day. This would correspond roughly to 76.5 kw. per 1,000 kg. of ore, or estimating the efficiency of the dynamo at 90%, there would be required 114 brake horse-power of the engine. Estimating an evaporation of 10 lb. water per pound of coal, and the development of a horse power from 2 lb. of steam, there would be required 2,500 kg. of coal per 1,000 kg. of ore, which is 2.5 times as much as is consumed in the present best practice of smelting in the ordinary way.

It is to be remarked that there is considerable doubt as to the experiments of Salguès. At all events, he does not appear to have been able to make any commercial production of spelter, and has been baffled, it is said, by a commonly experienced metallurgical difficulty, which will be referred to further on.

Casaretti and Bertani are said (l'*Elettricita*, Milan, XIV., page 593) to have produced at Bergamo, Italy, 9 kg. of zinc per kilowatt per day, which is certainly much better than the figure reported by Salguès, and on the same assumptions as in connection with the latter would correspond to about 1.4 ton of coal per ton of ore. It is doubtful, however, if this result could be maintained regularly, and Casaretti and Bertani also seem to have experienced the same metallurgical difficulty that other experimenters have found. The reported yield of metal is so close to the theoretical that it is highly doubtful that it could be regularly maintained.

In default of any other data, I will assume that it may be possible to produce 7.5 kg. per kilowatt-day from ore containing 45 per cent. zinc. There would then be required about 70 electrical horse-power per 2,000 lb. of ore per day. The generation of this power from coal would be out of the question, because it would be much cheaper to burn the coal directly. Even assuming that the consumption of coal were 1.5 ton per ton of ore, at $2 per ton the fuel cost would be $3. In order to present equal terms, water power would have to be offered at $(360 \times 3) \div 70 = \$15.40$ per horse-power per year.

The heat equivalent of one horse-power-day is 61,080 B.t.u. = approx. 15,400 calories. Estimating that 1,000 calories are required to smelt one kilogram of ore, one horse-power in 24 hours would smelt 15.4 kg., or one kilowatt would smelt 20.6 kg. In the case of an ore assaying 50% zinc, at 90% extraction the yield would be 9.27 kg. of spelter per kilowatt per day. This indicates that the estimates in the previous paragraphs are not irrational.

However, in respect to the other details of cost, lack of experience leaves us quite in the dark. The electric furnace would save the expense of retorts, but on the other hand it would be subject to the expense of electrodes. It would be chiefly guess-work to go into the subject of labour, but it may be pointed out that the yard-labour, roasting-labour, and some other branches would be the same as in the ordinary process of smelting. The only possible chance for economy would be in the distillation furnace labour, but in the present state of the art I should hesitate to indicate either that there would be any economy of labour, or the opposite. The cost of plant would doubtless be higher than for the most expensive direct-smelting plant, because if 70 h.p. be required for the smelting of 350 tons of ore per annum, the cost of the electric plant alone would probably be in the neighbourhood of $10 per ton, which is much more than the cost of the ordinary furnaces.

Now to take up the metallurgical difficulty in the electrothermic smelting of zinc ore, to which I have previously referred, it has proved so far, both in experiment and in practice, impossible to condense in liquid form the zinc vapour produced by the electric furnace; the product is a blue powder, which must be resmelted. At first glance, there does not appear any reason why liquid zinc should not be obtained from the electric furnace, as well as from the retort, but it is the fact that it cannot be yet obtained, and the precise reason for the failure is still obscure. Various explanations of the difficulty have been offered, and upon their hypotheses remedies have been proposed, but so far as I am aware, none has yet been proved successful in practice.

In the application of the De Laval process in Scandinavia both ore and zinc ashes, the latter, being a waste product of galvanizers, are smelted. The works at Sarpsborg, use zinc ashes exclusively. In the smelting of ore at Trollhattan, the product of the first smelting is blue powder. This is resmelted, either alone or mixed with a further portion of ore, yielding a spelter rich in lead, the lead content of the ore being volatilized to a large extent by the high temperature prevailing in the furnace. The leady spelter is refined by remelting and settling out the lead in a reverberatory furnace, in the ordinary manner, or by a third distillation electrically. By the former process the refined spelter is of about the grade of ordinary Silesian; by the latter process, a high-grade spelter is produced.

From the present development of electrothermic smelting of zinc ore, and the theoretical calculations respecting it, the following conclusions may be drawn:

1. Electric smelting will never displace ordinary smelting, if it be necessary to generate the power from coal.

2. Electric smelting may be, in the future, economically conducted at places where very cheap hydro-electric power is available.

3. Aside from the question of power, up to the present time, certain peculiar and serious metallurgical difficulties in electric smelting have not been satisfactorily overcome.

Although the electric smelting works in Scandinavia have been running for several years, it must be recognized that the conditions under which they are operated, are exceptional, and regarded broadly the electric smelting of zinc ore has by no means passed beyond the experimental stage.

It is unlikely that the electric smelting of zinc ore can ever be profitably carried on in the zinc producing districts of the East and West Kootenays, B.C., because in many parts of those districts the water powers are small and irregular, being insufficient even to run the comparatively small concentrating mills during the winter season. Where there are large water powers, as for example, at Bonnington Falls, the power generated is so valuable for mining and miscellaneous purposes, that it could not be furnished at the low figure which would be required by an electric smelting plant.

On the coast of British Columbia, I am informed, there are many large water powers, which can be developed at so comparatively small a cost as to permit them to furnish power at a low figure. It is possible that at some future date, these may be utilized for the electric smelting of zinc and other ores. Possibly, also, water powers in eastern Canada may be similarly utilized. Any such development is, however, something of the future, and is not to be reckoned upon at the present time. Electric smelting of zinc ore must undoubtedly go through many stages of experiment before it can be pronounced a metallurgical and commercial success.

STATISTICS OF PRODUCTION, CONSUMPTION AND PRICE.

In the following tables are summarized all the available statistics as to the consumption of zinc in the Dominion of Canada. There having been no production of the metal in the Dominion, the statistics of imports may be regarded as representing the consumption. These statistics have been furnished by the Department of Customs.

IMPORTS OF ZINC IN BLOCKS, PIGS AND SHEETS.

Fiscal Year	Cwt.	Value	Fiscal Year	Cwt.	Value
1880	13,805	$ 67,881	1893	26,446	$124,360
1881	20,920	94,015	1894	20,774	90,680
1882	15,021	76,631	1895	15,061	63,373
1883	22,765	94,799	1896	20,223	80,784
1884	18,945	77,373	1897	11,946	57,754
1885	20,954	70,598	1898	35,148	112,785
1886	23,146	85,599	1899	18,785	107,477

Fiscal Year	Cwt.	Value	Fiscal Year	Cwt.	Value
1887	26,142	98,557	1900	28,748	156,167
1888	16,407	65,827	1901	20,527	103,457
1889	19,782	83,935	1902	34,871	141,560
1890	18,236	92,530	1903	26,646	142,827
1891	17,984	105,023	1904	25,553	138,057
1892	21,881	127,302	1905	25,141	141,514

IMPORTS OF SPELTER.

Fiscal Year	Cwt.	Value	Fiscal Year	Cwt.	Value
1880	1,073	$ 5,310	1893	10,721	49,822
1881	2,904	12,276	1894	8,423	35,615
1882	1,654	7,779	1895	9,249	30,245
1883	1,274	5,196	1896	10,897	40,548
1884	2,239	10,417	1897	8,342	32,826
1885	3,325	10,875	1898	2,794	13,561
1886	5,432	18,238	1899	5,450	29,687
1887	6,908	25,007	1900	5,836	29,416
1888	7,772	29,762	1901	14,621	58,283
1889	8,750	37,403	1902	18,356	80,757
1890	14,570	71,122	1903	23,159	110,817
1891	6,249	31,459	1904	33,952	164,751
1892	13,909	62,550	1905	37,941	206,224

IMPORTS OF ZINC, MANUFACTURERS OF.

Fiscal Year	Cwt.	Value	Fiscal Year	Cwt.	Value
1880	....	$ 8,327	1893	....	$ 7,464
1881	....	20,178	1894	....	6,193
1882	....	15,526	1895	....	5,581
1883	....	22,599	1896	....	6,290
1884	....	11,952	1897	....	5,145
1885	....	9,459	1898	....	10,503
1886	....	7,345	1899	....	14,661
1887	....	6,561	1900	....	11,475
1888	....	7,402	1901	....	6,882
1889	....	7,233	1902	....	6,683
1890	....	6,472	1903	....	9,792
1891	....	7,178	1904	....	14,065
1892	....	7,563	1905	....	11,912

QUANTITY AND VALUE OF ZINC IN BLOCKS, PIGS AND SHEETS AND OF SPELTER AND MANUFACTURES OF ZINC IMPORTED INTO CANADA FOR THE FISCAL YEARS ENDING JUNE 30TH, 1903, 1904, 1905, AND 5 MONTHS TO NOVEMBER 30, 1905.

Articles.	1903		1904	
	Cwt.	Value	Cwt.	Value
Zinc in blocks, pigs and sheets.	26,646	$ 142,827	25,553	$ 138,057
Spelter, zinc, in blocks and pigs	23,159	110,817	33,952	164,751
Zinc, manufactures of. Dutiable	....	9,792	....	14,065
Total	49,805	263,436	59,505	316,873

Articles.	1905		5 months to November 30th, 1905	
	Cwt.	Value	Cwt.	Value
Zinc in blocks, pigs and sheets.	25,141	$ 141,514	9,555	$ 61,074
Spelter, zinc, in blocks and pigs	37,941	206,244	20,417	112,709
Zinc, manufactures of. Dutiable	....	11,912	....	4,713
Total......................	63,082	359,670	29,972	178,496

In explanation of the above statistics, the following letter was received from the Commissioner of Customs:

DEPARTMENT OF CUSTOMS,
Ottawa, March 12, 1906.

EUGENE HAANEL, ESQ., Ph. D.,
Superintendent of Mines,
Ottawa, Ont.

SIR,—

I beg to acknowledge the receipt of your letter of the 8th instant enclosing a communication from Mr. W. R. Ingalls, in which he enquires as to what distinction is made at the Customs Department between (1st) zinc in blocks, pigs, and sheets, and (2) spelter zinc in blocks and pigs, and also as to the tariff on each kind of spelter and zinc.

In reply I beg to state that previous to 1897 the Canadian Customs Tariff provided for these articles as separate items and in 1897 the tariff was changed to embrace these articles in one tariff item, with the distinction that spelter and zinc were to be imported in blocks and pigs only, while zinc should be imported in blocks, pigs and sheets, so that to facilitate comparison with previous years the Statistical Branch of this Department continued the classification in its original form, as mentioned in the tariff of 1904, and at the present time the compilation of imports of these articles is made altogether from the information afforded by the entries taken at the various ports, sometimes being designated as "spelter," other times as "zinc."

Both these articles are free of duty and are imported in net hundred weight of one hundred pounds each.

I have the honour to be,
Sir,
Your obedient servant,

(Signed) JOHN McDOUGALD,
Commissioner of Customs.

PRODUCTION OF ZINC AND ZINC ORE.

The statistics of production of zinc ore in British Columbia have been given on an earlier page of this report. Besides in that Province, there has been a small production of zinc ore in Ontario, but the total has been relatively insignificant. For purposes of reference the following table of the production of zinc ore in the principal countries of the world is given. These figures are from official sources, being taken in part from the original authorities, in part from *Production and Properties of Zinc* (Ingalls) and in part from *The Mineral Industry*.

PRODUCTION OF ZINC ORE IN EUROPE AND AUSTRALIA.
(In metric tons.)

Year	Algeria	Austria	Belgium	France	Germany	Great Britain	Greece	Italy	N.S. Wales	Russia	Spain	Swed'n
1840	...	...	20,482	...	...	...	...	...	...	...	...	...
1841	...	...	18,380	...	...	...	...	...	...	...	...	...
1842	...	...	18,466	...	...	...	...	...	...	...	...	...
1843	...	...	25,668	...	...	...	...	...	...	...	...	...
1844	...	...	22,689	...	...	...	...	...	...	...	...	...
1845	...	...	30,027	...	...	...	...	...	...	...	...	...
1846	...	...	42,365	...	...	...	...	...	...	...	...	...
1847	...	...	47,663	...	...	...	...	...	...	...	...	...
1848	...	...	48,582	...	127,396	...	...	...	...	...	...	...
1849	...	...	49,712	...	128,743	...	...	...	...	...	...	...
1850	...	...	69,501	...	147,840	...	...	...	...	...	...	...
1851	...	...	80,266	...	150,950	...	...	...	...	...	...	...
1852	...	...	78,345	...	181,048	...	...	...	...	...	...	...
1853	...	...	80,092	...	162,383	...	...	...	...	...	...	...
1854	...	...	79,472	...	178,929	...	...	...	...	...	...	...
1855	...	5,465	81,273	...	214,365	...	...	...	...	...	...	...
1856	...	5,236	83,274	...	226,625	...	...	...	...	...	18,293	...
1857	...	5,916	76,236	...	218,890	...	...	...	...	...	...	...
1858	...	5,304	75,392	...	244,363	...	...	...	...	...	...	...
1859	...	5,382	70,390	...	278,277	...	...	...	...	...	...	...
1860	...	...	66,141	...	303,596	15,807	...	162	...	...	...	...
1861	...	...	73,155	...	328,682	16,029	...	168	...	...	24,744	7,209
1862	...	...	74,008	...	333,598	7,620	...	157	...	...	41,104	8,518
1863	...	...	61,767	...	291,693	13,153	...	265	...	...	48,124	9,033
1864	...	...	58,066	...	313,299	15,310	...	202	...	...	80,222	14,248
1865	...	...	56,185	...	335,348	18,135	...	732	...	...	70,158	25,949
1866	...	...	54,516	...	353,149	12,979	...	4,492	...	...	73,423	20,841
1867	...	...	58,046	...	368,929	13,710	...	6,443	...	...	86,822	18,790
1868	...	...	68,699	...	369,874	12,990	...	51,012	...	...	131,407	23,831
1869	...	16,564	66,918	...	405,025	15,788	...	80,524	...	...	113,485	31,741
1870	...	13,694	57,099	...	366,780	13,809	...	92,833	...	...	113,583	28,146
1871	...	...	61,129	...	335,173	18,027	...	56,426	...	...	107,380	32,172
1872	...	...	55,537	...	419,543	18,847	...	80,861	...	...	89,371	33,236
1873	...	14,642	42,582	1,279	444,950	16,231	...	79,036	...	...	101,010	27,444
1874	...	21,147	43,299	3,848	451,222	17,106	...	64,716	...	...	106,477	28,198
1875	...	25,728	42,504	4,088	467,953	24,371	...	61,968	...	...	100,174	31,642
1876	...	26,458	37,713	...	533,559	24,000	...	66,034	...	...	107,063	35,523
1877	...	24,002	44,987	...	577,312	24,806	...	88,844	...	...	70,951	39,966
1878	...	33,387	45,293	...	597,193	25,855	...	62,703	...	...	71,558	40,795
1879	...	19,389	42,689	...	589,546	22,564	...	73,411	...	...	60,980	43,817
1880	...	21,564	38,805	...	632,895	28,006	...	85,289	...	...	50,521	43,460
1881	...	27,340	23,553	...	659,530	36,109	...	72,176	...	...	42,911	43,811
1882	...	25,300	20,443	8,372	694,711	33,069	...	91,366	...	97,000	57,353	46,255
1883	...	28,749	20,738	7,156	677,794	30,215	43,731	100,574	...	...	54,193	45,347
1884	...	29,454	27,606	3,120	632,040	25,982	...	104,974	...	...	49,838	44,893
1885	...	23,598	18,185	5,078	680,654	25,072	...	107,887	...	...	49,509	48,589
1886	...	21,320	19,042	11,103	705,177	23,535	...	107,548	...	38,182	39,810	49,571
1887	...	21,099	20,879	13,321	900,712	25,862	42,258	93,143	...	...	39,012	46,241
1888	8,521	26,312	24,537	20,702	677,761	26,841	43,405	87,310	...	...	74,353	49,972
1889	12,556	30,096	21,184	34,290	708,829	23,582	33,025	97,059	...	...	71,774	59,381
1890	13,091	32,632	15,410	47,540	759,437	22,402	33,054	110,926	...	44,125	81,398	61,843
1891	13,636	28,828	14,280	56,300	793,544	22,580	28,344	120,685	...	47,390	...	61,591
1892	21,907	33,944	12,260	69,236	800,237	24,264	27,695	129,731	...	...	74,265	54,981
1893	24,400	30,531	11,310	74,400	787,911	24,134	22,589	132,767	...	...	62,616	46,623
1894	29,703	28,491	11,585	80,065	728,616	22,170	20,830	132,777	...	62,420	58,964	47,029
1895	14,300	25,862	12,230	72,989	706,423	17,758	24,031	121,197	...	57,213	54,109	31,349
1896	17,587	26,887	11,630	81,346	729 942	19,629	22,700	118,171	...	59,680	64,828	44,041
1997	32,269	27,463	10,954	83,044	663,850	19,587	30,906	122,214	29,303	54,524	73,848	66,636
1898	29,800	27,395	11,475	85,550	641,706	23,929	32,045	132,099	39,561	...	99,836	61,627
1899	42,970	36,100	13,190	84,813	664,536	23,505	22,907	150,629	50,680	...	119,710	65,159
1900	30,281	38,243	8,715	67,059	639,215	25,071	18,751	139,679	20,593	...	86,158	61,044
1901	26,913	36,072	6,645	61,539	647,496	24,133	20,926	135,784	642	...	119,708	48,630
1902	33,139	31,927	3,852	57,982	702,504	25,461	18,020	131,965	1,281	...	127,618	48,783
1903	43,313	29,544	3,630	66,922	682,853	25,286	13,884	157,521	21,086	...	154,126	62,927
1904	47,192	29,226	3,702	52,842	715,728	28,097	23,297	148,365	58,525	...	156,329	57,634
1905	...	...	...	...	731,281	24,025	18,134	...	105,189	...	...	...

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

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

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

WORLD'S PRODUCTION OF ZINC

(In metric tons.)

Year	Austria	Belgium	France	Germany	Great Britain	Netherlands	Russia	Spain	United States	Total
1845	388	7,221	...	17,100	...	...	3,569	...	...	...
1846	413	8,963	...	22,260	...	...	...	...	...	...
1847	352	10,241	...	22,400	...	...	...	...	...	...
1848	1,395	10,850	...	20,190	1,000	...	4,000	...	...	...
1849	...	13,579	...	26,270	...	...	...	...	...	...
1850	...	14,808	...	28,670	...	...	2,606	...	...	...
1851	...	15,250	...	30,620	...	...	...	...	...	...
1852	...	16,672	...	34,721	...	...	...	...	...	...
1853	...	18,817	...	34,673	...	...	...	...	...	...
1854	*1,500	19,553	...	36,873	*1,000	...	...	...	...	...
1855	...	20,633	...	38,254	...	...	1,107	...	...	...
1856	866	22,900	...	38,326	...	...	...	...	...	a 71,000
1857	1,055	24,526	...	43,611	...	...	...	...	...	a 79,000
1858	1,580	34,191	...	52,777	3,522	...	...	...	...	...
1859	1,246	28,631	...	49,282	3,757	...	...	...	...	...
1860	1,301	22,027	...	55,346	4,428	...	1,838	...	...	...
1861	...	28,150	...	58,573	4,487	...	...	...	...	...
1862	...	25,861	...	59,767	2,186	...	...	...	...	a 100,000
1863	...	28,978	...	60,315	3,897	...	...	...	...	...
1864	...	30,718	...	59,248	4,106	...	...	...	...	...
1865	...	34,244	...	56,490	4,533	...	3,089	...	...	...
1866	...	34,659	...	60,221	3,244	...	...	...	...	...
1867	...	38,684	...	63,874	3,811	...	...	...	...	a 110,000
1868	...	44,347	...	66,132	3,774	...	...	...	...	...
1869	1,866	47,407	...	69,851	4,573	...	...	...	...	...
1870	1,908	45,754	...	63,980	4,000	...	3,780	...	...	a 130,000
1871	...	45,623	...	58,297	5,047	...	...	3,166	...	...
1872	...	41,838	...	58,386	5,276	...	...	2,940	...	...
1873	2,285	42,314	12,627	62,755	4,544	...	3,378	2,993	6,664	137,560
1874	2,818	46,088	12,783	70,426	4,543	...	4,128	3,295	9,074	153,155
1875	2,940	49,960	13,739	74,337	6,823	...	3,988	3,831	13,914	169,532
1876	3,979	47,981	*14,000	83,227	6,750	...	4,626	4,349	14,520	179,432
1877	4,519	35,923	*15,000	94,996	6,384	...	4,635	3,780	15,281	200,518
1878	3,623	61,227	*16,000	94,953	6,412	...	3,646	3,775	17,242	206,878
1879	3,280	57,157	*17,000	96,756	5,645	...	4,321	3,800	19,057	207,016
1880	3,756	59,880	*18,000	99,646	7,279	...	4,390	4,221	21,080	218,252
1881	4,119	69,800	18,509	105,478	23,660	...	4,542	7,032	27,225	260,365
1882	4,791	72,947	18,525	113,418	26,501	...	4,462	7,310	30,642	278,596
1883	4,539	75,366	15,915	116,854	29,630	...	3,809	6,843	33,375	286,331
1884	4,536	77,487	16,884	125,276	30,238	...	4,313	4,295	35,585	298,614
1885	3,948	80,298	15,108	129,098	24,690	...	4,579	4,247	36,921	298,889
1886	3,843	79,246	16,132	130,854	21,572	...	4,190	4,327	36,696	298,860
1887	3,609	80,468	16,712	130,444	20,228	...	3,621	5,349	45,682	306,113
1888	4,001	80,675	16,960	133,224	27,214	...	3,869	5,117	50,731	321,791
1889	4,840	82,526	17,982	135,974	31,302	...	3,681	5,640	53,414	335,359
1890	5,449	82,701	19,372	139,266	29,614	...	3,768	5,919	61,111	347,200
1891	5,006	85,999	20,596	139,353	29,883	...	3,675	5,656	72,836	363,004
1892	5,237	91,546	20,680	139,938	30,798	...	4,369	5,925	76,279	374,772
1893	5,870	95,665	22,419	142,956	28,829	*2,743	4,522	5,752	69,159	377,915
1894	6,810	97,041	23,387	143,577	32,578	*3,565	5,014	5,100	67,135	384,207
1895	6,456	107,664	24,200	150,286	29,967	4,266	5,029	5,845	79,462	413,175
1896	6,888	113,361	35,585	153,082	25,278	4,770	6,257	6,133	74,280	425,711
1897	6,236	116,067	38,067	150,739	23,805	6,600	5,868	6,244	91,070	444,964
1898	7,302	119,067	37,155	154,867	28,387	6,700	5,664	6,031	103,514	468,937
1899	7,192	122,843	39,274	153,155	32,222	6,235	6,325	6,184	117,644	491,331
1900	6,742	119,317	36,325	155,790	30,309	6,845	5,970	5,785	111,794	465,438
1901	7,558	127,170	37,600	166,283	29,877	7,855	6,090	5,354	127,751	516,049
1902	8,309	124,780	36,282	174,927	40,244	9,910	8,280	5,560	143,552	552,338
1903	8,949	131,740	27,462	182,548	44,110	11,515	9,901	5,134	143,792	560,017
1904	9,159	137,330	43,196	193,058	46,218	12,895	10,607	5,887	164,921	623,453
1905.b	9,210	143,165	44,075	198,208	50,125	13,550	7,520	*5,500	183,014	654,367

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

The consumption of spelter in Europe did not attain much consequence previous to 1840. The statistics of production previous to 1845 are very incomplete. Germany produced 10,000 tons of spelter in 1838, 10,980 in 1840 and 18,000 in 1843. Austria produced 105 tons in 1840, and Russia made 2,739 tons in the same year. In 1846 the production of zinc in Europe amounted to 40,000 tons, rising to 50,000 tons in 1850 and 62,000 tons in 1853.

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

United States.—The statistics used are those of *The Mineral Industry* except for the years 1895 and 1896.

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

Germany.—Previous to 1862 the production of Germany is assumed as identical with the production of Prussia as reported in the statistics for that Kingdom. By far the larger part of the spelter production of Germany is due to the works of Upper Silesia. Practically the entire remainder of the German production is made in Rhenish Prussia and Westphalia.

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

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

Russia.—The production in 1889 and subsequent years is given as reported by Henry R. Merton & Co. The entire production of zinc in Russia is derived from the Kingdom of Poland, where there are only two producers, the reports for which by Messrs. Merton are always practically the same as the official figures.

CONSUMPTION OF ZINC.

The world's consumption of zinc during the last 10 years, as reported by the Metallgesellschaft of Frankfurt am Main, is given in the following table, which presumably groups Canada with "other countries". However, the actual consumption of zinc in Canada has been stated separately on a previous page of this volume.

THE WORLD'S CONSUMPTION OF ZINC.

(In metric tons.)

Country	1895	1896	1897	1898	1899	1900	1901	1902	1903	1904
Austria...........	25,140	25,686	24,143	23,516	20,916	23,782	23,212	23,506	22,523	25,287
Belgium..........	38,308	38,920	38,650	39,785	46,700	31,562	34,410	24,492	32,338	38,465
France...........	49,267	62,563	63,326	61,224	56,991	62,266	55,610	64,922	64,630	67,241
Germany.........	110,895	112,831	120,064	128,167	130,295	125,806	133,151	131,880	143,017	151,576
Great Britain.....	83,828	92,651	86,587	98,647	96,770	92,726	90,384	122,859	124,082	129,144
Italy.............	2,374	2,563	2,969	2,657	3,271	3,328	3,797	3,588	4,158	5,079
Netherlands.......	*3,600	*3,600	*3,600	*3,600	*3,600	*3,600	*3,700	*3,700	*3,700	*3,700
Russia...........	11,670	11,913	13,653	14,665	15,624	14,668	17,530	17,200	17,800	20,000
Spain............	4,505	2,698	4,158	1,575	3,627	3,704	3,530	4,060	2,906	4,010
United States.....	78,424	65,427	77,778	95,711	111,896	92,743	122,068	137,918	140,880	157,061
Other Countries...	*7,000	*7,600	*7,000	*6,500	*7,000	*8,000	*8,700	*8,000	*9,100	*9,100
Total...........	415,011	426,452	441,928	476,047	496,690	462,185	496,092	542,125	565,134	611,663
Production........	416,625	424,141	443,300	469,026	489,189	478,475	507,448	545,349	571,322	625,134
Ave. price, London.	£14⅝	£16⅝	£17½	£20½	£24⅞	£20¼	£17–	£18½	£21–	£22½
" " "	a$3,179	$3,615	$3,805	$4,457	$5,408	$4,402	$3,696	$4,022	$4,565	$4,892
" " New York	b 3,630	3,940	4,120	4,570	5,750	4,390	4,070	4,840	5,375	5,100

a. The average price at London in pounds sterling per 2,240 lb., as reported by the Metallgesellschaft has been converted into dollars. per 100 lb., at the exchange £1= $4.87.

b. Averages reported by *The Mineral Industry.*

CONSUMPTION OF ZINC IN THE UNITED STATES.

(In tons of 2,000 lb.)

Year	Production	Imports			Supply	Exports	Consumption
		Spelter	Sheet	Total			
1873.........	7,343	3,420	5,561	8,981	16,324	37	16,287
1874.........	10,000	1,797	3,008	4,805	14,805	22	14,783
1875.........	15,833	1,017	3,660	4,677	20,510	19	20,491
1876.........	16,000	474	2,306	2,780	18,780	67	18,713
1877.........	17,500	633	671	1,304	18,804	720	18,084
1878.........	19,000	635	628	1,263	20,263	1,273	18,890
1879.........	21,000	710	556	1,266	22,266	1,066	21,200
1880.........	23,239	4,046	2,035	6,081	29,320	684	28,636
1881.........	30,000	1,430	1,364	2,794	32,794	746	32,048
1882.........	33,765	9,204	2,207	11,411	45,176	745	44,431
1883.........	36,872	8,534	1,655	10,189	47,061	426	46,635
1884.........	38,544	2,935	476	3,411	41,955	63	41,892
1885.........	40,688	1,758	920	2,678	43,366	51	43,315
1886.........	42,641	2,150	546	2,696	45,337	459	44,878
1887.........	50,340	4,194	463	4,657	54,997	68	54,929
1888.........	55,903	1,913	148	2,061	57,964	31	57,933
1889.........	58,860	1,026	507	1,533	60,393	440	59,953
1890.........	67,342	1,000	391	1,391	68,733	1,648	67,085
1891.........	80,262	404	11	415	80,677	2,147	78,530
1892.........	84,082	149	14	163	84,245	6,247	77,998
1893.........	76,255	213	14	227	76,482	3,723	72,759
1894.........	74,004	194	20	214	74,218	1,804	72,414
1895.........	87,591	372	21	393	87,984	1,530	86,454
1896.........	81,878	520	14	534	82,412	10,130	72,282
1897.........	100,387	1,453	8	1,461	101,848	14,245	87,603
1898.........	114,104	1,371	....	1,371	115,475	10,500	104,975
1899.........	129,675	1,493	....	1,493	131,168	6,755	124,413
1900.........	123,231	1,007	....	1,007	124,238	22,410	101,828
1901.........	140,822	388	....	388	141,210	3,390	137,820
1902.........	158,237	619	....	619	158,856	3,237	155,619
1903.........	158,502	364	....	364	158,866	1,521	157,345
1904.........	181,803	467	....	467	182,270	10,073	172,197
1905.........	201,748	521	....	521	202,269	5,516	196,753

In the above table, consumption has been computed without taking into account the difference in stocks of metal on hand at the beginning and end of each year, statistics for which are unavailable over the whole period covered by the table.

CONSUMPTION OF ZINC WHITE IN THE UNITED STATES.
(In tons of 2,000 lb.)

Year	Production	Imports	Supply	Exports	Consumption	Tenor in Zinc at 80%	
						Short tons	Metric tons
1894	22,814	1,686	24,500	nil	24,500	19,600	17,777
1895	22,690	2,273	24,963	24	24,939	19,951	18,096
1896	15,863	2,286	18,149	2,324	15,825	12,660	11,483
1897	26,262	2,782	29,044	1,859	27,185	21,748	19,725
1898	32,747	1,671	34,418	3,925	30,493	24,394	22,125
1899	39,663	1,506	41,169	5,343	35,826	28,661	25,995
1900	47,151	1,309	48,460	5,656	42,804	34,243	31,060
1901	46,500	1,600	48,100	4,561	43,539	34,831	31,607
1902	52,730	1,636	54,366	5,358	49,008	39,206	35,577
1903	59,562	1,743	61,305	7,215	54,090	43,272	39,267
1904	59,613	1,293	60,906	8,157	52,749	42,199	38,293
1905	65,403	1,718	67,121	11,280	55,841	44,673	40,538

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

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

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

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

Year	1886	1887	1888	1889	1890	1891	1892	1893
Production	18,000	18,000	20,000	16,970	20,000	23,700	27,500	25,000
Imports	1,763	2,481	701	1,343	1,316	1,410	1,221	1,950
Supply	19,763	20,481	20,701	18,313	21,316	25,120	28,721	26,950

STATISTICS OF PRICE.

The chief markets of the world, wherein the prices which govern the zinc industry are established, are New York, St. Louis, London and Breslau. The price made in London practically governs the industry in Europe, although the business in Upper Silesia is transacted on the basis of the price at Breslau; the latter generally preserves, however, a certain relation to the London price. Similarly in the United States the business of the Kansas-Missouri district is transacted on the basis of the price of spelter at St. Louis. The St. Louis price, however, corresponds with the New York price, minus the difference in the cost of transportation to the two points. That difference is not merely the freight rate from St. Louis to New York, but is the variation between the rates from Kansas smelting points to New York and St. Louis respectively. The difference between the New York and St. Louis price varies consequently from time to time according to the freight rates, but during the last six years it has been quite steadily $0.15.

Although the importation of spelter into the United States from Europe is practically prohibited by the tariff of 1.5c. per lb., and the American price of spelter is consequently to a considerable extent independent of the European price, there is nevertheless at many times an intimate relation between the prices of the two Continents, because of the ability of American producers to export zinc at a profit under certain conditions. The freight rate on spelter from Kansas smelting points to Liverpool is ordinarily from \$0.25 to \$0.35 per 100 lb. (it has been as low as \$0.22 per 100 lb.) Consequently if at any time the price of spelter at London rises materially above the price at New York, the exportation of spelter from Kansas to Europe tends to reduce the European price to the American level, or vice versa to cause the American price to rise to the European level.

The average monthly price of spelter at New York and the average annual price at London, for a long period of years, are given in the subjoined tables. The New York prices are as given in *The Mineral Industry.*

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

	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year
1875..	6.56	6.46	6.35	6.75	7.20	7.20	7.30	7.175	7.175	7.275	7.275	7.275	7.00
1876..	7.50	7.625	7.685	7.80	7.875	7.625	7.185	7.125	6.96	6.685	6.495	6.435	7.25
1877..	6.375	6.56	6.435	6.31	6.125	5.995	5.745	5.85	5.81	5.80	5.745	5.625	6.03
1878..	5.625	5.435	5.435	5.125	4.81	4.435	4.625	4.685	4.81	4.66	4.625	4.31	4.88
1879..	4.375	4.51	4.495	4.50	4.375	4.245	4.56	5.21	5.81	6.185	6.06	6.125	5.036
1880..	6.185	6.56	6.625	6.31	5.81	5.31	4.935	5.06	4.935	4.935	4.775	4.70	5.51
1881..	5.06	5.185	4.935	4.935	5.935	4.875	4.875	5.06	5.215	5.31	5.685	5.935	5.243
1882..	5.875	5.685	5.495	5.375	5.435	5.31	5.245	5.31	5.245	5.245	4.995	4.685	5.325
1883..	4.56	4.56	4.685	4.675	4.625	4.495	4.40	4.35	4.45	4.40	4.385	4.36	4.495
1884..	4.285	4.325	4.50	4.575	4.525	4.455	4.50	4.57	4.56	4.475	4.35	4.125	4.443
1885..	4.31	4.275	4.21	4.21	4.175	4.05	4.25	4.50	4.56	4.56	4.525	4.525	4.345
1886..	4.40	4.455	4.55	4.55	4.50	4.375	4.35	4.35	4.325	4.275	4.275	4.425	4.40
1887..	4.55	4.55	4.475	4.45	4.55	4.55	4.575	4.55	4.50	4.525	4.775	5.40	4.625
1888..	5.425	5.35	5.10	4.85	4.65	4.55	4.55	4.75	4.975	5.05	4.90	4.875	4.91
1889..	5.00	4.95	4.75	4.675	4.75	4.975	5.10	5.20	5.175	5.10	5.20	5.40	5.023
1890..	5.41	5.28	5.187	5.085	5.35	5.575	5.55	5.275	5.06	6.012	6.122	6.106	5.55
1891..	5.55	5.025	5.125	5.00	4.85	5.083	5.063	5.01	4.958	5.02	4.83	4.75	5.02
1892..	4.69	1.62	4.89	4.68	4.79	4.71	4.78	4.69	4.53	4.41	4.47	4.40	4.63
1893..	4.39	4.33	4.28	4.38	4.41	4.27	4.13	3.89	3.69	3.68	3.65	3.80	4.075
1894..	3 56	3.85	3.89	3.62	3.47	3.40	3.43	3.38	3.44	3.45	3.36	3.43	3.52
1895..	3.28	3.20	3.23	3.30	3.50	3.65	3.75	4.15	4.30	4.10	3.55	3.49	3.63
1896..	3.75	4.03	4.20	4.09	3.98	4.10	3.97	3.76	3.60	3.72	3.99	4.14	3.94
1897..	3.91	4.02	4.12	4.13	4.21	4.21	4.32	4.26	4.18	4.17	4.03	3.89	4.12
1898..	3.96	4.04	4.25	4.26	4.27	4.77	4.66	4.58	4.67	4.98	5.29	5.10	4.57
1899..	5.34	6.28	6.31	6.67	6.88	5.98	5.82	5.65	5.50	5.32	4.64	4.66	5.75
1900..	4.65	4.64	4.60	4.71	4.53	4.29	4.28	4.17	4.11	4.15	4.29	4.25	4.39
1901..	4.13	4.01	3.91	3.98	4.04	3.99	3.95	3.99	4.08	4.23	4.29	4.31	4.08
1902.	4.27	4.15	4.28	4.37	4.47	4.96	5.27	5.44	5.49	5.38	5.18	4.78	4.84
1903..	4.87	5.04	5.35	5.55	5.63	5.70	5.66	5.73	5.69	5.51	5.39	4.73	5.40
1904 .	4.863	4.916	5.057	5.219	5.031	4.760	4.873	4.866	5.046	5.181	5.513	5.872	5.100
1905..	6.190	6.139	6.067	5.817	5.434	5.190	5.396	5.706	5.887	6.087	6.145	6.522	5.882

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

(From statistics of the Metallgesellschaft.)

	£	s.	d.		£	s.	d.		£	s.	d.		£	s.	d.
1870	18	10	4	1879	16	12	0	1888	18	1	9	1897	17	9	10
1871	18	8	9	1880	18	7	1	1889	19	15	7	1898	20	8	9
1872	22	9	1	1881	16	5	6	1890	23	4	6	1899	24	17	2
1873	26	3	6	1882	16	19	9	1891	23	4	8	1900	20	5	6
1874	22	17	7	1883	15	6	6	1892	20	16	6	1901	17	0	7
1875	24	1	4	1884	14	8	11	1893	17	8	0	1902	18	10	11
1876	23	6	3	1885	13	19	11	1894	15	9	2	1903	20	19	5
1877	19	18	8	1886	14	5	1	1895	14	12	2	1904	22	11	11
1878	17	17	10	1887	15	4	0	1896	16	11	10	1905	25	8	8

Equivalent Prices of Spelter in Pounds Sterling per 2,240 lb., and dollars and cents per 100 lb.

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

$$P=(P^1 \times E) \div 22.4$$

in which

P=price per 100 lb. in dollars and cents.
P^1=price in pounds sterling, shillings and pence being expressed in decimal parts of a pound.
E=the value of £1 in U.S. currency—i. e., at the current rate of exchange.

For example, the equivalent of £22 10s. 6d. per 2,240 lb. in dollars and cents per 100 lb., when exchange is at $4.866, is computed as follows:

£22 10s. 6d.=£22.525
£22.525×4.866÷22.4=$4.983

In the same manner the equivalents of £14 to £29 at rates of exchange of $4.83 to $4.89, with intervals of £1 and 1c. respectively, have been calculated in the following table:

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

£ 1=	$4.83	$4.84	$4.85	$4.86	$4.87	$4.88	$4.89	Increase
£ 14	†$3.019	†$3.025	$3.031	$3.038	$3.043	†$3.050	$3.056	0.6250
15	3.234	3.241	3.248	3.254	3.259	3.268	3.275	0.6696
16	†3.450	3.457	3.464	3.471	3.479	3.486	3.493	0.7143
17	3.666	3.673	3.681	3.688	3.696	3.704	3.711	0.7589
18	3.881	3.889	3.897	3.905	3.913	3.922	3.929	0.8035
19	4.097	4.105	4.114	4.122	4.131	4.139	4.148	0.8482
20	4.313	4.321	4.330	4.339	4.348	4.357	4.366	0.8927
21	4.528	4.538	4.547	4.556	4.566	†4.575	4.584	0.9375
22	4.744	4.754	4.763	4.773	4.783	4.793	4.803	0.9821
23	4.959	4.970	4.980	4.990	5.000	5.011	5.021	1.0268
24	†5.175	5.186	5.196	5.207	5.218	5.229	5.239	1.0714
25	5.391	5.402	5.413	5.424	5.435	5.446	5.458	1.1161
26	5.606	5.618	5.629	5.641	5.653	5.664	5.676	1.1607
27	5.822	5.834	5.846	5.857	5.870	5.882	5.894	1.2054
28	6.038	†6.050	6.063	†6.075	6.088	†6.100	6.113	1.2500
29	6.253	6.266	6.279	6.292	6.305	6.318	6.331	1.2946
Increase	21.5625c.	21.6071c.	21.6518c.	21.6964c.	21.7411c.	21.7857c.	21.8304c.	0.0446

The figures marked with daggers in the above table are absolutely correct, the dividend giving with the divisor, 22.4, the quotient as entered, without remainder. The other figures in the table involve errors to the maximum extent of plus or minus $0.0005=1c. per 2,000 lb. The extreme right hand column gives the differences in cents per 100 lb. caused by fluctuations of 1c. in the rate of exchange, from which the equivalents of prices at intermediate rates of exchange can be found by interpolation. At the foot of each column headed by the rate of exchange is given the difference in cents per 100 lb.

caused by a difference of £1 per 2,240 lb., from which the equivalents of fractional values can be quickly arrived at by interpolation. It may be assumed roughly that a difference of 1s. per 2,240 lb. corresponds to 1c. per 100 lb., 6d.=0.5c. and 3d.=0.25c. for all rates of exchange between $4.83 and $4.89 In determining the equivalent of a fractional price with an intermediate rate of exchange with absolute accuracy a double interpolation must be made.

For convenient comparison of the prices of spelter at New York and London, the averages reported in pounds sterling per long ton have been reduced in the following table to the common basis of cents per pound. The statistics for New York have been taken from *The Mineral Industry;* those for London represent the price of ordinary Silesian spelter as reported by the Metallgesellschaft.

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

Year	New York	London	Year	New York	London
1871	.....	4.008	1889	5.023	4.299
1872	.....	4.881	1890	5.550	5.048
1873	.....	5.691	1891	5.020	5.049
1874	.....	4.973	1892	4.630	4.527
1875	7.000	5.232	1893	4.075	3.783
1876	7.250	5.068	1894	3.520	3.359
1877	6.030	4.333	1895	3.630	3.175
1878	4.880	3.888	1896	3.940	3.608
1879	5.036	3.609	1897	4.120	3.803
1880	5.510	3.989	1898	4.570	4.443
1881	5.243	3.538	1899	5.750	5.405
1882	5.325	3.693	1900	4.390	4.407
1883	4.495	3.329	1901	4.070	3.702
1884	4.443	3.140	1902	4.840	4.321
1885	4.345	3.043	1903	5.400	4.560
1886	4.400	3.097	1904	5.100	4.913
1887	4.625	3.302	1905	5.882	5.529
1888	4.910	3.932			

The price of spelter has been subject to wide fluctuations, especially in the United States, where the range has been from 7.875c. (the average for May, 1876) to 3.20c. (the average for February, 1895).* The maximum price was attained at a time when the market was controlled by a combination of producers which was organized for the purpose of enhancing the value of spelter and was temporarily successful in doing so. The minimum price was quoted during the period of depression which followed the panic of 1893.

The decline in the price of spelter which began in 1874 and continued, with only two checks, until 1885 is attributable chiefly to the new supply of rich ore that in 1870 began to be offered in the form of blende. At about the same time the United States began to be an important producer of zinc, its importations from abroad dwindled down, and this outlet for European spelter gradually became closed. Simultaneously there was a heavy increase in the world's production. Up to 1870 the spelter product of Europe was derived

* In February, 1895, spelter at one time touched 2.90c at St. Louis.

almost exclusively from calamine. Blende had been mined and smelted in Belgium as early as 1845, but the output of that Kingdom never attained much magnitude. The Belgian production of calamine had been on the wane since 1856. In 1870 the Scharley and Marie mines, which had previously been the most important producers in Upper Silesia, came to the end of their resources, but in the same year the blende of the district began to be utilized, although its production did not assume large proportions until nearly 10 years later.

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

Since 1890 the predominant features in the zinc market have been the sagging of prices under increasing production in the early part of the decade; the enormously increasing production in the United States and the beginning (in 1896) of large exports to Europe; the decrease in the cost of smelting in the United States because of the utilization of the natural gas resources of Kansas and improvements in the metallurgical practice; and the increase in the cost of smelting in Europe because of the rise in the value of coal, especially toward the end of the decade. There was a period of industrial depression in both Europe and America in 1893 to 1895; and a recovery, which culminated in a boom in 1899 and the early part of 1900; followed by a depression in Europe, which caused a great decline in the price of spelter there, and sympathetically a corresponding decline in the price in the United States, although the period of general industrial prosperity continued here. It will be observed, from the accompanying tables how since 1890 the American price for spelter has preserved a rather constant relation to the European price, although spelter in Europe can no longer, under normal conditions, enter the American market. Beginning in 1902, and continuing to the present writing, there was another period of great industrial activity, creating increased demand for spelter, which again carried prices to a high level.

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

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

Yr.	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	12 mos.
	$	$	$	$	$	$	$	$	$	$	$	$	$
1896	24.00	23.50	23.00	23.00	21.50	21.00	21.50	21.00	20.00	20.50	23.50	25.50	22.33
1897	22.125	21.50	21.00	21.125	21.60	21.875	22.50	22.50	22.625	22.75	23.50	24.25	22.28
1898	23.00	22.50	23.00	24.62	26.50	28.50	28.00	28.37	31.00	33.70	36.25	37.00	28.44
1899	32.25	43.37	43.40	51.50	50.50	45.50	44.20	45.00	43.75	43.50	35.00	36.00	38.54
1900	30.23	29.36	28.45	28.42	26.92	25.00	24.23	25.67	24.25	24.25	24.45	25.40	26.50
1901	23.73	23.96	23.70	24.58	24.38	24.22	24.68	23.88	22.82	21.63	26.15	28.24	24.21
1902	26.75	27.00	28.00	28.85	20.23	34.10	34.37	32.50	33.58	33.58	32.10	29.25	30.73
1903	34.50	32.50	35.75	37.75	36.60	36.50	36.00	36.00	34.40	34.40	30.75	30.00	34.44
1904	32.12	34.00	36.00	36.40	34.63	32.62	35.00	37.00	40.40	40.00	44.25	46.13	37.40
1905	51.04	53.65	47.40	43.93	43.74	40.75	43.00	50.24	46.80	49.37	50.37	47.67	47.40

The statistics of the above table are taken from various volumes of *The Mineral Industry* with the exception of those for 1900 and 1901, for which years the values reported by the U. S. Geological Survey have been adopted. *The Mineral Industry* and the U. S. Geological Survey agree as to the values for the years 1896–1899, both inclusive. There are no complete statistics available for the years previous to 1896. *The Mineral Industry* reports the following averages: 1889, $25; 1890, $23.90: 1891, $25.90; 1892, $22.50; 1893, $19.25; 1894, $17.10. *The Mineral Industry* made no quotation for 1895; Prof. Erasmus Haworth, State Geologist of Kansas, gives the average for that year as $19.68. In the earlier years the figures represent the value of the average grade of ore marketed. Since 1899 they represent the average value of ore assaying 60% zinc.

In the early part of the decade, 1891–1900, the average grade of the ore produced in the district was probably between 56% and 58% Zn; certainly not more than 58%. About 1900 the average was probably very close to 60%, which is rated as the "standard" ore of the district. A good deal of ore assaying 62% to 63% Zn is produced, and occasionally lots assaying as high as 64.5%. During the last two or three years the average grade has been about 58% zinc, which is representative of the present production.

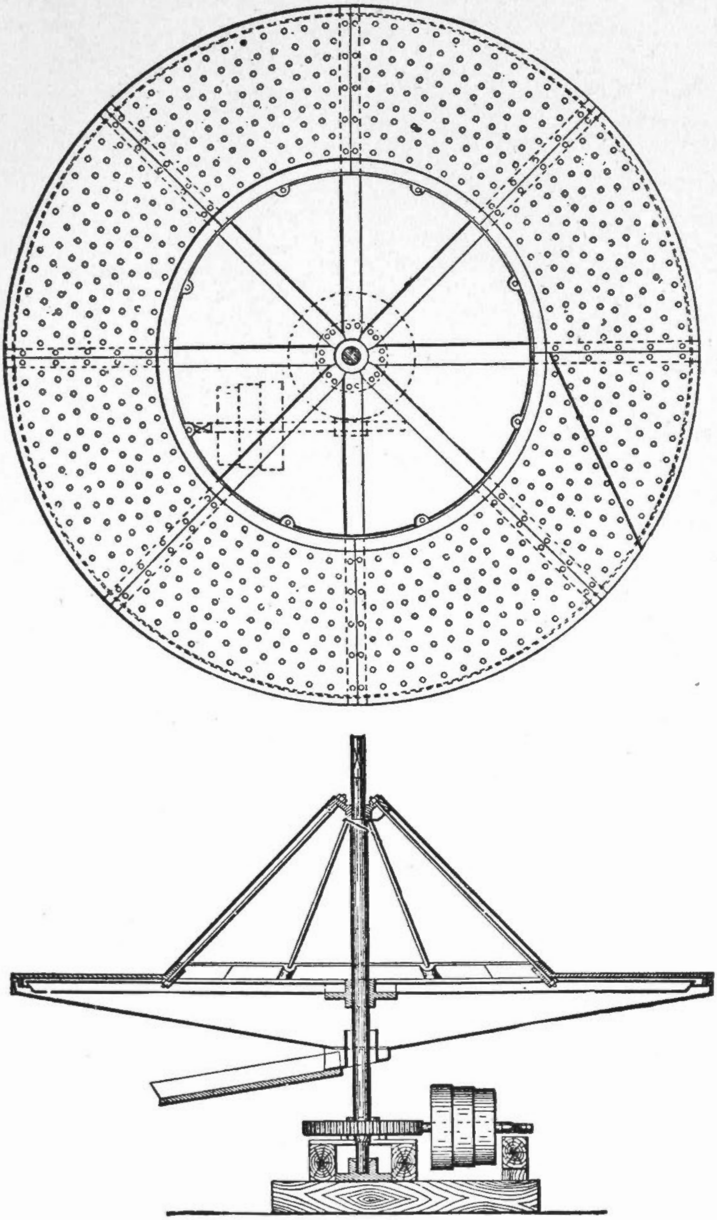


FIG. 1—Plan. FIG. 2—Vertical Section.

ORE PICKING TABLE.

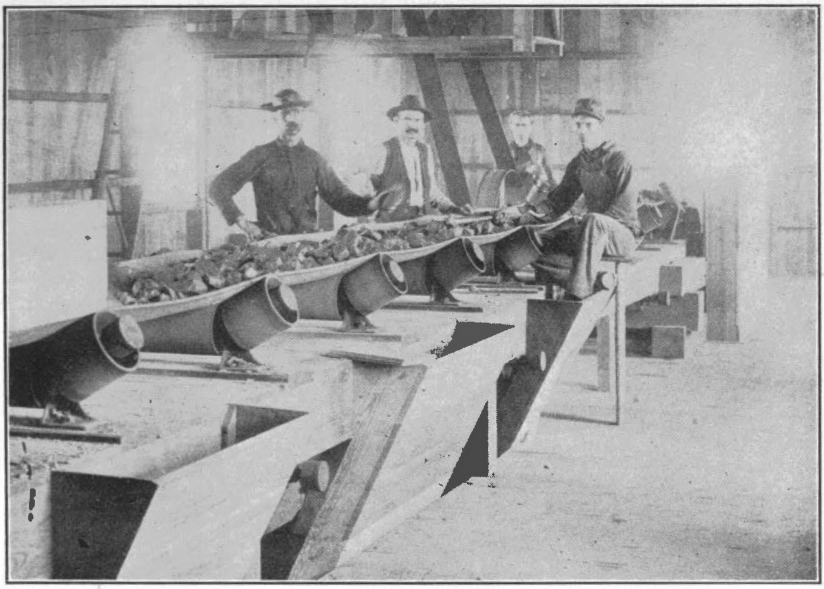


FIG. 3.—Ore-picking Belt.

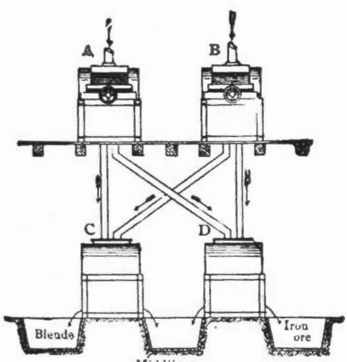


FIG. 4—Arrangement of Magnetic Separators at Friederichssegen, Germany.

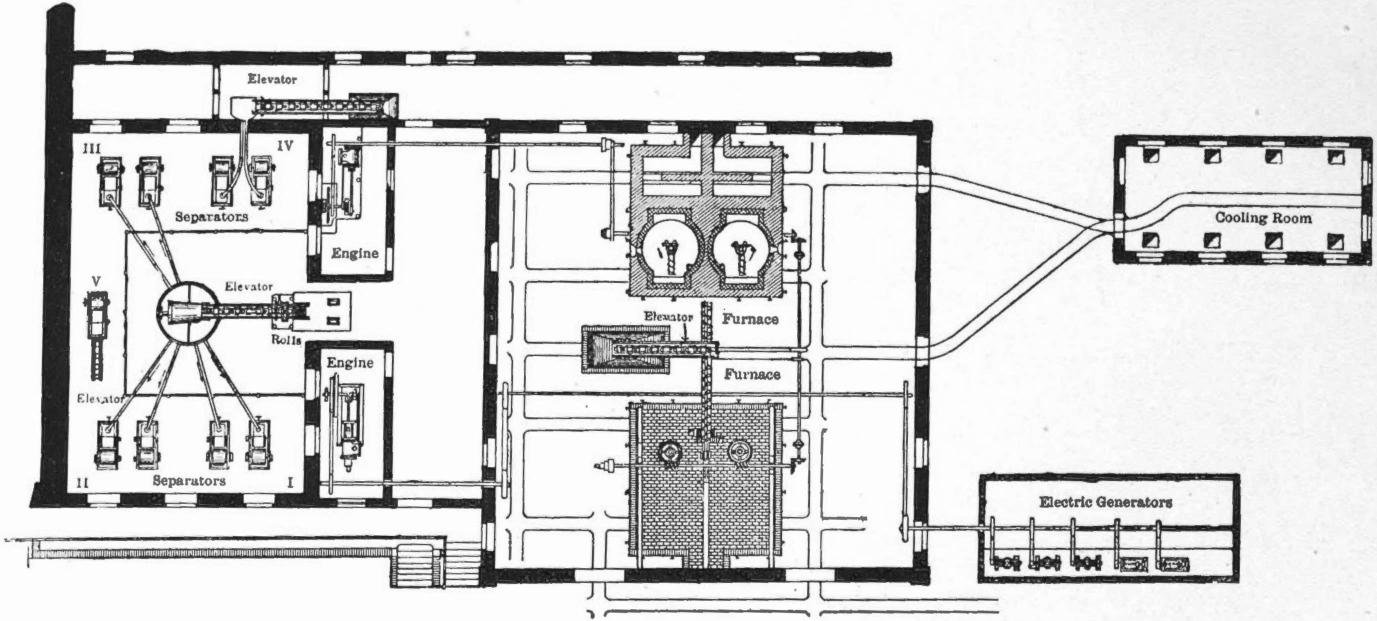


FIG. 5—Plan of Magnetic Separating Works at Friedrichssegen, Germany.

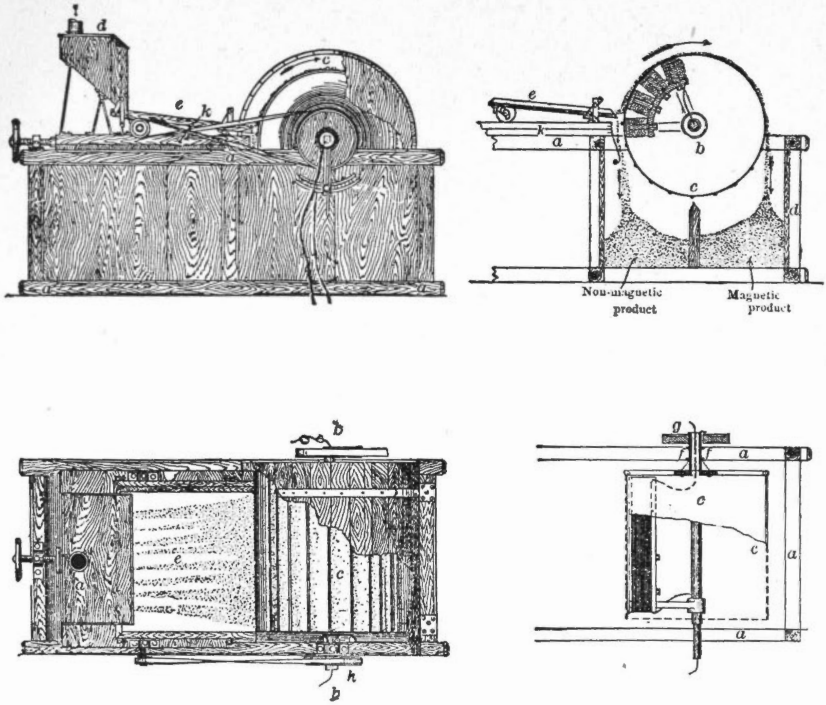
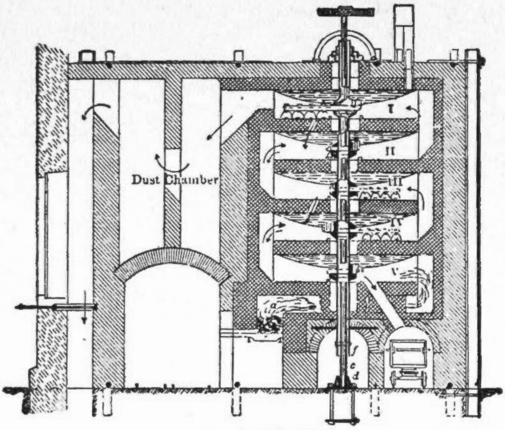
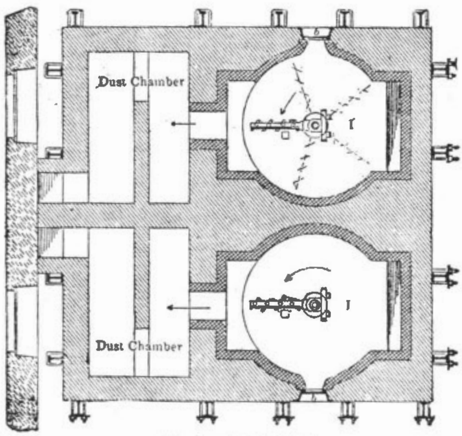


FIG. 6 to 9—Magnetic Separator formerly used at Friedrichssegen, Germany.



Vertical Section



Horizontal Section

FIGS. 10—11—Calcining Furnace used at Friedrichssegen, Germany.

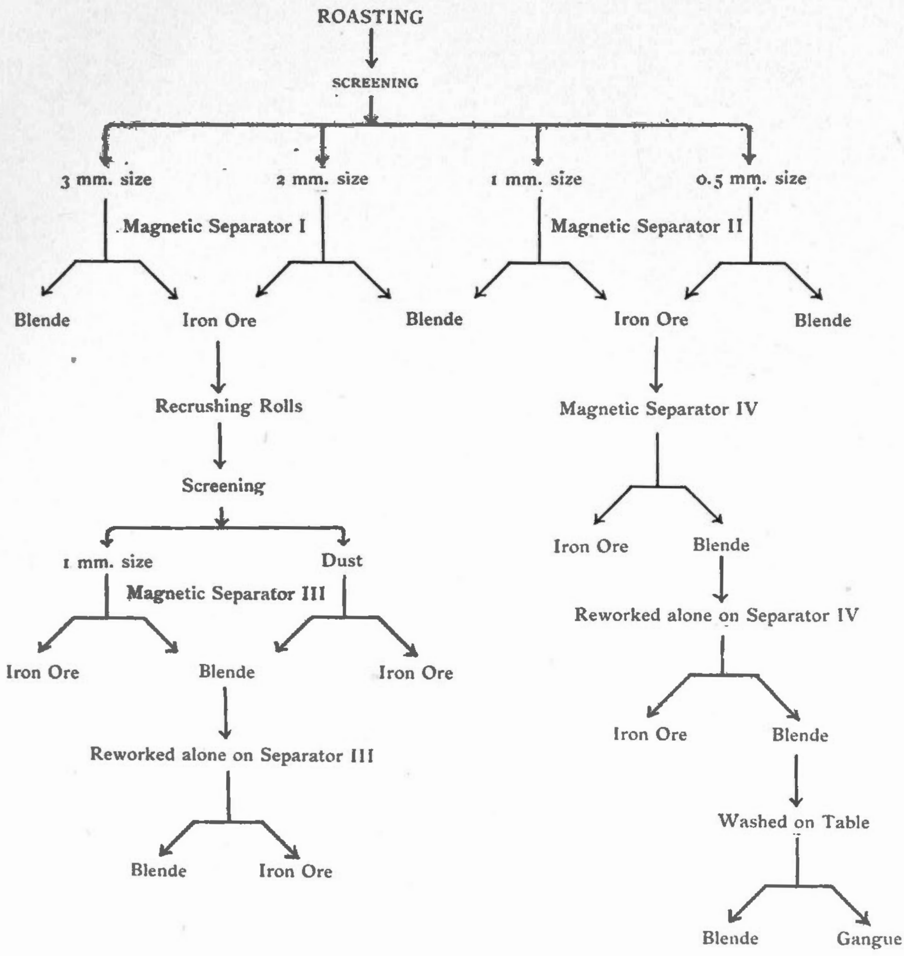


FIG. 12—Flow Sheet at Meirn, Tyrol.

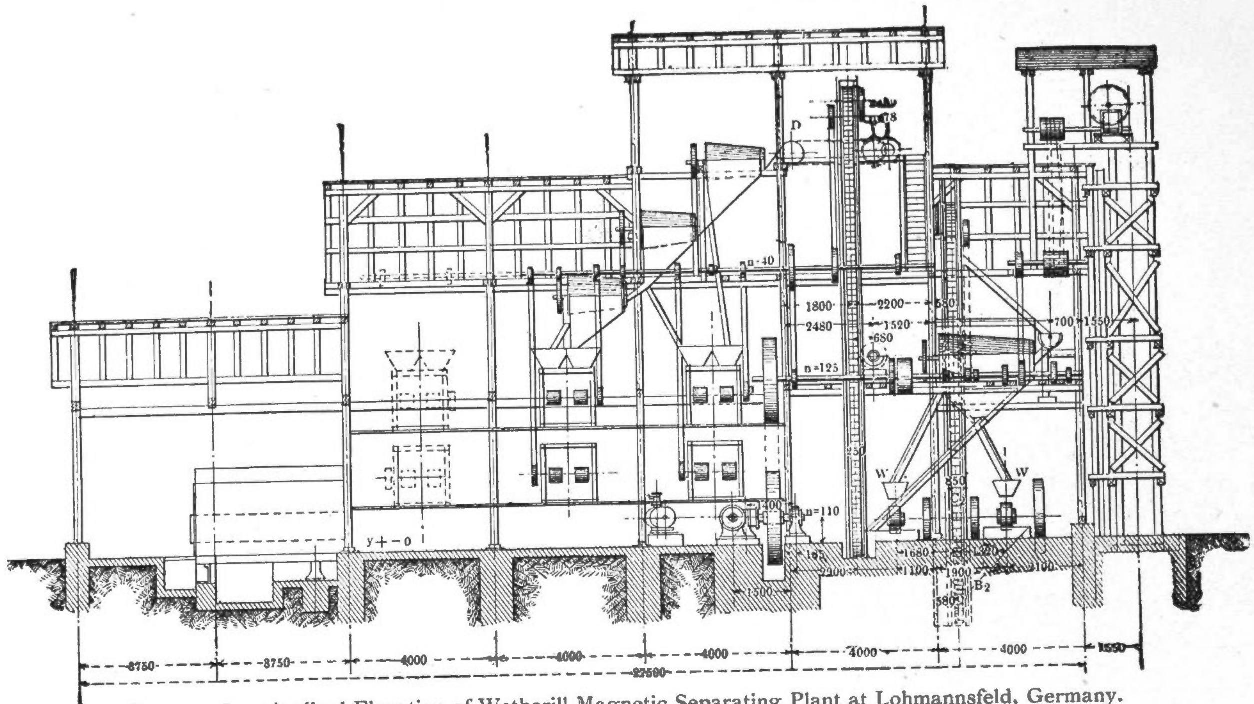


FIG. 13—Longitudinal Elevation of Wetherill Magnetic Separating Plant at Lohmannsfeld, Germany.

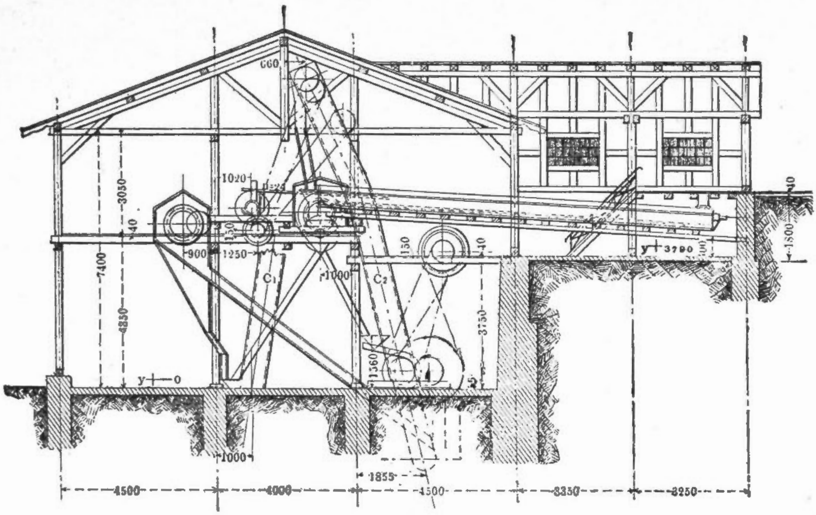


FIG. 14—Transverse Section of Wetherill Magnetic Separating Plant at Lohmannsfield, Germany.

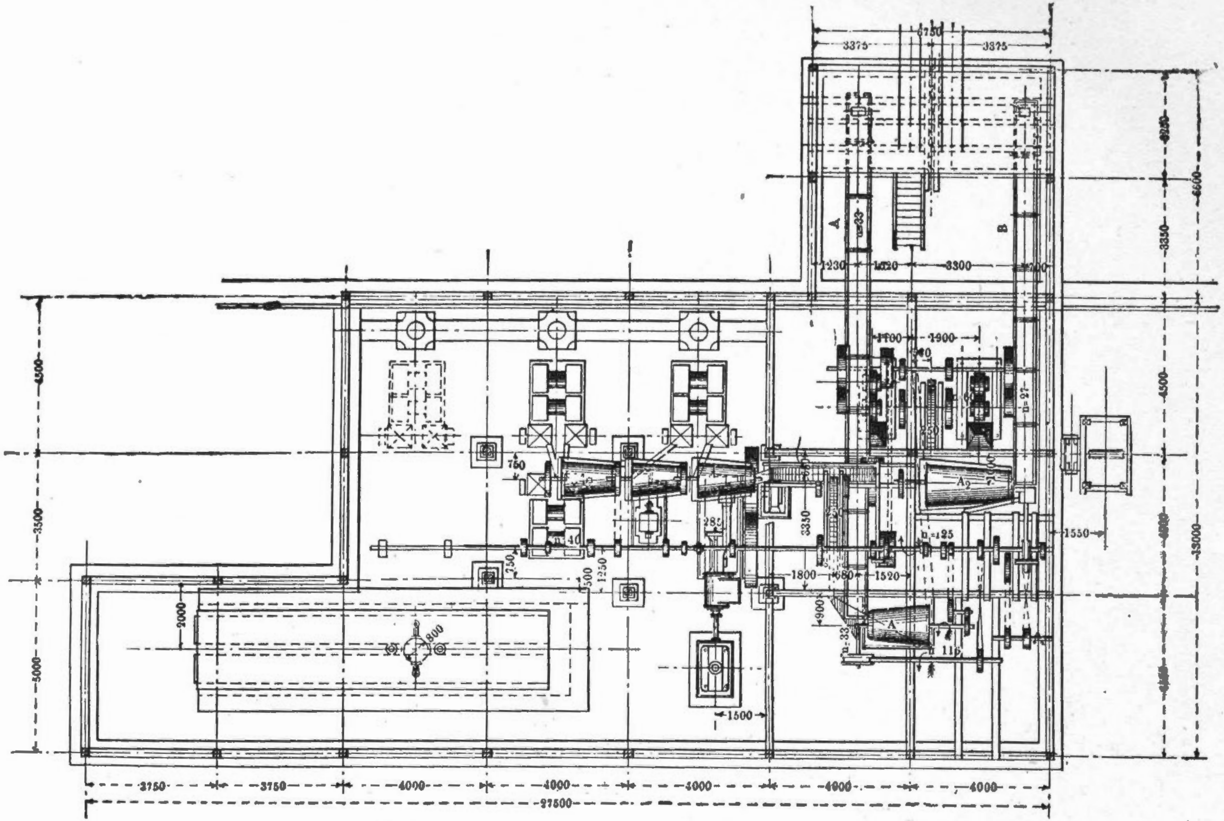


FIG. 15—Plan of Wetherill Magnetic Separating Plant at Lohmannsfeld, Germany.

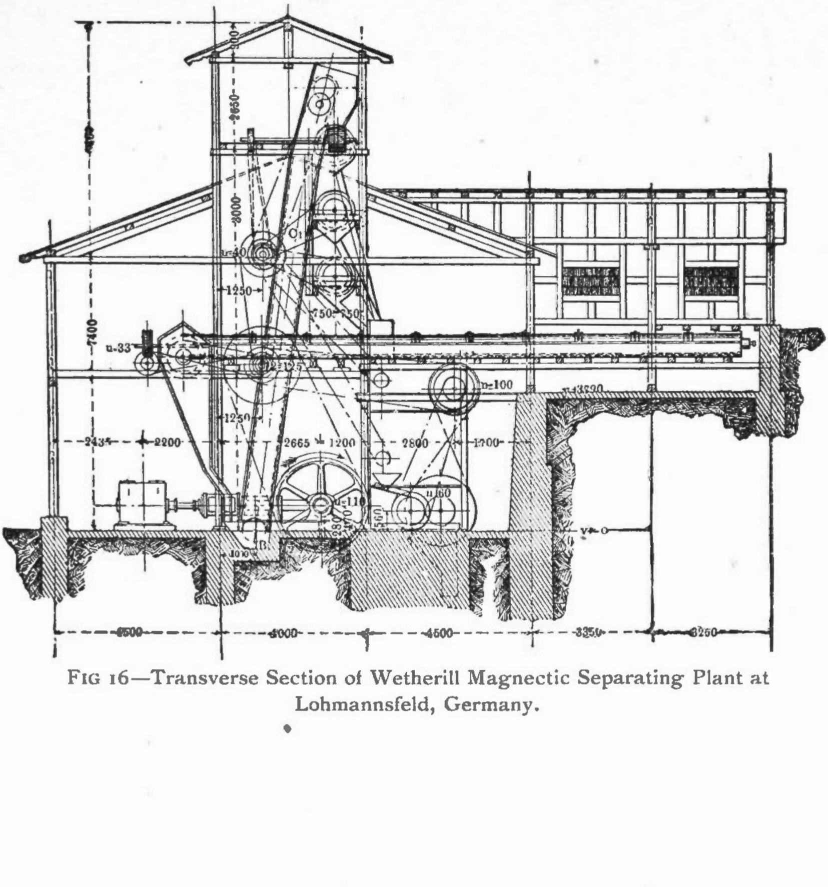


FIG 16—Transverse Section of Wetherill Magnectic Separating Plant at Lohmannsfeld, Germany.

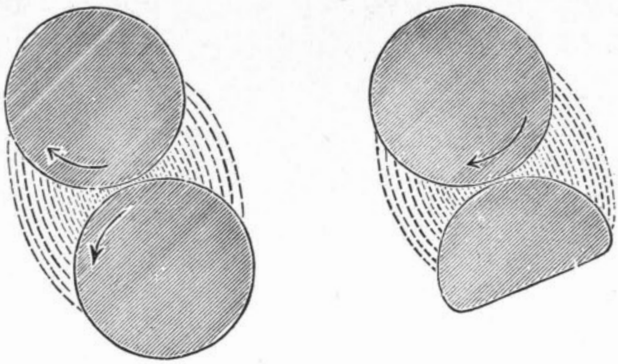
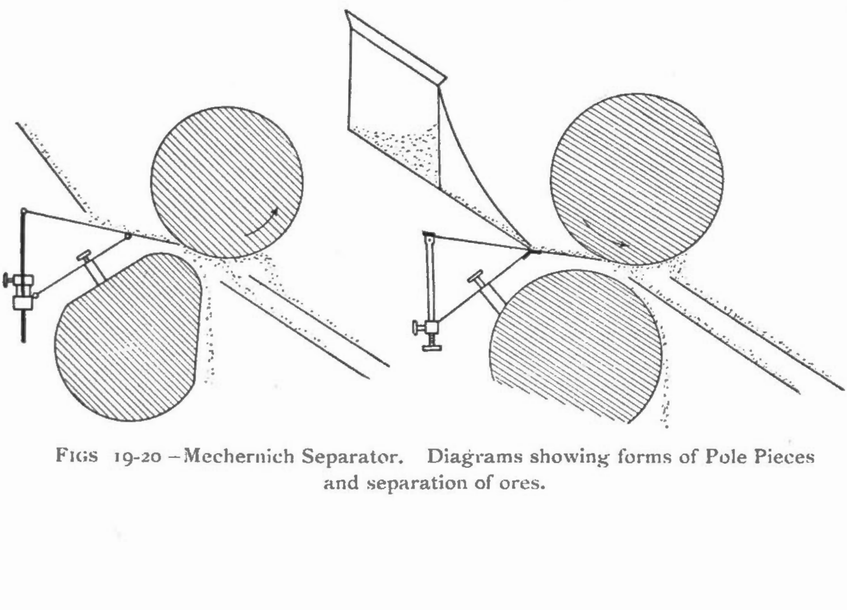


FIG. 17-18—Mechernich Separator. Diagrams showing forms of Pole Pieces and Distribution of Lines Force.



FIGS 19-20 – Mechernich Separator. Diagrams showing forms of Pole Pieces and separation of ores.

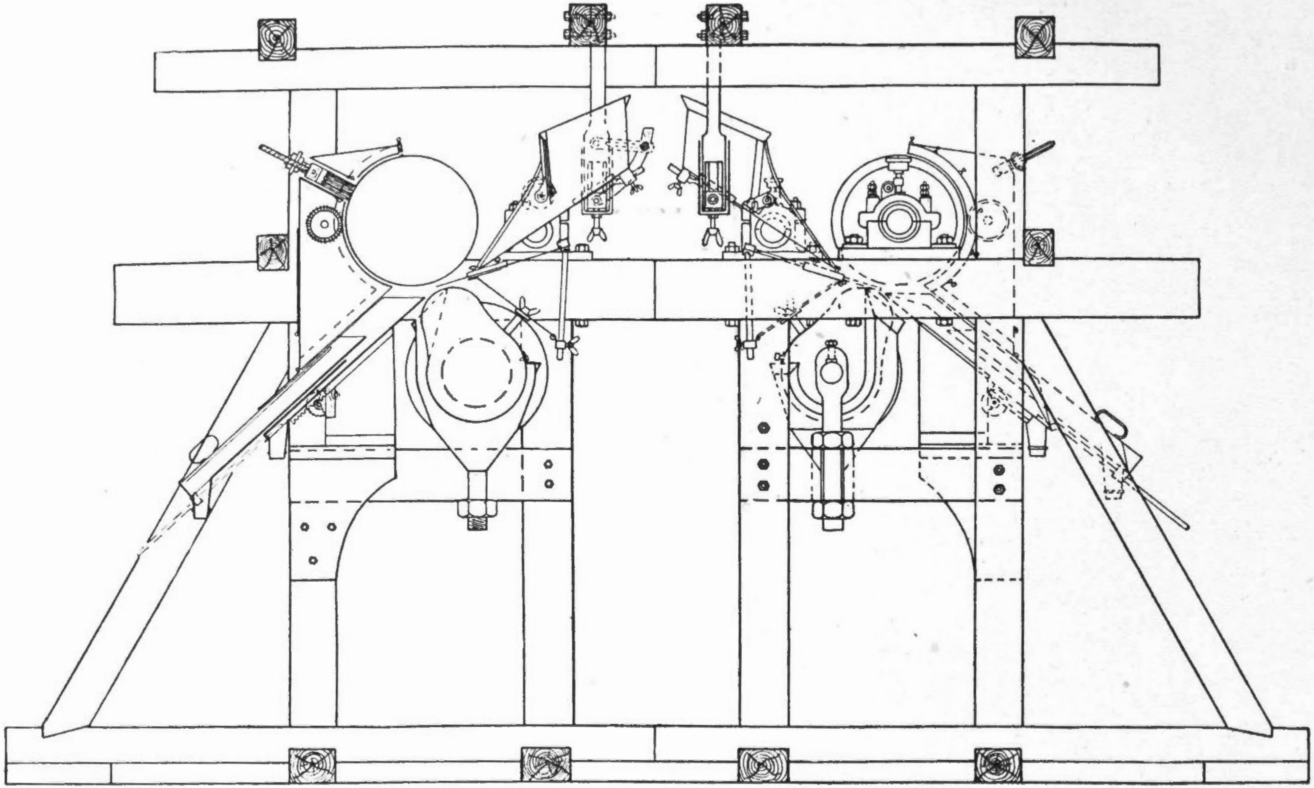


FIG. 21—Mechernich Separator. Side Elevation.

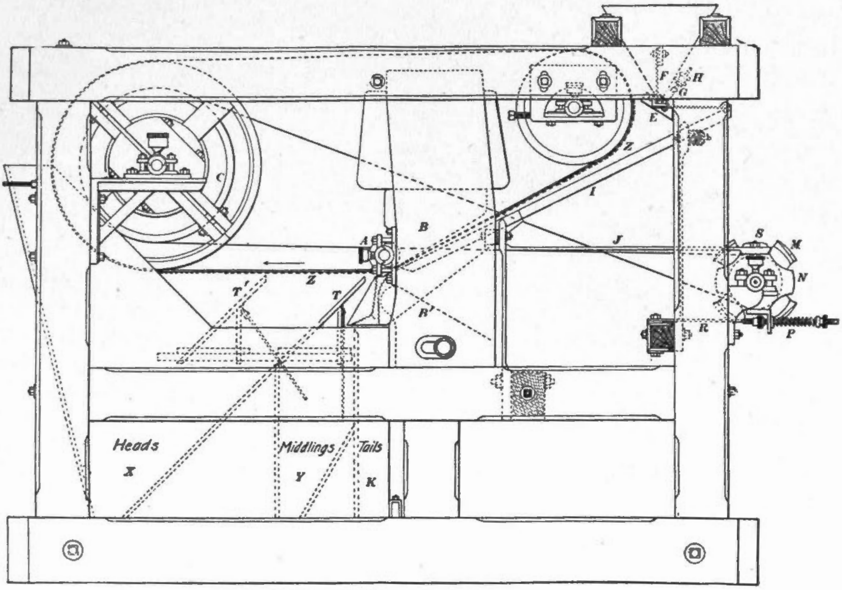


FIG. 22—Knowles Separator. Side Elevation.

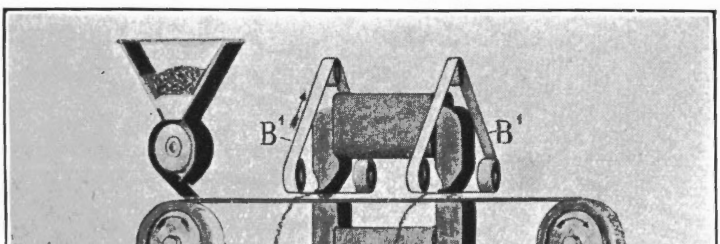


FIG. 23—Principle of Wetherill, Type "E" Separator.

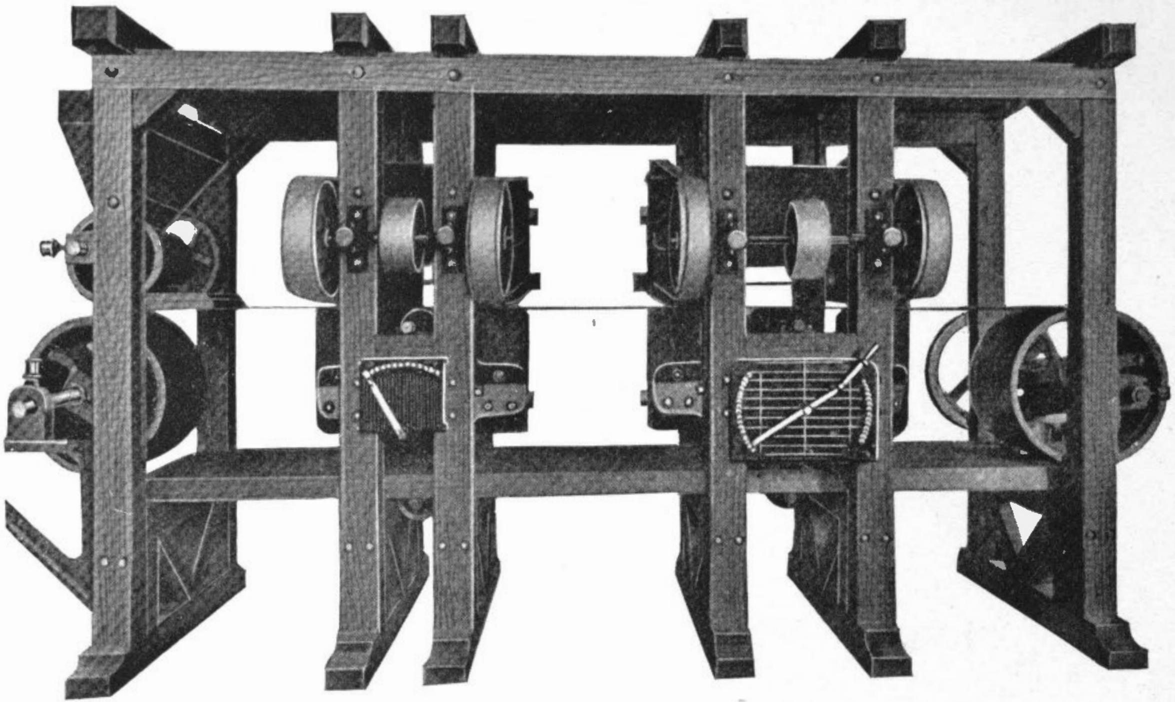


FIG. 24—Wetherill, Type "E," No. 2 Separator.

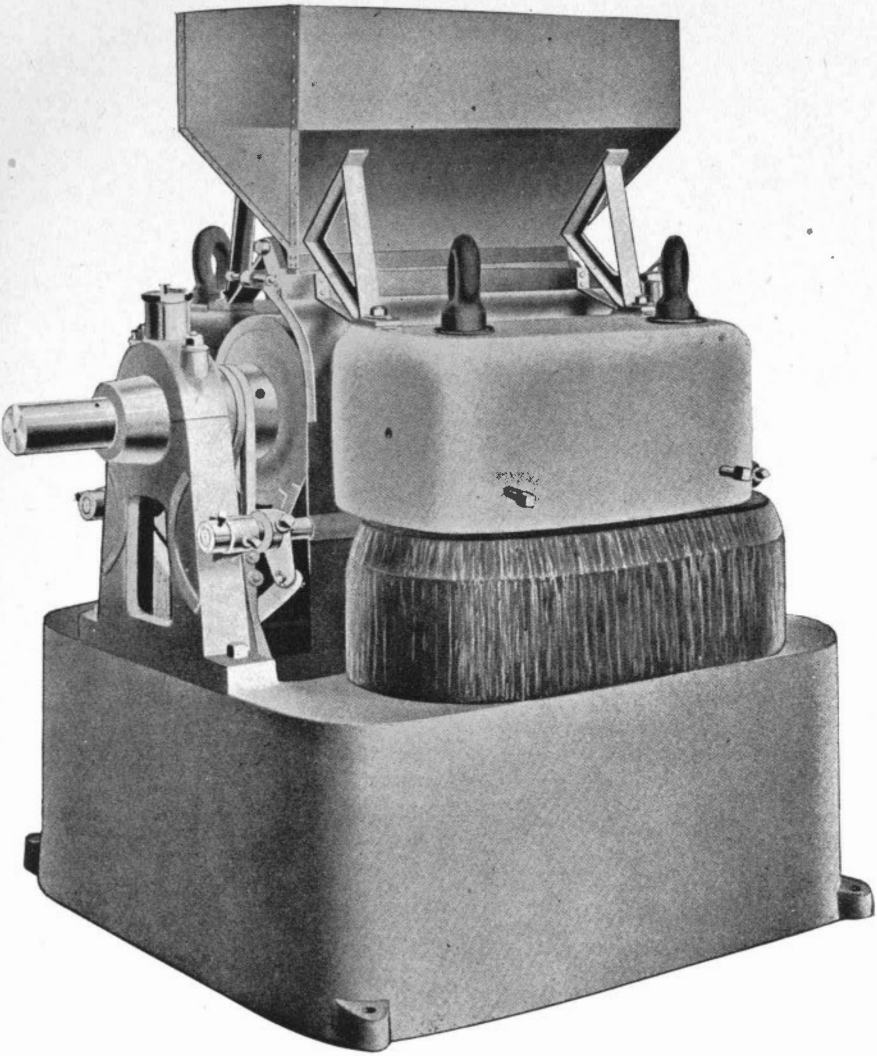


Fig. 25—International Separator

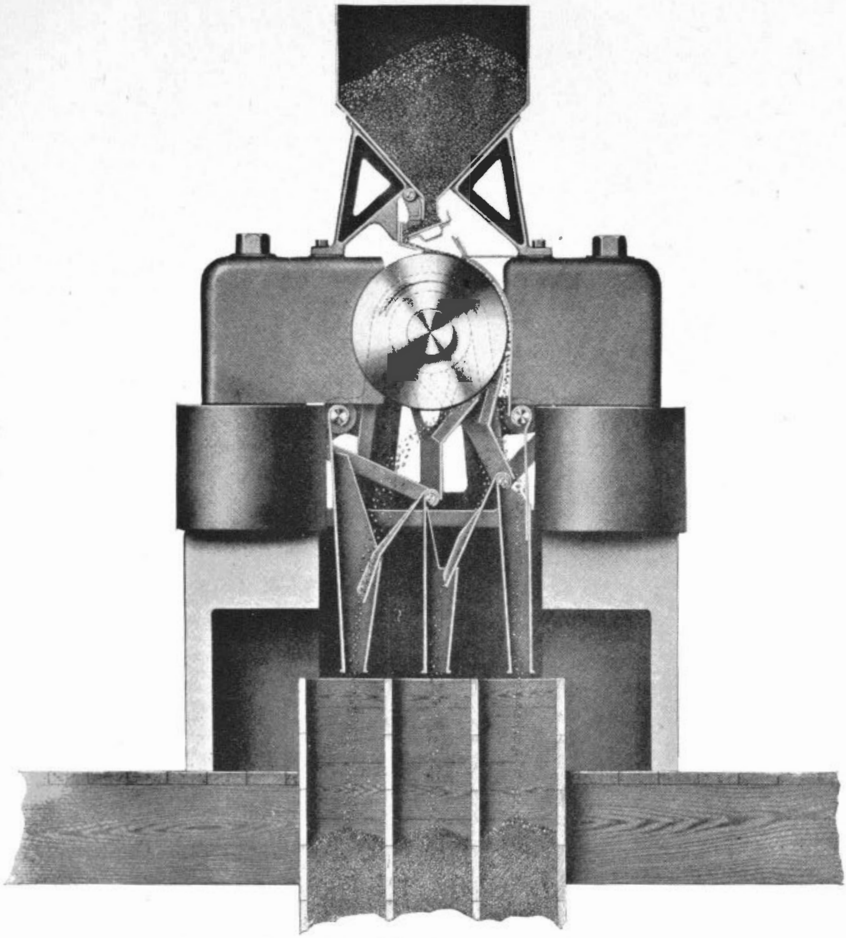


FIG. 26—International Separator. Sectional View.

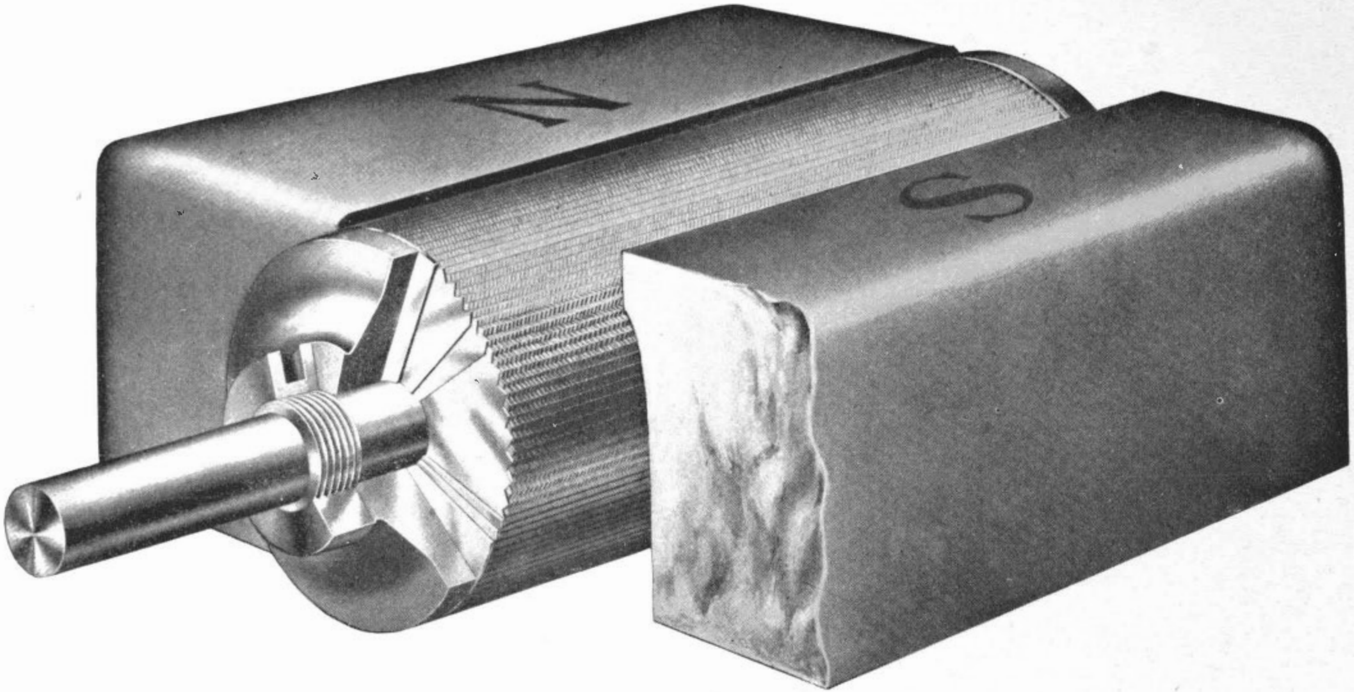


FIG. 27.—International Separator. Armature and Pole Pieces.

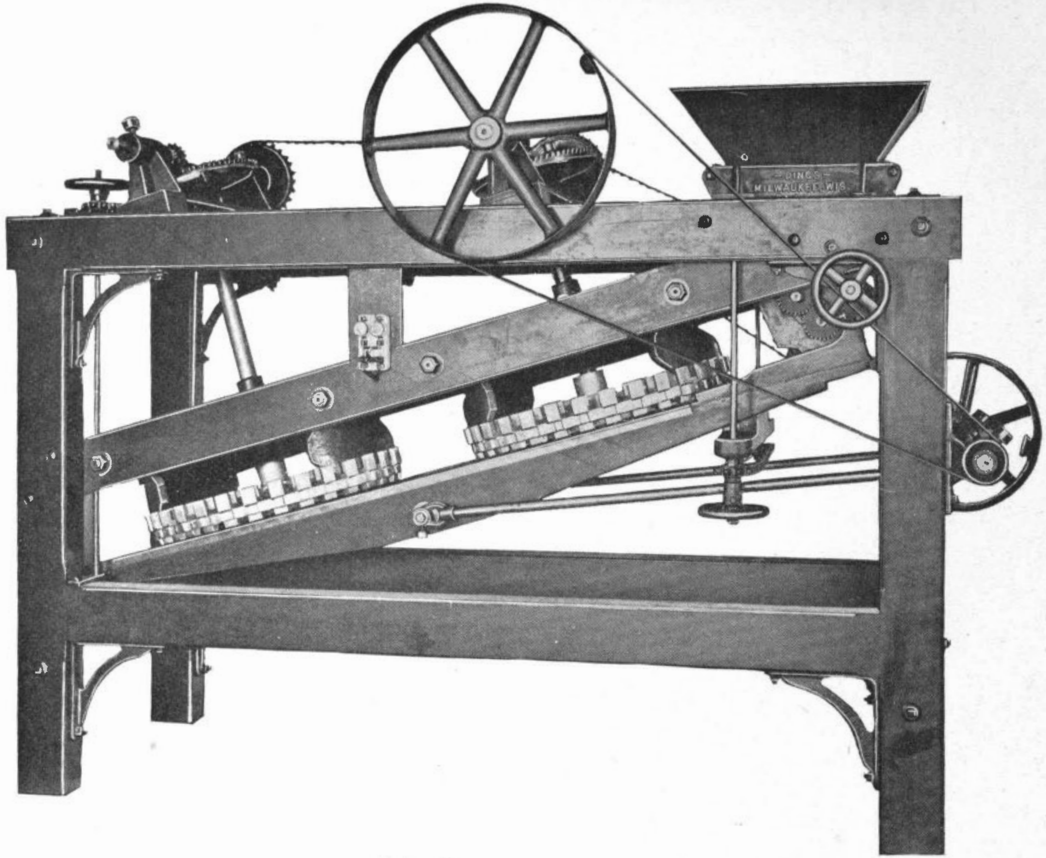


FIG. 28.—Dings Separator.

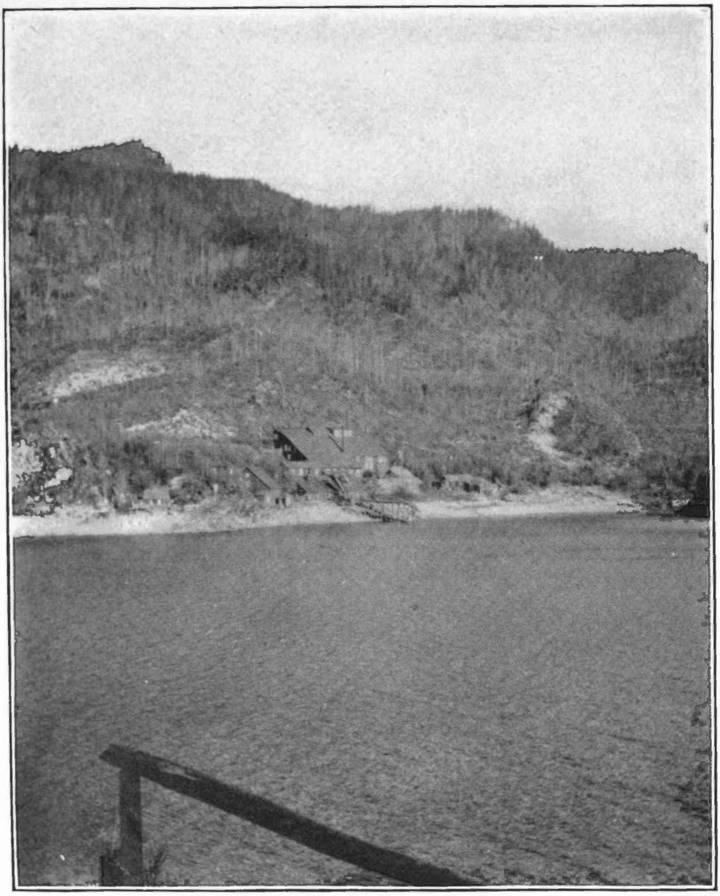


FIG. 29 —WORK OF KOOTENAY ORE CO , KASLO.

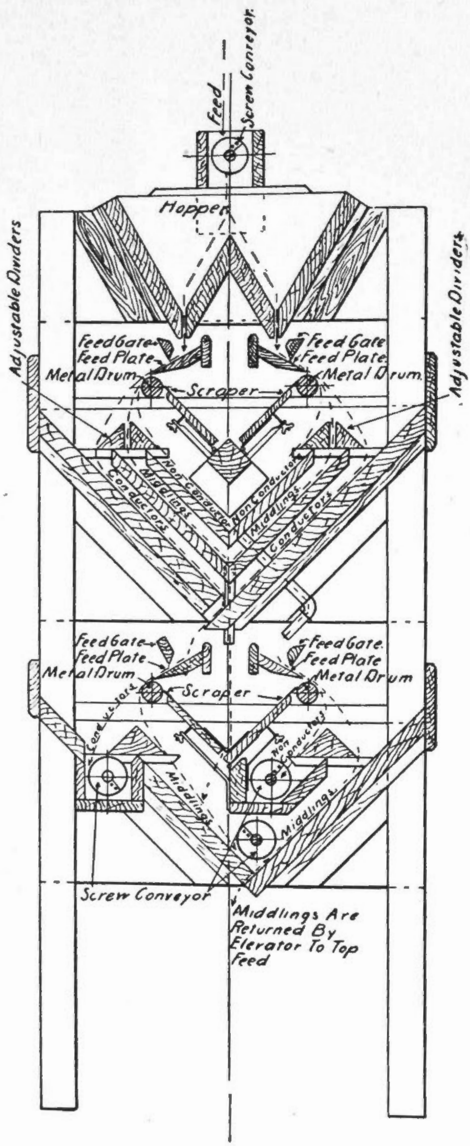


FIG. 30.—Blake Separator. Vertical Section.

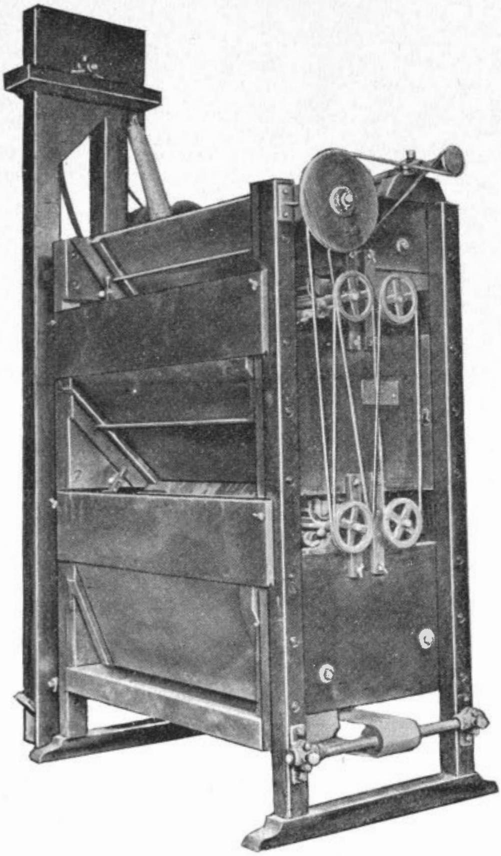


FIG. 31—Blake Separator (Experimental Testing Size) In the U.S. Geological Survey's testing plant at Portland Oregon. Geological Survey, April, 1906.

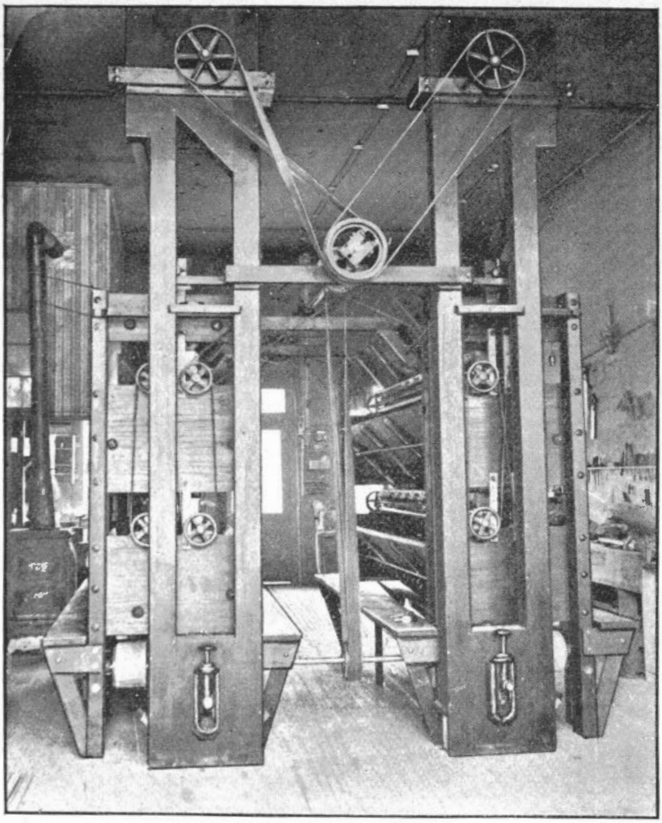


FIG. 32—Standard Blake Separator. (Double) Set up in Denver Shop for Testing, April, 1901.

REPORT

ON THE

ZINC MINES OF THE EAST AND WEST KOOTENAYS.

BY

PHILIP ARGALL.

Member of the Royal Irish Academy. Member of the American Institute of Mining Engineers. Member of the Institution of Mining and Metallurgy. Fellow of the Geological Society of America. President of the Colorado Scientific Society, Etc.

1906.

SIR,—

Herewith I beg to submit my report on the mines of the East and West Kootenay districts, British Columbia, that showed developed resources of zinc ore in 1905.

This report is based on an examination of the mines which was begun Sept. 1, 1905, and was concluded Nov. 20, 1905. The Geological notes, based on one or two days examination of the workings of a given mine, are intended to convey only a general idea ot the geology of the ore deposits; any detailed geological examination was not only impossible to carry out in the time alloted to the field work, but was also outside the scope of this report.

The tons referred to in this report are uniformly of 2000 pounds. Percentages of lead are stated on the results of assays made by the volumetric method.

Yours respectfully,

PHILIP ARGALL.

WALTER RENTON INGALLS, ESQ.,

Chief Commissioner.

AINSWORTH CAMP.

Ainsworth Camp, or Hot Springs, as it was formerly called, is one of the oldest mining camps in British Columbia. It is situated on the eastern shore of Lake Kootenay, about 28 miles by steamer from Nelson. The formation is classed by The Geological Survey of Canada as "Shuswap series," Crystalline Archaean; consisting of mica schists, quartzites, gneiss and crystalline limestone, penetrated by many recent basic dikes. The ore deposits in the schists usually conform to the planes of schistosity, but in the limestone are generally of irregular nature, usually replacement.

The veins while fairly rich in lead are of low silver value, consequently when the richer silver lead veins of the Slocan were discovered in 1891, there was a stampede from Ainsworth to the richer fields, and since then the camp is best described as in a state of suspended animation. Zinc ore occurs quite extensively with the galena in the veins; hitherto it has been as far as possible eliminated in concentrating the ores and sent down the creeks with the waste. With the present high price of zinc and lead, and the decided demand for zinc ore of all classes, Ainsworth should again come to the front, more particularly if modern methods of mining and concentration are put into efficient use. The ore deposits are on the whole of fair size—the Blue Bell on the west shore of the lake is, however, an enormous mineral deposit—and the tenor of the ores in lead and zinc is fairly good; hence with efficient mining and milling on a scale commensurate with the deposits, success would undoubtedly accrue and Ainsworth once again become a lively mining district.

Several warm springs issue from the hill side in and near the town of Ainsworth and have deposited beds of travertine on the steep slopes and along the lake shore. The following analysis of these waters is said to have been made in 1899, by A. H. Holdich, Royal School of Mines, London, England:—

"The water was colourless, not quite clear, no particular smell, but taste salty. Reaction to litmus paper distinctly alkaline.

The solid matter in solution is as follows:—

Sodium carbonate	31.1	grains per Imperial gallon.
Sodium silicate	9.5	" "
Sodium chloride	7.2	" "
Lime carbonate	26.2	" "
Magnesia sulphate	3.6	" "
Oxide of iron	0.9	" "
Total solids	78.5	"

No bromine or iodine, and the iron which probably exists as carbonate is very small. I consider the water too alkaline for general use though there may be special causes where it might be useful from a medical point of view."

The amount of carbonates and silica in solution is very suggestive, and it may be that mineral veins are now in course of formation by these waters in rock fissures and cavities adjacent to the town, while fully matured veins are being worked in similar formation three quarters of a mile away.

HIGHLANDER MILL AND MINING COMPANY.

The property of the Highlander Mill and Mining Company at Ainsworth Camp consists of seven (7) claims aggregating 380 acres, developed by 2,766 feet of tunnel work, 1,227 feet of drifts in the veins and 430 feet of winzes and raises.

The Highlander vein varies from 25 feet down to about 2 feet in width, and may be said to average 3 feet of concentrating ore in the shoots opened on the main Highlander tunnel-horizon south drift. About 700 tons of lead concentrates and 100 tons of hand sorted lead ore have been shipped from this property and is said to average 67% lead, 4% zinc and 22 ozs. of silver.

HIGHLANDER MINE.

The Highlander vein was discovered in 1890. It outcrops on the summit of the steep escarpment on the western side of Lake Kootenay, about 1,100 feet above high water. A short tunnel intersects the vein 100 feet below its outcrop, and development is continued by a shaft for a depth of 170 feet from the outcrop of the vein. Later, a main tunnel was started, now known as the Highlander tunnel, to open up this vein at a depth of 750 feet vertically below its apex, or 1,000 feet on the dip of the vein. The portal of this tunnel is situated about 350 feet above the lake and one mile south of the town of Ainsworth.

After penetrating the wash, the tunnel entered the mica schists of the district, and at a distance of 225 feet from the portal intersected what is known as the "Tariff" vein. (See Plate No. I). At 350 feet the schists gradually became harder, passing by insensible gradations into gneiss, through which the tunnel penetrates for a distance of 1,200 feet. In the centre of this mass of gneiss the crystallization becomes coarse, the rock in places presenting a granitic appearance, but on nearing the main lode again becomes schistose and the vein is intersected at 1,560 feet from the portal, with a westerly dip of 45° giving the following general section. The footwall portion shows a banded structure of quartz and dark slaty rock, containing some seams of calcite, occupying a width of 2 feet (Plate No. II), next comes 2 feet of gray porphyritic vein filling, possibly a portion of a small dike, the hanging wall of which is polished and shows slickenside markings. Resting on this wall is a seam of quartz and slate breccia cemented by a porphyritic ground mass. The remaining portion of the vein consists of dark schists containing irregular lenses of vein quartz, the whole occupying a width of 25 feet between walls, but does not contain pay ore in any part, truly a rather disappointing showing after so much expense. On drifting to the south, however, pay ore was found in the foot wall portion of the vein.

On reference to Plate No. I it will be seen that this main vein was cut in the exact position that the prolongation of the dip from the surface workings on the Highlander vein would indicate; therefore it was assumed to be the Highlander vein; but considering the 900 feet of unexplored ground between this tunnel and the surface workings on the Highlander, it must be freely admitted that it is simply a matter of conjecture. A basic dyke four feet in thickness cuts almost vertically through the vein on the tunnel horizon, as shown in Plates I and II.

The hanging wall streak of this composite lode is continuous quartz that has been drifted on northerly for about 80 feet, showing a fairly regular seam (averaging 30 inches thick) of white and blocky quartz, but devoid of mineral.

Crossing these quartz lenses in the tunnel, at about right angles to the dip, is a sheeted zone showing four open fissures, varying from 0.5 to 4 inches in width, as seen on the line of the main tunnel; giving a basis for the impression that this portion of the vein has an easterly dip. Passing westerly along the main tunnel, the mica schists become harder, more silicified, and various breaks or open fissures occur between the hanging wall of the Highlander vein and the face of the main tunnel, a distance of 1,014 feet.

On account of poor ventilation and the amount of water that was issuing from the joints and fissures, I found it impossible to examine carefully the rock structure between the Highlander vein and the face of the tunnel. One large fissure was, however, encountered, said to contain mostly loose calcite, which burst out several times in the tunnel, but was timbered up closely at the time of my examination.

At the present face of the tunnel, 1,014 feet from the hanging wall of the Highlander vein, another vein was encountered, with a strike North 50° West, and dip 25° westerly. This has been opened up on both sides of the tunnel for a total distance of 50 feet, and consists of an irregular quartz lens varying from 2 inches up to 15 inches in thickness, showing some very pale iron pyrites with a little siderite and calcite, a clay gouge of about 6 inches and a hanging wall of mica schist. This point is 2,610 feet from the portal of the tunnel and about 1,400 feet vertical below the surface of the mountain. The vein conforms, as far as can be seen, with the strike and dip of the schists, and while the showing is very poor, there is a possibility that drifting on the strike of the vein might open up some pockets of pay ore. Work, however, had been suspended in this place for some time prior to my examination.

The last 300 feet in the main tunnel is a poor piece of work, crooked, and at extremely bad grade, but on account of the numerous slips and open fissures that have been encountered, all of which are discharging considerable water, it was doubtless an expensive and rather difficult place to operate.

Returning now to the point where the Highlander vein was intersected (1,560 feet from the portal of the tunnel) a drift extends southerly along the foot wall; while the vein was entirely barren where intersected, some blende was discovered at about 50 feet south of the tunnel and continued for 100 feet in length; then 50 feet of barren vein came in, followed by about 150 feet of vein matter, averaging about 5 feet in width and containing about 2 feet of pay streak, showing zinc blende and galena. Sample No. 51 was taken across the pay streak 320 feet south of the tunnel, and gave the following result: Silver 4 ozs., Lead 21.8%, Zinc 7.2%; width sampled 2 feet. At 400 feet south a winze had been sunk 52 feet, the vein having here a dip of 45 degrees to the west and varying from 6 to 12 feet wide. At the collar of the winze there is a very good bunch of galena which carried down nearly 2 feet wide of a pay streak, following the hanging wall, for a distance of 30 feet along the winze, and from 18 inches to 24 inches of mixed zinc blende and gangne resting on the foot wall.

The vein here shows a banded structure with a large development of calcite as vein filling, together with siderite and schist; the hanging wall is very smooth and regular, next to it a 6 inch clay gouge occurs, the whole formation presenting the general appearance of a bedded deposit. A drift has been advanced about 40 feet northerly from the bottom of the winze and in places this shows quite a mass of porphyry and zinc blende fragments, encrusted with siderite and zinc blende in alternating layers. The galena in the vein rather favors a calcite gangue, makes near the hanging wall gouge, and invariably contains a sprinkling of chalcopyrite. Plate III is a good illustration of crustification and the order of mineral deposition in the Highlander vein.

(P) is a porphyry fragment cubical in shape, covered with a film of siderite; next an envelope of zinc blende ¼ inch in thickness surrounds the cube-like piece of porphyry, following which is a coating of siderite ⅛ inch thick, which is in turn surrounded by a thin film of zinc blende forming the outer layer, coincident with that of the fragment above it.

Immediately above the porphyry fragment marked (P) a similar one occurs with its longer axis at right angles to that of (P). On this upper fragment the same order of deposition can be seen, but not in such regularity; the zinc blende is merely a film around the bottom and sides of the fragment, while on top it is ½ inch thick; the siderite layer following the zinc is quite regular on the sides and top, while the outer film of zinc blende roughly traces a figure 8 around both fragments.

To the left of (P) occurs a solid piece of zinc blende enveloped in siderite and outside that, in places, zinc blende, but not in regular envelope. To the right of (P) another porphyry fragment can be seen surrounded by blende and siderite in the manner previously described. In the lower part of the specimen, which corresponds with one of the vein walls, the zinc blende is in streaks parallel with the wall, and contains some pyrites and galena associated with quartz. The white ground-mass is silicious siderite.

The general order of mineral deposition around the porphyry breccia is first a film of siderite; then zinc blende, third, siderite and fourth zinc blende as a very thin outer coating of the fragments and beyond that the siderite ground-mass cements everything solidly.

Thirty feet south of the winze on the main level, a raise has been put up about 200 feet, which, for the first 60 feet shows a very fair vein with a pay streak, varying from 18 inches to 24 inches of blende and galena, the siderite and calcite gangue being very similar to that seen in the winze below, but the galena is less plentiful. Sample No. 52 was taken across this vein in two cuts, for a width of 18 inches, about 30 feet above the level. It assayed: Silver 1.9 ozs., lead nil., zinc 13.9%. The main level extends on the vein 300 feet southerly from the winze, for the last 200 feet of which the lode becomes hard and unproductive, the face of the drift showing a porphyry structure, and it looks as if at this place a gray porphyry dyke occupied part of the vein.

Two cross-cuts have been driven into the hanging wall of the vein between the main cross-cut tunnel and the face of the southern drift, without showing up any mineralization, but the foot wall portion, as opened on this level for a distance of 350 feet, shows an extremely well mineralized deposit with a pay streak that would average about 2 feet wide of a composition similar to that indicated in the samples above referred to, both of which were taken from two cuts across the vein. The best showing is in the bottom of the level but owing to the wet nature of the country and the prevalence of fissures in this rather open vein structure, mining below the level will, no doubt, be hazardous and expensive. A tunnel can be brought in 320 feet deeper, but it will necessitate driving about 2,800 feet and would not be warranted without considerable further development on the vein and the opening up of large quantities of pay ores.

The water is pumped from the winze and the ore hoisted by compressed air delivered from the Taylor Air Compressing Plant situated on Coffee Creek about two miles south of the mine. There is no opening to surface except the main tunnel; the only ventilation in the mine is that supplied by the compressed air, which is insufficient, as carbonic acid is very prevalent, and in many of the workings a candle will scarcely burn.

Although there are 900 feet of backs above the tunnel level available for stoping no prospect raises have been put up, except the one described. One or more raises should be pushed through to surface, as in such a strong vein there is every probability of good shoots of pay ore being found above the tunnel level.

It is interesting to note a comparatively flat deposit, conforming to the planes of schistosity of the enclosing rock, which continues so regular and strong at a depth of nearly 1,000 feet from surface, and under such a mass of superincumbent rock.

The appearance of the Highlander vein in places would suggest that a small stratum of limestone had been in part replaced by ore, and the solution and re-crystallization of the limestone had resulted in considerable settling of the hanging wall and general rock movement adjacent to the vein. There is also evidence that in various places a porphyry dyke occurs within the vein.

The Highlander property has the appearance of having had far too much money squandered on the main tunnel, in search of elusive veins at great depth, to the neglect of the large vein, which is really a first class prospect and deserving of thorough development.

The Highlander tunnel is connected by wire tram with a concentration mill on the shore of the Kootenay Lake. The tram serves to transport both the milling and hand-sorted ore, and the latter, together with the concentrates is shipped in barges from the mill. This mill was not examined as it has not been operated for some time. The management was, at the time of my visit, figuring on shipping the crude ore in bulk to the smelter, after a preparatory hand sorting, and in this way securing the benefit in a smelter rate from the carbonate of iron and lime that the ore contains, and saving the loss incident to concentration. Though the freight and smelting charge must be necessarily higher than the combined rate on concentrates, yet, the management considers the bulk shipments more profitable. Considering the low tenor of the ore and its intimate association with the gangue, the experiment does not appeal very favourably to me.

TARIFF MINE.

The so-called Tariff vein, where intersected in the Highlander tunnel, is simply a clay gouge with an inclination varying from 10° to 30° westerly. The vein, as followed in the south drift, varies from merely a clay parting up to a couple of feet in thickness of quartz and calcite. Near the end of the drift a cross-cut was pushed into the foot wall 80 feet, and a drift was advanced 20 feet southerly on a small veinlet, showing a sprinkling of blende and galena near the face. Generally speaking one would scarcely recognize anything in this drift as an important mineral vein. It is practically an irregular series of small quartz lenses conforming to the planes of schistosity, containing in places some calcite and a little clay gouge. Practically, there is no mineralization in the vein at this depth, and as no connection has been made with the Tariff workings at surface there is nothing to show that it is the same vein, though the probabilities are strongly in favor of its being so.

The main workings on the Tariff vein consist of an incline-shaft, 175 feet vertically above the Highlander tunnel (aneroid reading). Passing down this incline everything is stoped out from the first level to surface.

On the second level the vein has a strike of north 20° west and the dip from surface to this level is 25 degrees westerly. The level extends for about 250 feet, and the ground has been fairly well stoped out on either side of the shaft. The vein is very regular in strike and dip and is stoped to a width of about 3 feet, the workings in places reaching 5 feet in width. The vein occurs in mica schist and is comformable with the planes of schistosity; the hanging wall is extremely smooth and regular.

On the third level a cross-cut east, behind the shaft, passes through the foot wall and opens up a parallel vein, 50 feet distant from the main vein, on which about 250 feet of drifting was done. I could not detect any particular values in the north drift, but the southern one showed some zinc near the face.

Passing south from the incline-shaft on this level I found the stoping had not been carried below the floor. The vein there is practically all quartz, showing a little disseminated pyrites. In the southern face of the stopes it consists of 2 feet of quartz on the foot, and close to the hanging wall a 1 inch seam of fine grained blende carrying galena in fine specks.

The face of the drift 10 feet further south shows quartz and schist only, for a width of 2 feet, and is barren of pay mineral. In the north end of this stope, 30 feet north from the face of the drift, the vein shows for a width of 30 inches. Next the hanging wall there is 1 inch of blende, then 2 inch of quartz, followed by 9 inch of fairly solid blende (containing, however, some quartz breccia), and below that 18 inch of quartz and schist with a little blende distributed through the joints and cleavage plains. This ore, however, does not continue for any considerable distance along the level, in as much as at a point 10 feet farther north the vein is apparently filled with quartz for a width of 3 feet and shows no appreciable mineralization.

Descending to the next level, which extends north only, or in the opposite direction to the last, considerable stoping has been done, and in one place a winze has been put down and a small stope opened below the level. Practically, however, this level forms the bottom of the northern stopes, just as the level above forms the bottom of the southern ones. This north level for the greater part of the distance consists of quartz and calcite, the former predominating. I could detect but little mineralization at any point. At this level a short drift extends southerly, showing a vein about 2½ feet wide, mostly quartz but not containing any pay minerals. A winze was started in the floor of the level and sunk 5 feet, showing a fair vein of quartz and calcite, but no pay minerals. The shaft extends at least 50 feet below this level at which point water was encountered. The level previously described has been recently under water and was smeared with mud and very difficult to examine. There was no one present to represent the owner of this property, which has been abandoned for several years, and the workings were scarcely safe, but having been informed there was considerable zinc showing in the old workings, I ventured to make the examination without a guide.

Judging from the stopes, the vein so far as developed, contained one good shoot of galena together with a little blende, near the surface; which has been to all intents and purposes stoped out to the extreme boundaries of the shoot, and in depth, so far as any pay ore could be followed. Other shoots may, however, be found by development on the strike of the vein to the south.

It it evident that the ore shoot splits in depth in two prongs, and the ore becomes more silicious; quartz predominates and little, if any, galena or blende can be noticed in the lowest workings. This fact, taken in connection with the showing made on the drift from the Highlander tunnel augurs badly for a profitable exploration of this vein in depth. The easterly vein reached by a cross-cut from the incline-shaft has the best showing at that depth, and the few traces of ore found in the workings from the Highlander tunnel also occur in this easterly vein, at least the probabilities are such; for, as previously stated, neither vein has been connected between this tunnel horizon and the old workings above.

BLUE BELL AND KOOTENAY CHIEF MINING CLAIMS.

On the east shore of Kootenay Lake, opposite the town of Ainsworth, important ore deposits occur on what might be described as a small peninsula, 5,400 feet in length by about 1,000 feet in width. This peninsula rises to a height of from 150 to 200 feet above high water level of the lake, and is partially separated from the main land by a shallow valley terminating in two small bays known as North Bay and Galena Bay. The mineral outcrops, and their supposed continuations are covered by the Arcade, Comfort, Blue Bell and Kootenay Chief mining claims, the last covering the extreme southern point of the peninsula, and extending 1,000 feet up the west shore of Galena Bay.

KOOTENAY CHIEF.—On the southern point of the Kootenay Chief (Plate VI) there is considerable exposure of mineral in marbleized limestone, exhibiting a well pronounced shearing structure, in the larger and more open zones of which mineral solutions have circulated and replaced certain strata of limestone, presumably by metasomatic process of ore deposition. This mineralized zone may possibly extend northerly for a distance of 800 feet, the shearing taking an east and west (magnetic) course, crossing the point of the peninsula. The ore exposed in the limestone around the point of the peninsula is mostly pyrites and pyrrhotite, with a little galena, and in places a considerable showing of blende. The mineral in the larger shear zones, or east and west fissures, appears for the most part thoroughly oxidized and stands out in marked contrast to the horizontal deposits in the limestone; it is, in fact, an admirable place to study the genetic relation of the ore deposits to the sheeted fissures. The limestone is here covered with hard micaceous schist; the upper strata of the limestone also show considerable muscovite in the bedding planes, and in a few instances it can be noticed in the joint structure of the limestone rock. This limestone, as previously noted, has been throughly marbleized, and though it contains numerous streaks of greyish slaty matter, yet large beds occur of white crystalline marble. I could not determine the thickness of this limestone formation, but I have no doubt the ore deposits will continue to the bottom of the limestone, and some of the larger fissures will probably extend, carrying pay ore, to that depth at least.

On the shore of Galena Bay, at a place about 100 feet north from the extreme point of the peninsula, I took sample No. 54 from a bed of ore 6 feet thick, making two cuts for the entire height. This sample showed considerable iron pyrites, some pyrrhotite a little lead and considerable blende. A little siderite occurs in the bed with the other minerals, and some actinolite. The sample assayed: silver 1.3 ozs., lead 4.9%, zinc 27.5%. About opposite to this sample and around the point on the lake shore, I took sample No. 53 from a bed consisting almost entirely of pyrites and pyrrhotite, though showing a little blende and galena. It assayed: silver 0.8 ozs., lead 2.4%, zinc 5.9%, for a thickness of seven feet. Stratigraphically this bed is about 20 feet above the one sampled by No. 54. It, however, contained a considerable development of actinolite in the ore deposit. The east and west fissures in the sheeted zone stand almost vertical, and could in places be traced along the shore and into the cliff. They contain oxidized ores, and occasionally developments of calcite crystals.

Very little work has been done here, simply a few shots in one or two places, the whole mass being no doubt too low grade to pay for working in the past. It also looks as if a better mineral would be found at greater depth, but owing to the close proximity of the lake and Galena Bay, and the fissures striking east and west from river to bay, the water question will undoubtedly prove troublesome in mining below the water level.

BLUE BELL.—The principal deposit, so far developed on the peninsula, occurs on the Blue Bell claim adjoining the Kootenay Chief to the north, and at a point about the centre of the peninsula where two large open cuts have been excavated in another sheeted zone, cutting through limestone formation and consisting of four principal and several minor fissures converging slightly on their western strike as more clearly shown from the workings in the main tunnel (Plate IV). It would appear that the main fissures will converge at a point on their strike 500 feet from the place where the main tunnel intersects the western fissure. The open cuts on the surface expose the formation for a width of 150 feet in one cut and 75 feet in another, separated from the first open cut by a band of limestone 50 feet wide. The limestone in the main open cut shows in the face 60 feet high, replaced in part by pyrites, pyrrhotite, blende and galena, while immediately below the floor of this open cut an immense gallery has been excavated to an average width of 80 feet, a length of 200 feet and height of 25 feet.

At the time this excavation was made, blende was of no value and the stoping was conducted in search of galena. The property has been idle for almost 10 years, but has recently been taken up by the Canadian Metal Company of Frank, Alberta, and the workings, at the time of my visit, were being cleaned out preparatory to resuming operations.

The property was last worked through a main tunnel having its portal in a small bay at an elevation of 25 feet above the low water mark of Kootenay Lake. A substantial wharf is erected here, together with ore bins to facilitate shipping the ore to Pilot Bay, where the former company established a concentration and smelting works (Plate XII). Just why it went to the expense of loading the crude ore on barges and towing them down to Pilot Bay, a distance of 8 miles, and then unloading the ore again before concentrating, it is difficult to imagine. The concentrator should by all means have been erected at the tunnel mouth where a good water power might easily have been developed from the creek north of the mine, and the ore reduced and concentrated without further expense in moving waste rock.

A good air-compressor, operated by steam, is erected at the tunnel mouth, together with a blacksmith's shop, and the usual mining buildings, all in a fair state of preservation (Plate X).

A map of the property made in 1892, shows the strike of the Blue Bell vein to be practically north and south, whereas the strike of the sheeted zone and mineralization is almost east and west (magnetic), or at right angles to that shown on the map as the "strike of the vein or deposit." Proceeding on these lines the main tunnel was doubtless advanced on a course that intersected a main fissure 560 feet from the portal. Had it been advanced in a more northerly direction, say on the line A.B. of Plate IV, the principal ore body would have been intersected in about the center of the proved mineralization. The sheeted zone as proved by this main tunnel is 120 feet in width and there should be 80 to 100 feet more of this sheeting to the north, as will be observed from the section, Plate V.

The tunnel gives the following section: 20 feet of wash, 103 feet of mica schist, then a bed of white marble which has been opened up somewhat and in places shows a development of large calcite crystals. The strike of the beds as shown by this opening is north and south (magnetic), with a dip of 35 degrees to the west. The vein of marble is 10 feet thick and its foot wall on the tunnel is also mica schist, but in a few feet a basic dike is encountered 15 feet in width. The tunnel next penetrates 50 feet of mica schist, then 28 feet of quartzite, followed by 50 feet of schist. When the main limestone series is met with, in which the ore occurs, and at a point 360 feet from the tunnel's portal, drifts extend easterly and westerly in this ore zone, and considerable stoping has been done for a distance of 60 feet west of the main tunnel. The ore exposed is mostly heavy iron sulphide, showing in places a little galena, together with some blende. The west face is apparently nearing the schist, while the sides of the drift or stope, 18 feet wide, is in white marble with quartz stringers. This stope was probably worked for lead, and doubtless with unsatisfactory results. This mineral zone has a width of 60 feet.

As previously stated, the main sheeted zone was intersected at 560 feet from the portal, at which point the tunnel was turned to a magnetic north course, intersecting four or five of the fissures and extending west on the first main fissure 160 feet, at which point the stope was abandoned in very good ore. Sample No. 55 was taken on the south side of the breast, 8 feet in thickness, assaying: 4.8 ozs. silver, 23.4% lead, and 19.5% zinc. No. 56 was taken in the center of the breast extending from the roof to a depth of 6 feet. The ore I believe extends to the floor of the stope, a distance of 10 feet from the roof, but could not be sampled on account of an accumulation of broken rock in the face. This sample assayed: silver 0.5 ozs., lead 0.4%, zinc 31.9%. No. 57 was taken on the north side of the stope beside the main fissure. It assayed: silver 2.1 ozs. lead 11.3%, zinc 10.9%, for a height of 8 feet. All these samples show very good blende and rather pure galena, together with some iron pyrites and pyrrhotite, in a crystalline limestone gangue; some quartz containing chalcopyrite is in places developed adjacent to the fissures. The face of this stope is 22 feet wide, and will average, for a height of 8 feet, 2.5 ozs. silver per ton, 11.7% lead, and 20.8% zinc, a remarkably fine showing. Fifty feet east of the face a stope over the level communicated with the floor of the grand gallery above. Sample No. 59 was taken in this stope 6 feet below the floor of the gallery, for a height of 3 feet; it gave the following assay: silver 2.1 ozs., lead 15.7%, zinc 21.1%. Passing up into the grand gallery, sample No. 58 was taken near the face of the old workings immediately above the stope on the level below, on which Nos. 55-57 were taken. This sample was 4 feet thick, of high grade zinc ore, assaying: silver 1.3 ozs., lead 2%, and zinc 34.5%. It was, however, from a stratum about 15 feet above the one on which the lower stope was operated. Sample No. 58 was from the south wall of the grand gallery, opposite a cross-cut, extending 30 feet into the north wall of these workings. The sheeted zone exposed here, for a width of 110 feet, shows six almost parallel fissures.

Returning to the main tunnel level, the stope where No. 56 sample was taken, will (when advanced 40 feet westerly) break into the largest stope on this level, which has been opened on a main central fissure and followed for a distance of 240 feet from the point where the tunnel intersects the first main fissure. Here the ore has dipped below the floor of the stope and the overlying muscovite schist must be close at hand. The western face of the stope for its full width is practically barren of ore, but contains a little quartz in the massive beds of white marble. There are, however, a few patches of pyrites and pyrrhotite in the marble and several grey slaty streaks following the assumed bedding planes. Along the north side of this stope a little ore is seen, mostly, however, pyrites. This probably results from the next northern fissure, which passes through the shaft and has nor been explored beyond that point. The strike, however, will bring it neat the present northern face of the big stope. The prospect of finding ore by drifting north-east from the big stope is therefore somewhat encouraging. The shaft, I am informed, is about 36 feet below the tunnel level, and a drift extends southerly 60 feet, with what results I could not determine as the shaft was full of water.

Reference to the photographs (Plates Nos. VII, VIII, IX, and XI) will show the general appearance of the open cuts, and in part, the structure of this immense deposit much clearer than any written description can portray it.

The Blue Bell ore deposit is, in my opinion, genetically related to the zone of fissuring extending in the crystalline limestone across the peninsula. Through the fissures in this sheeted zone mineral-bearing solutions circulated and attacked certain of the favourable beds or bands of limestone, depositing metasomatically the great ore beds and irregular masses of mixed sulphides. Sometimes they completely replaced a given bed for considerable distances, but more often with the irregularity of such deposits, skipped certain portions of an apparently favourable bed, and then again formed another deposit in this same bed, further along the strike of the fissure. A certain stratum for example, may at one point be replaced by ore for a thickness of 7 feet, extending across the full width of the sheeted zone, while 50 feet away the same stratum may only carry ore adjacent to the main fissures; or it may be entirely barren. Usually, however, certain favourable strata contain irregular ore deposits adjacent to the fissures so far as development has followed their western strike; therefore it is reasonable to assume that if the converging fissures continue westerly and intersect, large ore bodies should be found in that extensively sheared and fractured zone.

The key of this deposit is apparently the fissures. There is good reason to expect pay ore in the crystalline limestone anywhere along their strike and in depth to the bottom of the limestone, in such places as favourable beds have been subjected to the mineralizing influence of circulating solutions. It is difficult to form an opinion as to the condition incident to a change of formation in depth, say from limestone into schist. However, as there is no appreciable faulting as a result of the fissuring, I rather incline to the opinion that the fissuring will disappear at a moderate depth in the underlying schist, though some one main ore bearing fissure may be found to continue to considerable depth in the underlying formation.

About the only development work carried out by the former owners of the Blue Bell—outside the main tunnel—was by stoping; no effort was made to show up either the length, width or depth of the ore deposit, though the conditions for proving up its extent by diamond drilling are ideal. The new company had not commenced mining work at the time of my examination; hence any attempt to estimate closely the tonnage or even the value of this very large ore deposit, is simply an impossibility.

During the former working of the Blue Bell property for lead, the zinc and pyritic ores were avoided; the new company is, however, primarily in the field for zinc ore and should mine the entire deposit up to the limits of pay ore. On such a basis there is undoubtedly ore enough above the level of the lake to keep a 200-ton concentrating mill well supplied for several years. Considering the great size of the stopes and chambers now existing, the great width of the mineralized zone, and the depth to which the ore extends, the economic mining of this ore, in mass, presents an interesting problem, the proper solution of which is a matter of the very first importance, to the end that the best system to meet the peculiar conditions shall be adopted.

KRAO MINE.

This property lies at an elevation of about 1,260 feet above Ainsworth, with which it is connected by a good mountain wagon road, about 3½ miles in length. The property, consisting of one mining claim (20.66 acres), is located on a creek following a depression along a comparatively flat step, which joins the steep escarpment rising about 1,100 feet above Lake Kootenay. Along this bench a thick seam of white, crystalline limestone (marble) outcrops and can be traced for a couple of miles along the mountain side, chiefly to the north of this property, which I shall for distinction call the Krao limestone. A continuous line of claims is staked along this limestone outcrop for probably a mile and a half.

At the Krao mine the limestone exhibits a sheeted structure and there is probably a fault running parallel with the creek at this point, but no evidence of it could be detected other than the continuous sharp depression, as previously noted. In the sheeted zone at the Krao the limestone is impregnated with galena and some blende, siderite often filling the fissures, while the circulating waters have dissolved irregular pipe like cavities in the limestone, which have later become lined with mineral, chiefly galena, next to the limestone, with an outer coating of blende and siderite. The deposit has been stripped at surface for a width of about 60 feet and a length of 150 feet, and 6,858 tons averaging 22 ozs. silver, 12% lead, and 8% zinc had been quarried out, hand sorted and shipped to the smelters prior to October 31, 1905. The ore apparently runs fairly well in silver, and wire silver is a common occurrence in the cavities and joint planes within the mineralized area.

Three men were at work shooting out the ledge for a width of about 30 feet, sorting it over and sacking the sorted ore for shipment. The open cut had reached a depth of about 12 feet. The fissure near the hanging wall appears to be the larger of the group, and contains considerable iron oxide resulting from the decomposition of the originally contained ores, in fact all the sheeting planes near the surface, and many of the vugs and pipes, are filled with this oxidized material in which rich silver ore and wire silver are very often found.

A shaft was sunk on the hanging near the main fissure to a depth, I am informed, exceeding 100 feet, but at the time of my visit the water level was reached at 35 feet at which point a cross-cut extended about 10 feet into the sheeted zone, exposing a mass of white crystalline limestone penetrated by numerous irregular tubes and vugs, lined with galena and siderite (Plate XIII, Fig. 2). The galena has also penetrated the crystalline limestone for some distance from those tubes and vugs and from the fissures in its mass (Plate XIII, Fig. 1). The vein was exposed at this depth for a length of about 30 feet, extending for a short distance on either side of the shaft. The northern face exhibited a very rich appearance owing to the good showing of galena, while the south side was much poorer in galena and more quartzose. The shearing has a strike of north 45° west and a westerly dip of about 80 degrees. I should judge from the hardness of the limestone that it was somewhat silicified by the ore bearing solutions which circulated within it.

This property presents, on a small scale, the same phenomena as the Blue Bell property on the east side of Kootenay Lake. Very little development has been carried out, and one can only class it to-day as a very promising prospect, in which development in depth may prove up good payable ore deposits.

STAR VEIN.

Proceeding northerly along the Krao limestone for a distance of about 2 miles and past several prospects, the Star vein is reached at an elevation of 1,800 feet above Ainsworth. The Krao limestone outcrops boldly at this place with a strike of north 30° west, and a westerly dip of 30 degrees. Two short inclines have followed down a mineralized stratum for some distance and a center incline is evidently communicated with a tunnel run in from the bluff below. In the shaft-house at this incline some galena and lead carbonates were sacked, showing that some values had been obtained in the course of development. There was very little zinc showing in any of the ores at the shaft-house, and the workings were not in a safe condition for examination. I was informed that a new tunnel had been started last year to get the vein at considerable depth, but work had been abandoned during the winter.

SPOKANE.

Coming down from the Star claim to Ainsworth over the old Bridle trail, the Spokane vein was crossed, bearing north 50° west, with a westerly dip of 40 degrees, showing an extremely regular vein about 2 feet in width, consisting of 6 inches of gouge on the hanging wall, followed by 4 inches of galena, under which was 10 inches of quartz and calcite. This vein has mica schist hanging and foot walls. A lead streak crops out with considerable regularity for 100 feet along the hillside where the vein had been stripped, almost as regular as a line (Plate XIII, Fig. 3). The prospect is at an elevation of 940 feet above Ainsworth, measured at the one incline-shaft on the property, depth of which could not be ascertained.

OLD JEFF.

Descending along the Bridle trail the Old Jeff location was visited, situated at an elevation of 500 feet above Ainsworth. Two short inclines have slightly opened this vein, which appears to be about 2 feet wide, enclosed in mica schist and showing a little galena and blende intimately mixed with silica, as the vein is largely composed of quartz. The development is insignificant, and further work is necessary to show up pay ore, provided it occurs in the vein in the locality of the present prospect holes.

GLENGARRY.

This property joins the Krao to the south. It has a fissure vein, which bears east and west at surface, with a southerly dip of 70 degrees, crossing the schist formation. The workings were full of water at the time of my visit, but the late owner of the property informed me that the shaft was 65 feet deep, and that a little drifting had been done at the bottom. The ore from the drift he said ran from 8 to 10 ozs. in silver and 44% zinc.

A small open cut at the surface showed a vein about 3 feet wide, containing some very fair looking blende associated with quartz and decomposition products.

This property has recently been taken in hand by the Canadian Metal Company of Frank, Alberta, which was erecting a hoist and gallows frame preparatory to pumping out the water and resuming work. The machinery will be operated by compressed air from the Taylor Compressing Plant, situated on Coffee Creek one and one-half miles distant from the mine. The same air line is also being extended to the United mines about 1,600 feet northwesterly from the Glengarry.

UNITED MINES.

This property is situated a little further up the hill in a northwesterly direction from the Glengarry, the collar of the shaft being about 1,350 feet above the lake, and connected with the wharf at Ainsworth by a very good mountain wagon-road. The vein is exposed in two places at surface, and contains considerable galena together with some blende. The shaft, which is full of water, is said to be 170 feet deep, with levels extending east and west in the vein at a depth of 50 feet, and for a distance of 80 feet on either side of the shaft. From the shaft and this drifting (it is claimed that the vein has not been stoped at any place) it is said that 500 tons of ore was shipped to Revelstoke by the former operators, who were also interested in a small smelter erected at that place. This is also a fissure vein with an east and west course, and a southerly dip of 60 degrees, and is said to average 12 feet wide at the 50 foot level; it appears to be half that width at the surface.

About 130 feet west of the shaft a ravine courses along the foot of a steep bluff and may possibly follow a line of faulting, although no evidence of actual faulting could be detected. Considerable trenching on the west side of this creek has failed to discover the vein, so that it is evident, if it is not faulted, it terminates about 130 feet west of the shaft. The vein has not been traced by open cut or otherwise for any great distance easterly from the shaft, and like the Glengarry, appears to be a short fissure crossing the strike of the formation, and probably not extending to any great depth. Both these veins are in a formation occurring between the crystalline Archaean and the Slocan slates.

There is about 50 tons of zinc ore lying on the surface at the United shaft from which a heap of very good lead ore has been hand sorted. The property is being re-opened (after several years of idleness) by the Canadian Metal Company, which is erecting a hoisting plant to be operated by compressed air from the Coffee Creek plant.

Cooking and bunk-houses for the men are also under construction.

GENERAL CONCLUSIONS.

Assuming, for the purpose of an estimate, that the Blue Bell ore if mined *en masse*,—rejecting only the barren lime rock in the stopes,—would average 15% zinc, or 300 lbs. per ton, that the net recovery would be 65%, or 195 lbs. of zinc per ton of ore milled, then the daily treatment of 200 tons of crude ore would give 39 tons of 50% zinc concentrate.

A net recovery of only 65% of the total zinc in the ore may be considered low, but allowing for zinc associated with the galena, or retained in the lead concentrates, the zinc lost in the mill tailings, and, lastly, the loss in the magnetic separation of the zinc from iron pyrites and pyrrhotite in the zinc-iron concentrates, the result will not appear much out of the way. It practically amounts to a saving of 77% of the zinc in the zinc-iron concentrates and a loss of 15% of this zinc in the magnetic treatment, which, everything considered, is about as good as could be reasonably expected from this class of ore. I should note in passing that the galena concentrate recovered ought to go a long way towards paying for both the mining and milling of the ore.

The other mines examined in Ainsworth Camp could produce from present developments 15 tons of 50% zinc concentrate per day, or its equivalent in lower grade ore, if a satisfactory price were obtainable. With further developments the output would no doubt steadily increase, probably reaching a production of 100 tons of 45—50% zinc concentrate per day in the course of a year or so.

This estimate, it will be observed, is based on mining and milling the low grade ores now exposed in the various mines. Any selection of ore, or new strikes of higher or lower grade will necessarily affect the results. The Blue Bell alone could by selecting the milling ore produce for some time 60 tons of 45—50% zinc concentrates per day; whereas my estimate is based on fairly mining the entire workable deposit.

THE SLOCAN.

Geological.—The Slocan series of rocks are classed by the Geological Survey of Canada as upper or middle Palaeozoic (Carboniferous in part). "The series consist of dark carbonaceous argillites, dark limestones (often impure), quartzites and greywackes, together with beds of tuff and ash. Only an obscure fossil has been found in these rocks, which are in all likelihood upper Palaeozoic, probably Carboniferous. They are mostly thin bedded and fissile, and where the conditions were favourable seem to have been particularly susceptible to mineralization, forming the country rock of a large part of the highly productive silver-lead mines of the Slocan district."*

The Slocan rock series as exposed in the mines examined, consist essentially of black, fissile, carbonaceous slates, intercalated with which are beds of limestone and quartzite. The limestones although usually dark and slaty, passing over into calcareous slates in many places, are sometimes crystalline and of exceptional purity. In the Cork mine, for example, a very fair light coloured marble occurs, while in the Lucky Jim a grey coloured semi-crystalline limestone contains 53% of CaO, less than 1% silica and only a trace of MgO.

The slates are penetrated in almost every direction by porphyry dikes, closely related to the younger granites, which form massive intrusions, sheets and dikes throughout the argentiferous galena producing area of the Slocan.

The silver-lead-zinc deposits (hereinafter called Slocan veins) are almost exclusively confined to fissure veins formed in the slates, subsequent to the porphyry dikes and the granitic intrusions. They are divisible into two series, the dry ore or silicious veins, adjacent to and sometimes penetrating the granites, and the lead ore veins in the slates.

The vein fissures are closely associated with the dikes and intrusions, some of which they follow for considerable distance, but more often cut across the dikes on their strike. One of the largest Slocan veins, the Star and Silversmith, passes around a very large intrusion or stock, forming a rough semi-circle through the slates in doing so.

* British Columbia, West Kootenay Sheet, by R. G. McConnell and R. W. Brock.

Subsequent to the formation of the fissure veins, a very complicated and widely extended faulting occurred, displacing both the dikes and the veins, while in the softer slates, recent slight crushing and settling has resulted in incipient faulting that is even now in progress.

Another class of deposit in the Slocan series is a replacement of limestone beds with zinc blende and pyrites, in chimney-like ore shoots associated with fissures, barren in the enclosing slates and probably more recent than the Slocan veins. This class of deposit is best represented by the Lucky Jim ore shoots.

The Slocan veins are primarily deposits of argentiferous galena of very high grade; the average of nearly 8,000 tons shipped during 1905 is: silver 90 ozs. per ton, lead 40%, and zinc 9.8%. Blende, siderite and pyrites are the common associated minerals. Tetrahedrite (Freibergite) and chalcopyrite occur in small quantities as primary ore associated with the galena, and occasionally with the blende; in the latter case the blende contains high silver values.

There is no well marked or definite order of mineral succession in the Slocan veins. Galena occurs in the outcrops of nearly all the pay veins and may or may not be associated with blende and siderite. Galena also occurs in some quantity to the greatest depths I have seen the veins exposed. Taking the relative abundance of the minerals, however, I find as the result of my observations, that galena favours the hanging wall of the veins and is more abundant in say the first 300 feet than at greater depths, zinc blende in the compact veins is very often found next the galena, while siderite and quartz usually form the foot wall portion of the veins. And so in depth, blende becomes more plentiful as the galena becomes less; at still greater depths siderite and quartz fill the veins almost to the exclusion of both the galena and the blende; while at the greatest depths quartz predominates. Calcite is a common mineral in the veins, and in some cases forms the exclusive gangue accompanying the ore. The Wakefield mine is a good example.

In the dry ore veins, silver ores are associated with galena, some copper mineral and zinc blende, with perhaps a little siderite, in a highly quartzose gangue. These pass in depth through the stage of zinc and quartz into almost straight quartz veins, or a breccia of quartz and slate, and thus both class of veins at the greatest depths are of practically the same composition.

SOUTH FORK OF THE KASLO RIVER.

This district is reached by the Kaslo & Slocan Railway to South Fork station, about five miles from Kaslo; thence an excellent wagon road connects with the Cork mine, a distance of five miles, in which the road rises only about 1,000 feet, following the South Fork of the Kaslo through a densely wooded country.

At four miles from the railway a mill was erected some years ago by the Montezuma company, and a wire tram probably connects direct from the mill to the mine. The mill is now partially dismantled, not having been operated for about seven years. It was, however, fairly well arranged, operated by water power from the South Fork, and contained 8 jigs and two double deck revolving buddles. The tailings show considerable zinc, and some ore left in the bins, also contains zinc, though it was evident that a lead concentrate was aimed at.

Mr. Geigerich, who is now the principal owner of the property, gave me the following note: "The Montezuma claim was operated through a cross-cut tunnel from surface, 450 feet long. There are four levels on the property and all connected to surface. About 1,000 tons of lead concentrates were shipped, containing so much zinc that the operation of the mine on a lead basis was abandoned. The property finally went into liquidation with a debt of $75,000, and has not been operated since 1898. There are 6,000 to 7,000 tons of ore in the mine and on the dump."

The zinc ore showing on the Cork mining claim and on the Province was not, in my opinion, sufficiently important to warrant the examination of the Montezuma property, more especially as the owners had not made any preparation to clean up and prepare the mine so that the workings could be properly examined after their seven years abandonment.

CORK MINE.

This property is, in point of development, the most important now being operated on the South Fork. The vein has been tapped by three tunnels, and considerable lateral development conducted. A large mill has been erected, and 30 men are employed on the property.

Geology.—The mineral channel containing the Cork vein would appear to occur in the basal series of the Slocan slates. The geology, as seen in the No. 1, or lowest, cross-cut, consists of indurated dark semi-crystalline slates and schists in which are strata of crystalline limestone, the principal one occurring about midway between the portal of the tunnel and the vein, and the next in size, 110 feet east of the vein. The last, varying from 3 to 4 feet in thickness, is found near the face of the main tunnel, 200 feet easterly of the vein. The line of demarkation between the crystalline limestone and the slates is everywhere clear cut and distinct. Usually the slate adjoining the limestone is quite soft, and in places mere clay. There are other beds of fine grit included in the series, all of which have a general easterly dip of about 55 degrees, though the innermost bed of limestone, where intersected by No. 1 tunnel, dips easterly at about 75 degrees. The limestones, easterly of the vein, are very thinly bedded and contain dark bands of apparently clayey material. On examining this rock under the glass it presents a coarse aggregation of calcite crystals.

The vein, or veins, occur in a channel of faulted or disturbed slates from 30 to 50 feet in width. The ore makes along irregular slips, or incipient faults, inside the main walls of the faulted zone, which bears approximately north 40 degrees east, the vein proper dipping at 62 degrees easterly, or into the mountain.

Mill.—The No. 1, or lowest, tunnel is connected with a new mill by a level tramway, the cars being pushed out and dumped into the mill bin, the ore passing first through a 10 by 20 inch Blake crusher, following which it passes through a No. 4 Gates crusher, and without screening between the crushers, which is rather bad practice. The crushed ore is then spouted to a stock bin, whence a Challenge feeder delivers it to a set of 10 by 30 inch Allis-Chalmers rolls, running at 80 revolutions per minute. The crushed ore is then elevated to revolving screens which size it to 12, 8, 5 and 2 mm. the oversize returning to the first roll, and the middlings from the jigs to two other rolls of the same make and running at the same speed. The fines from the 2 mm. screens goes to two "V" boxes, the product of which is also jigged, making six jigged products. The overflow from the "V" boxes passes to a settling tank, the product from which is treated in three sizes on three Overstrom tables, and the overflow from the large settling tank passes to a second tank of similar construction, simply an elongated "V" box, where the entire product is treated as three different sizes on as many Frue vanners, but this feed is entirely too thin for any effective work, and it looks as if the amount of concentrate obtained by the vanners would not pay for their lubrication.

Both the lead and the zinc in the mill feed is very finely distributed through the ore, and is also of very fine grain, so much so, that the four compartment jigs give but one finished product and three middlings for regrinding.

I was informed by the management that the ore now being treated, (which, by the way, is first roughly sorted in the mine) is producing from 6 to 7 tons of concentrates per 100 tons of feed, the concentrates running from 50 to 55% lead and 30 to 33 ozs. of silver per ton. The average capacity of the milling plant is 3 tons per hour.

The machinery is operated by Pelton wheels, one situated on the crusher floor operating the crushers, screening machinery and the jigs, while a smaller wheel operates the tables and Frue vanners.

The water has a head of 700 feet and is conducted to the mill in an 8 inch pipe line, but the present flow (November, 1905) is insufficient to operate the milling plant alone, to say nothing of the compressor and sawmill; consequently, we find this new mill, which has not operated 30 days, short of power, and arrangements are under way to install a turbine wheel on the South Fork creek, where there appears an ample flow of water, practically at the foot of the mill. This new power will, I suppose, be used to operate the mill, while the 8 inch line will operate the sawmill and air compressor. An enlargement of the mill is also contemplated by an addition of one rough jig and two jigs to treat medium product. This addition is somewhat in the nature of an adjustment, some of the present jigs having too much feed and others too little. The mill is fairly well built and the arrangements are on the whole good. There is no doubt if sufficient galena exists in the mill feed, paying results will be obtained. No effort is made to save any of the zinc blende. It is of such a fine nature that the ore would have to be crushed very much finer, and more slime machinery introduced before any attempt could be made to save the blende. The examination of the present mine workings, however, shows that the Cork property does not at present come within the definition of a zinc mine.

Mine.—This mine is developed by three tunnels. No. 1, or the lowest, is connected with the concentration mill, as previously described. The vein is reached in this tunnel at a distance of 900 feet from its portal, the tunnel extending, as previously noted, 200 feet beyond the vein. At the point of intersection the vein is very poor, consisting almost entirely of crushed slate, with a little siderite and quartz on the joints and faces of the crushed rock.

On extending the drift northerly in this faulted zone, siderite became more plentiful, and at about 150 feet an 8 inch streak of mixed galena and siderite with a little zinc blende occurs and has been stoped in the roof for a distance of 70 feet. An examination of the stope shows that the lens-like mass of ore is cutting out 15 feet above the level, though another lens will doubtless come in higher up. The ore, however, is low grade. Sometimes it is found near the hanging wall of the faulted zone, as at the present face of the north drift, which is entirely in the hanging; at other times near the foot wall, or in diagonal seams crossing between these walls which are about 30 feet apart—but it is doubtful if the mineralization in these northern stopes, drifts and cross-cuts is sufficiently rich to pay for mining and milling under the conditions that now obtain. An examination of the No. 2 tunnel, 203 feet vertically above, shows a more regular vein, but it consists almost entirely of iron pyrites and siderite, with small bunches of zinc blende occurring at intervals, so intimately mixed with dense siderite as to be almost impossible to concentrate, as well as expensive to separate the mixed minerals after concentration.

Going south on the vein from No. 1 cross-cut tunnel, no particular mineralization can be detected until the raise is reached at a distance of 150 feet. This raise connects with No. 2 tunnel at a place where there is a good development of ore, but on No. 1 tunnel the vein at the raise is not of much value. Immediately to the south, however, a stope was being operated that had reached a height of 60 feet above the level, the vein being then 1 feet wide between regular walls, much striated, and showing deep scars and slickensides. Formerly the full width was stoped, but now an effort is being made to stope a width of five feet only on the foot wall. On the northern face of this stope, at the height indicated, there is a streak on the foot wall of 8 inches of quartz and galena, and on the hanging a little galena associated with blende, the central portion being massive siderite with a sprinkling of blende. At the south face of the stope at about 70 feet from the level, a cutting-out stope has advanced about 10 feet south of the regular line of stoping. The pay streak is there 4 feet wide of very solid blende (sample No. 87)., which assayed 6.7 ozs. silver, 13.7% lead, and 28.5% zinc. This was the only place in the mine where the occurrence of blende was sufficiently good to warrant sampling. This stope is only about 70 feet long, and is more in the nature of a chimney than the regular occurrence of ore in vein-like form. It has, as previously noted, good walls, but lacks continuity in length. If the deposit were stoped to the main hanging wall it would be about 30 feet in width by 70 feet in length. Considering the showing on No. 2 level above, I rather expect that some good bunches of both lead and zinc ore will be developed as the stoping is carried upwards. On the main drift, south of this stope, the vein again splits up, and nothing but a clay gouge exists in the face.

Tunnel No. 2 intersects the vein 70 feet northerly from the raise connecting with No. 1. The main drift extends northerly and southerly along the foot wall portion of the vein, and on the south extension shows but little mineralization until a point is reached 20 feet north of the raise when ore comes in which can be traced along the level for 100 feet, at which point it shears off into the foot and is exposed in a short cross-cut, but has not been followed. The vein there consists of massive siderite, with bunches of very good blende distributed irregularly through it, also considerable pyrites. The vein in this south drift might be considered a sort of cross vein, that is to say, running diagonally across a mineralized and faulted zone. The ore at the raise makes back into the crystalline limestone, which there forms the hanging country rock. At this place massive siderite and blende, with a little galena, occurs for a width of 20 feet and a length of 50 feet, the cross vein terminating in solid limestone, evidently a replacement of the lime by the mineralizing solutions (See Plate XIV.) South of the short cross-cut the pay ore gradually plays out in the main vein, and the face is entirely barren. This south drift is 300 feet in length and the short cross-cut indicated occurs at 150 feet. Going northerly from the main tunnel cross-cut, the vein is picked up at 50 feet, showing a little mineral, and continues in rather irregular bunches to the face, a distance of 200 feet. Siderite is, however, the predominant mineral. Hard white iron pyrites would come next in order of abundance, while blende occurs at irregular intervals; also a little galena in places. This portion of the vein, however, like that in the level immediately below it, is low grade. So far as present development shows, the only good ore in the mine is the short chimney-like shoot immediately south of the raise and extending from No. 1 level to possibly No. 3, or the uppermost level. There are, however, possibilities that if the drifts are pushed further north and south in the mineralized zone, at least another shoot or chimney may be met with. Considering, however, even in the best stope, the very scattered occurrences of mineral in this wide, crushed and faulted zone, it becomes clear that it must be worked on broad lines, at big capacity and with all modern improvements, in order to score complete success.

Tunnel No. 3 starts on the outcrop of the vein, which shows a considerable development of quartz and spathic iron with very little lead mineral. The drift advances southerly to a point over the main raise from No. 1, near which communication was made with No. 2 tunnel. The workings here were caved and inaccessible. The management indicated that they were also unimportant.

It will be noticed that the vein carries good ore where the limestone forms one of the walls, as on No. 2 level. If the main vein could be traced into one of the limestone beds good discoveries of pay ore might follow.

PROVINCE MINE.

The Province property is situated immediately to the north of the Cork, the vein occurring in the same mineralized zone. The geology and ore occurrences are practically identical for both mines, but the Province has a better shoot of zinc ore than anything yet developed in the Cork.

The Province mine is opened by two tunnels, the upper, or No. 1, being merely a prospect drift extending northerly and southerly on the vein for some distance beyond its exposure in the creek.

Tunnel No. 2, about 65 feet lower, enters the mineral zone at a distance of 120 feet from its portal, after penetrating the slates and schists of the district, but no limestone could be detected. The vein was reached at a distance of 170 feet from the portal of the tunnel in an unproductive place; a drift northerly of about 100 feet (which is now caved) did not, it is stated, discover anything of value. The southern drift followed a fairly regular wall until a point was reached 20 feet north of the winze, where a nice bunch of galena was met with close to the hanging wall, and underneath it some zinc blende associated with siderite—The winze is sunk 65 feet on this showing, and at the bottom a cross-cut reached the foot wall giving a distance of 25 feet between walls. The galena and zinc blende continues very regular in the winze, and the cross-cut exposes for 20 feet a vein of massive siderite, through which blende is distributed in irregular streaks and bunches. The vein for a width of 20 feet would average one-third blende, and in the ordinary milling, 4 tons of feed should give one ton of blende concentrate, exclusive of siderite. There is also some galena showing on the foot wall as well as on the hanging, but the latter is the best showing. The hanging wall drift extended northerly about 20 feet carrying galena for most of the distance; the face, however, is rather poor.

Returning to the main (No. 2) level, the ore lens on which this winze and other workings occur, shows to be only 40 feet in length, but at 20 feet south of the winze a cross branch drops in, and at the intersection with the main vein, a raise was put up 30 feet, and a drift advanced 6 feet in the vein. The face of this drift contains for 4 feet in width a very good exposure of zinc (sample No. 89, assaying: silver 15 ozs., lead 20.7%, zinc 23.6%); and zinc and lead can be seen in the ends of the raise down to the No. 2 level. About 35 feet west on the cross seam a good bunch of galena occurred and has been stoped in the bottom of the drift for a depth of 12 feet. The main drift then takes a northwesterly course, following a zinc and lead cross seam for a further distance of 40 feet. I believe this cross vein is merely a local phenomenon. I would not expect it to extend for any particular depth or height; it is simply a small fracture carrying ore across the main mineral channel. Sample No. 88 was taken in the face of this drift; it assayed: silver 3.6 ozs., lead 1.7%, zinc 27.8%. Sufficient work has not been carried out to show the exact nature of the principal ore occurrence. It looks, however, as if this chimney could be depended on to give from present development a stoping width of 10 feet, a length of 50 feet, and a vertical extension of, say, 100 feet, which would give approximately 4,000 tons of milling ore, from which considerable galena could be picked out, probably 200 tons. The remainder, however, would require concentration, and owing to the dense nature of the siderite, would give a low grade concentrate containing approximately equal parts of blende and siderite. Arrangements may possibly be made by which this ore could be stoped out and treated in some local mill. Far more extensive developments would have to be carried out before any concentration mill would be justified for the treatment of Province ores on the ground. The property, however, is a very promising one, and may, on development, open out into a big mine of medium grade ore. The diamond drill could be used effectively to prove up the deposits and attention should be given to the location of the vein system in the limestone beds, where there is every reason to expect larger and richer ore deposits.

WHITEWATER MINE.

This property, owned by the Whitewater Mines, Limited, consists of five claims and a fraction, aggregating 113 acres. It is situated within 1¼ miles of the Whitewater station of the Kaslo & Slocan Railway, over an excellent wagon road; the ore is delivered from the mine f.o.b. cars, for 75 cents per ton.

There is but one vein opened. It was discovered in 1892, and worked vigourously for silver-lead for several years, but for the last 3 years intermittently under lease. The first shipment of zinc ore was made in 1904, though the vein contains large quantities of zinc, as is well shown by the analysis of the lead concentrates; the zinc blende is for the most part rich in silver.

The Whitewater mine is developed by 7 tunnels. The vein varies in width from 8 feet down and may be said to average 5 feet, the pay streak about 8 inches. The mine has produced 12,548 tons of lead concentrates and 8,435 tons of hand sorted lead ore, which averaged 84.4 ozs. silver per ton, 33.9% lead, and 18.5% zinc. The average price received for this ore, covering a period of ten years was approximately $37.00 per ton.

The first zinc shipment of 38 tons assayed as follows:—

Silver	33.4 ozs.
Zinc	46.0%
Lead	4.5%
Iron	7.2%
Silica	8.0%

Net price received, $16.92 per ton.

The second lot amounted to 61 tons, assaying

Silver	17.6 ozs.
Zinc	43.2%
Lead	3.7%
Iron	8.0%
Silica	9.0%

Net price received, $8.37 per ton.

Mill.—A concentration mill was erected in 1898, having a capacity of 8 tons per hour and costing approximately $45,000. It is of the usual roll and jig variety, operated by water power and designed to produce a lead concentrate only. The operating cost when worked up to its capacity in 1900 was 36.33 cents per ton, the following year 33.4 cents, and in 1902, 31.59 cents per ton of ore milled. A sample of mixed zinc ore was taken from the bed of a third compartment jig, to show the composition of the zinc ore (No. 40); it assayed: silver 28.4 ozs,. lead 6.7%, and zinc 27.8%.

Mine.—The vein with an east and west strike and southerly dip of 35 degrees, has been followed from its outcrop above No. 1 tunnel to No. 7 tunnel and quite extensively stoped for a vertical depth of 500 feet. The mine, at the time of my visit was being operated by three sets of lessees, one of whom was working around the pillars and in the old stopes, the other two advancing No. 6 and No. 7 tunnels westward in the continuation of the ore shoot and with fairly good results.

The map of the workings shows a regular and rather a long ore shoot for the Slocan. The No. 2 tunnel, for example, is stoped for a distance of 400 feet, while No. 6 tunnel has been stoped for a length of 450 feet on the same ore shoot, and all the intervening ground between these levels is also stoped within the lines of the main ore shoot, which appears to average 450 feet in length. These old stopes are said to be crushed, caved and inaccessible between the first and sixth levels. The stope map indicates a second, or eastern, shoot extending from No. 4 to No. 7 levels, showing by the stoping a length of 250 feet on No. 6 and 100 feet on No. 7 level. Towards the end of No. 7 tunnel, which by the way is situated at an elevation of 4,150 feet above sea level, the drift follows the foot wall too closely, and the lessees have recently cross-cut into the hanging, proving the vein, according to Mr. A. C. Gardé, to be 8 feet wide carrying 3 feet of zinc and lead ore from which he took sample No. 39, which assayed: silver 57.4 ozs., lead 26.2%, zinc 29.6%.

From the best information obtainable it would appear that considerable zinc ore has been left in the old stopes, in the form of broken ore and pillars, much of which may be recovered by lessees. While in a moderate depth below the seventh level, the Whitewater vein will pass into the Whitewater Deep property, yet there is considerable ground to the west of the Whitewater Deep covering the strike of the vein, on which no attempt has been made to open up the main vein, though the prospect of finding pay ore in that neighbourhood is promising.

WHITEWATER DEEP MINE.

The Whitewater Deep, as the name would indicate, covers the Whitewater vein on its dip after passing outside the lines of the Whitewater Mines, Limited.

The Whitewater Deep Company has a very extensive establishment in the town of Whitewater, consisting of a palatial residence for the manager, sumptuous offices and well arranged assay office, together with a large and well furnished hotel, now in the hands of a caretaker, as are also the other buildings of the company at this place.

Mine.—The mining operations of this company consist of a main tunnel with some cross-cuts and three raises on the vein, from which a slight amount of drifting has been conducted. The main tunnel starts at an elevation of 3,779 feet, and is advanced 1,500 feet westerly from its portal. For the greater part of the distance the tunnel has advanced in faulted ground, broken and crushed slate, with only one small ore lens showing on the tunnel horizon. In the face of the tunnel the vein consists of 18 to 24 inches of siderite dipping south at 40 degrees. A little quartz is showing in the footwall, but no zinc or lead is visible. A short cross-cut at the tunnel face exposes what is probably the hanging wall, but the vein should be cross-cut thoroughly at this point, and the true hanging wall determined. On the east side of this cross-cut about 1 inch of mixed galena and fine grained blende occurs on top of the siderite. The vein in the face while barren is well defined and looks to me a good prospect, more particularly as by extending the drift it would pass under the points where ore is now being mined in Nos. 6 and 7 tunnels of the Whitewater mines, which is undoubtedly on the same vein. I examined two of the raises over the main Whitewater Deep tunnel; the western one showing a lens of siderite standing almost vertical in the vein, and containing a little galena. Passing the next raise to the east, I went up the second one and in a branch drift saw a fair vein of siderite with a little blende and lead, averaging 15 inches wide in the aggregate. Near the head of the raise a drift extended west about 50 feet and carried a fair vein, averaging 18 inches wide of siderite, with occasionally a seam of mixed lead and zinc ore from 2 inches to 6 inches in width. This also looks a good prospect, but the company having expended the bulk of its capital in the acquisition of the property, and the erection of buildings, the actual development of the property has been apparently lost sight of.

There is abundant evidence of crushing and folding, resulting in slight faulting along the greater part of the distance in the main tunnel. It is beautifully illustrated (Plate XIV, Fig. 2) in one of the cross-cuts, where an intrusive dike has been crushed and folded along with the enclosing slates. The vein occurs, at least in some parts of its strike, practically in the cleavage of the slate and shows evidence of faulting to some extent, and I rather suspect that slight movement is still taking place. An important feature of this vein is the strong lens of siderite that occurs near the face of the main tunnel and continues for some distance easterly. The point to determine is, Does this siderite form the bottom of the pay ore? In other words, Do the zinc and lead minerals, which occurred so extensively in the workings of the Whitewater mine immediately above, give way in depth to barren siderite?

About 600 feet from the portal of the main tunnel a dike of basic rock was encountered, which in all probability cuts the vein, but on account of the lagging and rotten condition of the ground and timbers I could not determine the fact. A drift was here extended into the foot wall, following the dike for some distance, then passing through it and entering the ground of the Whitewater mine, but no ore was encountered in this drift.

A lower tunnel, at elevation of 3,375 feet, say 400 feet below the main tunnel, was also commenced in the early days of this property, but no serious attempt was made to push it ahead. The distance to be covered in order to intersect the vein is very great, and doubtless the showing made on the main tunnel discouraged the running of another tunnel at so much greater depth. An air compressor operated by a Pelton wheel was erected near the lower tunnel, and the air pipes were brought up to the main tunnel, although the ground there was so soft that machine drills could not be used to advantage.

No connection has been made between the main tunnel of the Whitewater Deep mine and the workings of the Whitewater mine immediately above. Three raises were put up on the vein and some drifting conducted from them, as previously described, but none of these has communicated with the workings above. It is obvious that the two properties could best be operated as one mine; then the raises could be connected with the old workings of the Whitewater, establishing good ventilation, and I believe they would open up good stoping ground. In this direction, and in the advancement of the main tunnel further westerly, lie the principal hope of future production from the Whitewater property.

JACKSON MINES.

The Jackson is the most important group of mines in Jackson Basin, both in point of development and production. It is owned by the Jackson

Mines, Limited. The property consists of five Crown granted claims, aggregating 230 acres. A fairly good mountain wagon road 5½ miles long connects the mine with Whitewater station on the Kaslo & Slocan Railway, over which the ore is hauled from mines to rail at a cost of $2.35 per ton.

The Jackson vein, originally called the Northern Bell, after one of the claims in the group, was discovered in 1892, and was worked intermittently for silver-lead up to the present year. The mine also produced heavily in zinc blende, which was in part rejected in tailings and in part shipped with the lead concentrates. In 1904 the concentrating mill was remodelled with a view to saving the zinc blende, the preliminary campaign in 1905 resulting in the production of 1,200 tons of zinc concentrates, which at the time of my visit was piled up at the mine awaiting the completion of the magnetic separation works at Kaslo.

The production of Jackson mine is approximately 2,000 tons of lead ore, about one-half of which was hand sorted, the remainder being lead concentrates averaging: silver 58 ozs., lead 60%, zinc 12%; and 1,200 tons zinc concentrates averaging: silver 10 ozs., zinc 38%, lead 2% and iron 15%, the iron being chiefly as carbonate (siderite).

There is but one known vein in the Jackson property; it has been developed by 5 tunnels, varying from 500 to 900 feet in length and aggregating 3,800 feet, nearly all on the vein, besides 700 feet of raises and winzes. The vein varies from 2 to 6 feet in thickness and would probably average 3 feet, with a pay streak of about 10 inches. It is a fissure vein occurring in the Slocan slates, remarkable, however, for the unusual number of porphyry dykes associated with it and the amount of porphyry breccia occurring in the vein.

Mill.—In 1898 a concentrator was built with a capacity of 4 tons per hour, connected with No. 3 tunnel by a gravity tram, and with No. 5 tunnel by an incline-shaft on the vein and a level surface tram. The mill machinery consists of one 7 by 10 inch rock breaker, 1 set of rolls 16 by 30 inch and one set 14 by 24 inch, seven jigs varying from 2 to 4 compartments, with the usual assortment of screens and hydraulic sizers.

The ore is jigged from 20 mm. (oversize), 15 mm., 10 mm., and 5 mm., and two products from the hydraulic classifiers; the remainder of the fine ore is fed to 2 Wilfley tables. This plant cost about $50,000, including the development of the water power and the auxiliary steam plant. The mill is operated by wire rope transmission from the water wheel in the valley below, and by a steam plant in the mill building when water is short.

Mine.—The workings on the upper tunnel are caved and inaccessible, at least around the portal.

During the summer of 1905, No. 2 tunnel was operated under contract, through which the concentrator was kept supplied with milling ore at a fixed price per ton of finished lead and zinc concentrates.

The contractors mined considerable ore in the foot wall of the lead streak, which had in part been stoped out several years previous. The vein in this mine carries a bunchy and irregular streak of silver-lead next the hanging wall, followed by either solid zinc blende, or zinc blende and siderite in a brecciated dike gangue, often filling the vein to the foot wall. The principal stope commences at the mouth of No. 2 tunnel and is carried up to the outcrop of the vein near the portal of No. 1 tunnel somewhat in the nature of a quarry. From the portal of No. 2 to the winze connecting with No. 3 tunnel, good ore is showing in the roof of this stope. Sample No. 1 is from four feet in thickness of zinc ore assaying: silver 2 oz., lead 1.9%, zinc 32.5%. No. 2 is from a 6 inch galena seam resting on the zinc ore and occurring immediately under the hanging wall of the vein; it assayed: silver 71 ozs., lead 73.5%. About 250 feet from the portal of No. 2 the vein makes almost a right-angle turn, and while little prospecting has been done on the vein, it does not look so good after changing its course. In one place, however, near the face of the drift, a bunch shows in the side for 6 feet in length, averaging 18 inches thick, from which sample No. 3 assayed: silver 49.2 ozs., lead 45%, zinc 7%.

At the turn in the vein 250 feet from the portal a fault cuts off the ore at the head of the raise (Plate XIV, Fig. 4), which connects with No. 3 tunnel. The vein above this raise is quite flat and contained perhaps the best lens of galena in the mine, extending from No. 2 through No. 1 levels to surface. A porphyry dike occurs near the bend, cutting acutely into the vein and following it for some distance.

Tunnel No. 3 crosses the formation slightly until the vein is reached, at which point a short raise was put up and a vein exposed with a thickness of 12 inches, 8 feet above the back of the level. A sample was taken at this point in two cuts (Sample No. 4) which assayed: silver 0.8 ozs., zinc 21.9%. No lead was showing at this point and the zinc was mixed with siderite. At distance of 30 feet further in, a winze was reached, connecting with No. 4 level, and a raise with No. 3 level, neither of which were, however, accessible. On the level a well defined vein could be seen showing a regular hanging wall carrying a strong gouge, and along the bottom of the level a lens of siderite continued for a length of 20 feet. Some small crystals of lead occurred in this siderite, and a considerable amount of blende crystals disseminated irregularly through the lens. Sample No. 5 across this lens was taken in three cuts, where it averaged 3 feet in thickness. It assayed: 2% lead, and 14.9% zinc. Following this lens comes 20 feet of vein, showing only gouge and brecciated dike with crushed slate for a distance of 20 feet, following which comes a second siderite lens 30 feet long, from which sample No. 6 was taken in two cuts, average width 3 feet. This sample assayed: 4.9% zinc, and 21.8% iron. Beyond this point the vein becomes more irregular, and at 185 feet from the winzes the cross-cut into the hanging wall shows a large dike almost directly below the one shown on No. 2 tunnel, while the face of the drift is abandoned in a massive porphyry dike from the cracks and fissures of which large streams of water are issuing. The level at this point has, however, passed the right-angle turn in the vein, and is advancing almost at right-angles to the strike of the vein, where it carries its best ores.

The portal of tunnel No. 4 was caved down and the workings were inaccessible. The plan of the property shows that drifting was suspended before reaching the right-angle turn in the vein.

The portal of tunnel No. 5 is situated on the side of the creek immediately below the concentrator. At a distance of 250 feet from the portal a main incline-shaft has followed the vein down from surface, showing a very fair streak of lead ore. Sample No. 7 was taken from the side of the shaft near the tunnel level from a 10 inch streak of very fine looking galena. It assayed: 41.3 ozs. silver, 59.4% lead. I am informed that this pay streak continued with but few interruptions to the bottom of the shaft, a distance of 150 feet below the tunnel. At 75 feet from the portal of the tunnel a porphyry dike strikes in from the hanging wall at an acute angle, and the vein opens along the course of this dike to the portal (Plate XIV), for which distance the vein is largely composed of brecciated porphyry with some zinc blende and a little galena. About 280 feet inside of the shaft a similar dike comes into the vein in precisely the same manner, near which, for a length of 20 feet, a small lens of zinc blende occurs, from which sample No. 8 was taken. It assayed: 1 oz. silver, 31.6% zinc. A cross-cut of 6 feet has here been made into the hanging wall, but without opening up any further vein matter. The sample was taken in two cuts and averages 12 inches in width. In the absence of a survey of this level it is difficult to say what its relation is to the main ore shoots opened in the upper tunnels. It strikes me, however, that the drift is not sufficiently advanced to reach the strike of these shoots, and while there is very little mineralization between the shaft and the face of the tunnel, except the zinc lens noted for a length of 20 feet, yet the face of the drift shows a fair vein, and doubtless on being extended and connected with the tunnels above, will open up considerable ore.

The galena in the Jackson vein occurs in lens-like seams immediately beneath the hanging wall, sometimes reaching a width of 3 feet and dwindling down to an inch, or less, as followed on either strike or dip, opening again in another lens-like deposit. The zinc ore, however, is separated from the hanging wall by the galena lenses just described, and occurs mostly in a brecciated mass consisting largely of a ground mass of porphyry, in which is distributed bunches and seams of zinc blende and siderite, and occasionally a little lead. The enclosing rock is very dark graphitic slate, and near the vein is much striated and the cleavage polished. The vein is a fissure with a dip varying from 10 to 45 degrees, in the same direction as the slate, which, however, has dip of 70 to 80 degrees.

At irregular intervals on the course of the vein eruptive dikes strike in at acute angles and the fissuring, which is now represented by the vein, followed along those dikes for considerable distances. The ore appears rather to favor the vicinity of the dikes. Like many of the Slocan properties the development work has not been pushed ahead as much as it should have been, but no doubt with a good market for zinc blende in addition to the silver-lead contents of the vein, development will be pushed and the mine once more brought into condition to produce a large tonnage of concentrating ore. At least one of the drifts, preferably No. 2 tunnel, should be continued on the vein until it either resumed its normal bearing or pinched out. For there is every probability that in the former case good pay mineral would be found at the point where the vein approximately resumes its normal bearing.

BELL MINE.

The Bell claim is situated at the head of Jackson Basin, on its eastern side, about ¾ miles from the Jackson mine where the wagon road from Whitewater terminates. This road must be extended to the Bell before steady shipments can be made, except by pack or by raw-hiding during the winter months. The property had just been sold when the Commission visited it, there was no person to show the workings or furnish particulars of the ore shipments.

The upper, or No. 1 tunnel, starts in on a vein of zinc blende, one foot in width, continuing quite regular for 10 feet from the portal, where an open cross-fissure comes in (Plate XV), along which the blende widens out to 10 feet and continues along the drift at a gradually decreasing width until a slip cuts it off, 30 feet from the portal, at which point the blende was only 5 feet wide. A stope in the roof from 10 to 25 feet from the portal shows a very good vein of zinc blende. Sample No. 10, taken across the roof stope for a width of 10 feet assayed: silver 2.2 ozs., lead nil, zinc 40%. A small stope in the floor of the drift shows at a depth of 5 feet an almost solid vein of zinc blende 8 feet in width, containing, however, some small ribs of slate, but showing in either end, 9 feet apart, almost solid blende. This shoot or chimney of ore should go down for considerable depth and produce a fair tonnage.

At a distance of 45 feet from the portal there is no vein to be seen, the slips that terminated the ore having reached the floor of the drift; but a cross-cut of 27 feet easterly reached what appears to be a parallel vein on which a drift extends 22 feet northerly, showing two small veinlets of zinc blende in very tight ground, and but a few feet in length. For 20 feet south of the cross-cut the drift is timbered, but from the end of the timbers to the face, a distance of 30 feet, a very solid lens of first class zinc blende occurs that will average 30 inches in thickness. Sample No. 11, taken from four cuts across this vein, assayed: silver 2.1 ozs., lead 1.0%, zinc 46.1%. At the south face of the drift, 50 feet from the cross-cut, a porphyry dike occurs which forms, or parallels, the hanging wall of the blende for some distance. About 30 feet south of the cross-cut a raise goes up 10 feet, apparently reaching the top of the zinc lens, which, however, continues strong and regular in the bottom of the level for the entire distance of 30 feet.

Two other tunnels are opened below No. 1, but the formation and ore occurrence are different, and I am doubtful if either of these tunnels is on the main deposit opened in the upper tunnel. Two approximately parallel lenses are opened in the upper tunnel as described, but in the absence of any connecting raises or even a survey of the workings, one is not justified in pronouncing a positive opinion. Tunnel No. 2 passes through dark slates for 50 feet and then turns northerly on an irregular deposit, showing small veinlets of zinc blende and galena. A short stope or raise has been started at the end of this drift, showing in one place a small lens of galena 10 inches maximum thickness. A winze was also started under the raise, following down the galena, but was full of water.

Judging from the ore on the dump some good grade galena and zinc blende had been taken out of these workings.

The lower, or No. 3 tunnel, goes in for about 100 feet through graphitic slate, and then turns northerly. At this point a streak of zinc blende 4 inches thick can be traced diagonally across the drift for a length of 15 feet. A little further on a 5 inch parallel streak of fine-grain crushed and slickensided blende occurs for a length of 20 feet. At the end of the drift, which is 30 feet from the turn in the main tunnel, a raise or a small stope was found. The platform in the raise, covered with loose rock, prevented one from passing up through it. The rock is black slate and the foot wall smooth and well defined with a southerly dip of 60 degrees.

The ore lenses are very irregular and have a peculiarity of running diagonally with the vein walls and terminating on contact with them.

The 20 foot lens was sampled in three places, which averaged a thickness of 5 inches, and assayed (No. 9): silver 2.2 ozs., lead nil, zinc 48.5%.

This property gives promise, when opened up, of being a good producer of zinc blende. It will be noticed that No. 1 tunnel shows rather a large vein of high grade zinc blende, with practically no galena, while the lower tunnels show small irregular lenses of both galena and zinc blende, like the average Slocan veins. The deposit of high grade, almost lead-free, blende in No. 1, however, more nearly resembles the Lucky Jim type of ore deposit.

ECHO-ALMEDA MINE.

This group of four claims is situated on the mountain forming the head of Jackson Basin, and about one mile south of the Jackson mine; whence it is easily reached over a good pack-trail. The ore is packed from the Bell and the Echo group to head of the wagon road for $3 per ton.

The principal vein bears north (magnetic) and dips easterly from 40° to 60°. Its most productive part is practically on the dividing line between the Echo and the Almeda claims, in close connection with a large porphyry dike. The property has been operated for silver lead and is said to have produced ore averaging about 120 ozs. of silver per ton.

The upper tunnel has its portal on the Almeda claim and is connected with the Echo tunnel below, and with surface through a shaft situated on the Echo-Almeda line. The tunnel extends 150 feet south of the shaft, exposing two lenses of concentrating ore, lead and zinc, 15 to 20 feet long swelling to 18 inches greatest thickness. Towards the end of these lenses some high grade zinc occurs, which in turn gives way to iron pyrites. A strong porphyry dike, apparently paralleling the vein, is exposed in several places along the upper tunnel, and when branches from this main dike occur in the veins good lead and zinc ore has been found. The southern continuation of this vein is cut off by a fault having a southern dip. I could not see that any effort had been made to locate the vein beyond the fault.

The Echo tunnel is in part located on the fault plane previously referred to and intersects the vein 200 feet from its portal, right on the fault, where, I am informed, a good bunch of lead ore was found. The tunnel followed the vein from the fault northward and connects with the shaft on the Echo-Almeda line. This shaft extends 25 feet below the Echo tunnel drift and at 20 feet a stope is carried back along a smooth and regular wall to the fault plane, where there is a very fair showing of galena and zinc blende in a vein 3 feet wide, as shown in a winze following the fault for a depth of 15 feet. From this winze to the shaft the vein shows one foot of high grade blende along the hanging wall, mixed with which is galena in irregular bunches and in veinlets. Below this pay streak comes 2 feet of siderite, mixed with zinc blende, forming good milling ore. The hanging wall streak was sampled in three cuts, No. 12 assaying: silver 14.6 ozs., lead 2.6%, zinc 46.3%.

The large porphyry dike described with the upper tunnel workings is also seen in this level, but has not been exploited much north of the shaft, though the prospects of discovering pay ore in that direction are, in my opinion, very good indeed.

The cross-cut tunnel has its portal in the Almeda claim, at a point about 100 feet below the upper tunnel. It continues as a cross-cut for 170 feet and then follows the assumed course of the vein, without however, having anything like a mineral vein to follow, so at 60 feet the cross-cutting was resumed and continued for a further distance of 140 feet easterly, without, however, finding anything of value. Drifting was then resumed along the course of the vein and some mineral encountered from 110 to 140 feet south of the last described cross-cut. The mineral occurs as two small lenses of siderite, showing some zinc with a slight sprinkling of galena.

The developed ore shoot in the shallow tunnels is about 150 feet in length. The high grade galena has been for the most part stoped out, but there is a fair tonnage of concentrating ore that will yield a galena concentrate, and, at least 20% of zinc blende concentrate, as well as some high grade blende that can be hand-sorted. The vein should be prospected for south of the fault, more especially as the best ore is found near the fault plane and the ore shoot appears to have a southern pitch in the vein.

SLOCAN STAR MINE.

This property, owned by the Byron N. White Company (Foreign), consists of 18 mining claims, 460 acres, situated near Sandon. The Slocan Star, the only developed vein on the property, was discovered in October, 1891. It has been operated for silver-lead ever since and for zinc blende during the last two years.

The mine has been opened by 4 cross-cut tunnels varying in length from 50 feet in No. 2 to 860 feet in No. 5, while a still deeper tunnel is contemplated. The drifts on the vein aggregate 8,389 lineal feet, with 3,107 feet of raises and winzes. The vein varies in width from a mere clay gouge up to a width of 40 feet and is said to have contained pay ore for the greater width in some of the shallow workings. The vertical depth from the outcrop to the lowest workings on the vein is 625 feet.

Production.—The Slocan Star has produced 34,000 tons of lead ore and 4,183 tons of zinc blende, the aggregate value of which amounted to $3,000,000, from which $567,500 was paid in dividends, or 18.92%.

Mill.—The mill has lately been remodelled for the saving of zinc ore, and appears fairly complete in all particulars. It is operated by a water wheel which develops 75 to 80 h.p., from a fall of 470 feet. The discharge water from the wheel situated at the top of the mill supplies the full amount for the jigs and tables used in concentrating the ore. Two flumes supply the wheel, that from Sandon creek gives water enough for five months, commencing about the middle of April. A much longer flume runs to Cody and is used when the supply from Sandon creek falls short. The mill has also a steam plant. It is at present idle and, I am informed, will not start up before spring. The litigation in which this property is involved is said to be the principal cause of the shut-down.

The mill is located on the hillside and connected with No. 5 tunnel by a gravity surface-tram which delivers the ore in the mill bins at 4 cents per ton. The machinery consists of a 9 by 15 inch Blake crusher, 3 rolls 14 by 26 inch and one 12 by 36 inch, 6 four compartment jigs, 2 three compartment and 4 two compartment, 6 hydraulic sizers, 4 Wilfley tables and 4 Frue vanners. The general ore treatment can be clearly followed on the excellent flow sheet compiled by Mr. W. F. Robertson, Provincial Mineralogist. (Plate XVIII).

The plant has a capacity of 5 tons per hour, cost $40,000 and represents about the best class of concentrating works at Sandon. It was built in 1896, and remodelled for zinc recovery in 1904. The operating cost is about 53 cents per ton.

The following zinc shipments made during 1905 are remarkable for their uniformly high silver contents:—

ZINC SHIPMENTS FROM THE BYRON N. WHITE CO. OF SANDON, B.C.,

TO

THE UNITED STATES ZINC CO. OF PUEBLO.

Returns made from Denver.

Date.	Gross Wt.	Moisture.	Net Wt.	% Zinc.	Ozs. Silver
1905.					
March..10	158,120	4.8	150,530	33.5	33.4
" 14	178,000	5.2	168,744	32.1	44.2
" 14	173,700	4.5	165,884	33.2	43.5
" 18	156,510	2.7	152,284	33.9	47.5
" 24	199,530	4.0	191,549	33.0	43.4
" 25	193,090	3.2	186,911	32.0	42.4
" 27	175,565	4.0	168,542	31.0	40.6
" 27	142,645	3.1	137,367	31.9	42.4
" 29	181,065	4.3	113,279	31.2	42.8
April... 3	177,803	4.5	169,802	33.5	43.7
" 8	180,583	4.7	172,096	31.5	42.0
" 13	180,005	4.3	172,265	32.5	43.6
" 13	175,623	3.2	170,003	33.1	42.3
" 25	155,925	2.3	152,339	33.0	42.0
" 25	123,340	2.3	120,503	31.3	44.0
" 25	216,780	2.8	210,710	32.7	43.0
May... 7	86,500	4.0	83,040	34.6	44.0
" 10	355,020	2.8	345,079	31.7	41.8
" 10	206,340	3.7	198,105	31.2	43.4
" 10	123,880	2.1	120,783	31.5	41.0
" 12	233,340	2.0	218,873	31.8	44.4
July...20	229,020	1.3	226,043	33.0	29.7
" 20	148,240	1.0	146,758	32.0	32.5
" 20	103,120	1.0	102,089	31.9	29.6
" 25	115,360	2.3	112,307	29.7	27.8
" 31	43,740	2.0	42,865	32.8	31.2
" 31	158,980	2.8	154,529	31.1	29.6
Aug.... 4	232,660	1.9	228,239	32.4	33.0
" 4	55,700	3.0	54,029	31.6	34.1
" 12	115,840	1.8	113,755	32.6	33.0
" 16	78,280	4.0	75,149	33.5	39.0
" 18	122,380	2.0	119,932	33.0	34.6
" 18	42,920	2.5	41,847	33.3	40.8
" 29	43,340	1.5	42,690	33.6	44.8
" 29	370,550	1.5	364,992	33.8	36.8
" 29	290,340	1.9	284,824	32.5	43.0
" 29	57,160	2.0	56,017	33.1	48.2
" 31	120,020	1.5	118,220	32.5	46.1
Sept.. 2	166,160	2.0	162,837	32.0	44.6
" 2	121,840	1.0	120,622	31.1	42.5
" 8	354,740	1.8	348,355	31.9	42.4

Mine.—This is situated on a steep mountain side, in the western part of which a very large granitic mass, or stock, occurs which has had considerable influence on the formation of the vein fissures. There are also numerous smaller dikes, in all probability branches from the main stock, which are intersected by the vein in various places throughout the workings. West of the shaft on No. 5 level, the vein is diverged by the main porphyry stock, which it courses around at a distance of from 2 to 50 feet, and passing round the western end of the stock resumes its general direction making large ore deposits, but from a point 300 feet west of the shaft until it reaches the western side of the stock, the vein or veins contain but little ore. This is one of the disputed points, on which it must be distinctly understood that I pass no opinion whatever, the plaintiffs in the litigation claiming that the Star vein is cut off immediately east of the shaft. The Star vein outcrops very prominently around No. 2 tunnel, and slightly west of it a spur from the main vein is exposed on the trail for a width of 5½ feet, 4 feet of which is good zinc and iron ore. Sample No. 15 gave: silver 2.2 ozs., lead 1.0% zinc 29%. Further westerly the outcrop of the main vein can be seen in two or three places adjacent to the trail, exposing a wide outcrop (including a slate horse) showing zinc and carbonate of iron with galena. Considerable surface gophering has been conducted around this place in search of pockets of galena, while east of No. 2 level or cross-cut, the vein was quarried out on surface and the workings are now caved. Tunnel No. 1, I was informed, is completely worked out, caved and abandoned.

Tunnel No. 3 intersects the vein practically at the east end of a large horse, or the point where two branches of the veins meet. East of the junction the vein has a bearing of north 50° east, with a dip into the mountain of 50 degrees. The stope immediately in front of the tunnel is 20 feet wide and comes to a point about 70 feet easterly. I went down a winze through this stope to an intermediate level 60 feet vertically below tunnel No. 3, and sampled a body of zinc ore exposed by a cross-cut 16 feet into the foot wall, including 4 feet of quartz and slate in the center. This foot wall lens is good concentrating ore, but rather silicious. Sample No. 25 showed: silver 1.6 ozs., lead 0.7%, zinc 19.2%. Mr. Gardé sampled the west end of this intermediate level finding 7 feet of zinc ore (Sample No. 31) showing: silver 3.1 ozs., zinc 27%.

Tunnel No. 4 intersects the vein also in a wide place, practically at the east end of the horse. All the lead ore has been stoped up clean to surface, leaving, however, lenses and veins of zinc ore very difficult to estimate, but no doubt in large quantity. The ore sampled on the intermediate level between 3 and 4 tunnels would be merely part of the available milling ore standing above this level.

At distance of 120 feet easterly of the main No. 4 cross-cut a winze connects with No. 5, the ground between the cross-cut and this winze being all stoped out below the level for a considerable depth; there is, however, a good bunch of zinc ore on the eastern side of this winze, and zinc can be traced 200 feet along the bottom of the level to where a second winze connects with No. 5. This block is represented by samples Nos. 16 to 20, inclusive. No. 16 assayed: silver 1.5 ozs., lead 1.2%, zinc 25.7%. No. 17 assayed: silver 2.9 ozs., lead 0.8%, zinc 26.7%. No. 18 assayed: silver 1.5 ozs., lead 2.2%, zinc 15.7%, No. 19 assayed: silver 1.7 ozs., lead 1%, zinc 17.4%. No. 20 assayed: silver 1.4 ozs., lead 1.3%. zinc 20.2%.

Passing easterly from the second winze on No. 4, I found 150 feet of practically barren vein. Quartz lenses occur at intervals and sometimes also a little siderite, but these cross the strike of the vein at acute angles and appear to terminate at the main walls. At other points the vein is merely a 6 inch gouge of crushed black slate. Succeeding this barren stretch of 150 feet there follows 100 feet of very good vein, mostly, however, siderite alternating with zinc blende. Samples No. 21 assayed: silver 1.6 ozs., lead 0.8%, zinc 31%. No. 22: silver 2.3 ozs., lead 1.3%, zinc 38.5%. No. 23: silver, 1.1 oz., zinc 29.2%. No. 24: silver 1.4 oz., zinc 20.5%. The level extends 22 feet beyond this lens and ends in barren ground, the vein being again reduced to about 10 inches of crushed black slate and gouge.

The siderite lens previously described consists in part of seams crossing the general strike of the vein at an acute angle and terminating at the walls; the last seen of this lens is at a point 22 feet from the face, where it crosses towards the foot wall, along which it probably continues, the drift having evidently followed the hanging wall.

No. 5 level east passes below this ground at a depth of 165 feet vertical, and shows siderite in several places and also some zinc. The vein on No. 5 east is very irregular and wavy, the drift very crooked. The vein would probably average about 4 feet in thickness and contains much more quartz than on the level above, but not nearly as much siderite as in No. 4 east. At about 500 feet east of the main cross-cut a raise shows 4 feet of siderite and zinc blende with a slight sprinkling of galena, and again at 400 feet east of No. 5 cross-cut there is a cut in the hanging wall showing 4 feet of siderite and a little blende.

In two places along this drift small stopes were started, but the vein did not open out above the level, and above all, the mineral that they were looking for, galena, did not occur in a profitable quantity, but much of this ground will, I believe, when opened up, produce milling ore when zinc can be mined and milled at a profit. I did not, however, at any place on this level see as good a run of either siderite or blende as on the No. 4 level. At one point a porphyry dike shows in the hanging and below it a spring of water issues. The vein immediately under the dike is chiefly quartz, carrying a little galena and fragments of blende with some siderite. It looked to me a good prospect to sink on.

Going westerly on No. 5 the vein has been stoped in the roof continuously, and in one place passes through a small dike and contains there about 20 inches of blende with a little siderite. The stopes increase in height as they reach the main winze (shaft) sunk below this No. 5 level, and the vein at this point is very wide as will hereafter be shown. Immediately west of the shaft the vein follows close to the main stock, previously described, but the pay streak is very narrow. At about 1,000 feet west of the shaft, what is called the main vein is again picked up and opened for a length of 350 feet, about 100 feet of which is in pay ore. Sample No. 26 was taken from a stope where the ore showed a beautiful ribbon structure of blende and quartz with galena and a little siderite, 3 feet wide. It assayed: silver

28.5 ozs., lead 0.7%, zinc 23.9%. Sample No. 27 was taken from the same Silversmith stope, 30 feet east of the latter, and for a width of four feet. At this point about 4 inches of galena occurs on the hanging wall as well as some little galena disseminated throughout the vein. The sample assayed: silver 50.8 ozs., lead 0.6%, zinc 19%. This very fine stope produced the bulk of the rich silver-zinc concentrates shipped in 1904.

In the main winze below No. 5 level, the west end shows a magnificent vein averaging 12 feet in width between the 5th and 6th levels. The pay streak is mostly galena. I sampled 4½ feet. Sample No. 30, on the foot wall side assayed: silver 111.9 ozs. lead 78%, zinc 3%. About 25 feet below No. 5 level there was another streak of about 18 inches of galena towards the hanging, but this was inaccessible for sampling. The vein filling between these streaks is quartz and slate with a little blende. About 6 feet has been stoped west of the shaft on this splendid ore lens, while a stope opening on the east side, about midway between levels, shows 7 feet of quartz and blende, a brecciated vein, mostly quartz with pieces of slate, fragments of zinc blende and a sprinkling of galena throughout the mass, and about 4 inches of galena on the hanging wall. Sample No. 28 was taken across this stope, width 7 feet. It assayed: silver 12 ozs., lead 4.4%, zinc 18.9%. This lens appears to have its greatest width at the winze and is of short length, probably not exceeding 80 feet.

No. 6 level extending west of the winze on the course of this lens I was not shown, while easterly on No. 6 level the ore lens has practically disappeared 30 feet from the shaft. The vein, however, continues 8 or 10 feet wide in a very irregular country, containing several small dikes, though small bunches of siderite, blende and galena occur at intervals in the vein. At one place the vein appears to fork and was followed southerly for some distance, when it turned and almost resumed its normal course. The main drift continuing through the dikes finally reached a distance of 400 feet from the shaft where I took sample No. 29, on September 19, from 12 inches of galena and blende in a quartz gangue; it assayed: silver 11.5 ozs., lead 28.9%, zinc 18%, iron 7.4%, insoluble 36.1%. This ore is just coming in, the pay streak is about 18 inches wide, and the ground very loose and extremely wet. There is considerable development of iron pyrites in the vein at this, the deepest point in the mine; a little yellow copper ore can also be noticed in specks associated with the quartz and galena.

In the upper portions of the mine, the galena, and sometimes the blende, contains a little chalcopyrite, but more often gray copper, particularly where the silver value is high. It would appear that on the 6th level copper ore occurs almost exclusively in the form of chalcopyrite. It is probably that the southern vein will soon fall into the main vein and include between them one of the slate horses which form so prominent a feature in this mine.

The 6th level is opened about 625 feet vertical below the outcrop of the vein at the highest point on the mountain, and there occurs on and above this level probably as rich a bunch of silver-lead as has been found anywhere in the mine. The blende also occurs in this lens at the main winze of normal character and in fair quantity, while the east face of No. 6 level appears to be entering another lens of normal character and composition. Consequently, it can be at least claimed that the vein at its deepest point, taking the mine as a whole, carries ore of normal richness and composition; in other words, there is no falling off in values with depth. There follows, however, the matter of quantity of mineral in the vein, and under this head the showing on No. 4 tunnel east is very fair, the development of siderite alone being abnormal; while on the 5th level, east of the main crosscut, the vein on the whole is poor, but with a market for zinc, would, in several places, pay to mine, and there is a strong probability that bunches of good lead ore would be found in stoping and possibly some lenses of normal richness; and if so, the whole vein could be profitably stoped. Siderite occurs everywhere in the vein, from the outcrop to the lowest depth developed.

Studying the mine sectionally we can roughly divide the ore deposits into a central or main shoot, an east and a west ore shoot. The central shoot is by far the largest of the three, and carries pay ore continuously from the outcrop to the 6th level.

The east ore shoot has an easterly pitch in the vein, develops massive siderite as vein filling on the 4th level and is poor on the 5th level, quartz predominating.

The west ore shoot has a westerly pitch in the vein, but owing to the shape of the hill the 5th level cuts it at much less depth from surface than the eastern shoot is explored. The western shoot continues strong on and below the 5th level, west of the granitic intrusion, and will probably carry down pay ore below the 6th level, and in the wider portions of the vein perhaps to much greater depths.

I had arranged for a number of illustrations to show clearly the ore occurrence in the Slocan Star vein, including a plan of No. 5 level and several sections across the vein, which is really one of the most interesting, and probably the largest developed vein in the Slocan. At the last moment, however, the manager refused to have any data whatever taken from the working plans, and hence I have been compelled to describe briefly this interesting mine without the aid of illustrations, so essential to a proper understanding of the great vein and its ore shoots.

It must be remarked, however, that the owners of the Slocan Star have been in litigation over that nightmare, "Apex rights," which I had heretofore believed was an expensive uncertainty confined exclusively to United States mining practice.

RICHMOND GROUP.

This property consists of six mining claims adjoining the Slocan Star group. The formation consists of the usual Slocan series of slates in which somewhat irregular faulted veins occur, carrying some high grade galena and considerable zinc blende. In fact the ores are identical with those of the Slocan Star, and will require the same treatment.

In the lowest (No. 5) tunnel a brecciated vein, 6 feet wide, could be seen outcropping and extending for a few feet in the tunnel, where it is apparently cut off. This tunnel is comparatively short, and at present unimportant, from the fact that while considerable fault movement can be noticed, but little ore or any regular vein formation has so far been developed.

Some zinc ore is visible in No. 4 tunnel near the mouth, but the workings inside do not appear to have exposed any regular vein.

No. 3 tunnel, or middle tunnel, shows a better defined vein than either of the others; in one place a short lens of zinc blende, 3 feet wide in its thickest place and 20 feet long, occurs, and towards the face of the tunnel a regular and well defined vein has been opened up for the last 40 feet of drifting, and looks as if pay ore might be developed at any moment. This property can only be classed as a "prospect," and owing to the steepness of the mountain side and prevalence of snow slides, it can only be operated satisfactorily during the summer season. The amount of work so far done cannot be said to show up a regular or continuous vein, excepting, perhaps, that near the face of No. 3 tunnel. The large brecciated vein showing zinc blende at the mouths of Nos. 4 and 5 tunnels, appears to be out of place. It is, however, possible to operate the property by a main tunnel having its portal in some place secure from snow slides, but such development would scarcely be warranted until further prospecting shows up a regular and profitable vein in place. The chances for developing a pay mine in this group of claims is, in my opinion, good. The work so far done shows the upper tunnels to be the more promising, and in these, or say in No. 3 and the next above, it would be safest to develop the vein first.

RUTH MINES.

The Ruth mines, owned by the Ruth Mines, Limited, consist of 14 Crown granted claims—about 400 acres—situated at Sandon. There are two developed veins, known as the Ruth and the Hope, discovered in 1892 and worked since that date for silver, lead, and the Ruth for zinc blende since 1904. The Hope not having produced zinc blende in commercial quantity, was not examined.

The Ruth vein has been developed by five tunnels to a vertical depth of 600 feet below the outcrop. The tunnels vary in length from 600 feet in No. 1 to 3,000 feet in No. 2, while No. 5, the lowest, tunnel is 2,000 feet long. Nearly all the tunnel work is on the vein and fault, and aggregates about 10,400 feet of drifts, while the raises and winzes amount to 2,500 feet of additional development. The vein varies from 2 feet upward and would average about 4 feet in thickness; it is a typical Slocan vein carrying galena, blende and siderite in lens-like masses. Between the lenses in places there is nothing but a clay gouge or crushed slate to follow, and sometimes this is very difficult to follow, as for example in No. 2 tunnel, where, in the early development of the property very extensive cross-cutting (amounting to 1,300 feet) together with 1,200 feet of drifting failed to find the continuation of the vein beyond the fault.

The property has produced 17,410 tons of silver lead ore, 15,000 tons of which was hand sorted. About 1,000 tons of blende concentrate was produced and stored at the mill during 1905, which is said to assay 36% zinc, 1.5% lead, 14% iron and 12 ozs. of silver per ton.

The silver-lead ore averages about 65% lead and 85 ozs. of silver. The mine is credited with the payment of $165,000 dividends.

Mill.—The Ruth mill is situated on a hill side, practically in the town of Sandon and connected with the Kaslo & Slocan Railway, over which concentrates can be shipped direct. The mill is operated by water power and connected with the principal tunnels of the Ruth mine by a short wire tram. It has a capacity of 4 tons per hour and cost approximately $65,000, including flume and power plant. The Ruth mill is well constructed, fairly well arranged, and is the best lighted hill-side mill in the Slocan. It was erected in 1899 and re-arranged for zinc saving in 1904. The machinery consists of one 7 x 16 inch Blake crusher, two 14 x 24 inch rolls, four 2 compartment and four 3 compartment jigs. The sizes jigged are the oversize (about 20 mm.), then 15 mm., 8 mm. and 4 mm., together with the coarser product from 4 hydraulic sizers. Four Wilfley tables and one double-deck Evans buddle take care of the finer product of the milling operations.

Mine.—The vein above No. 3 tunnel has been stoped out and the fillings run down and passed through the mill for their contained lead. The vein appears to have carried ore from near the mouth of the tunnel to the fault, and it also appears that the ore was continuous for similar distances in the tunnels above No. 3.

No. 4 tunnel (see Plate XVI for plan of workings) is presumably driven in on the vein, which is extremely irregular and small; so much so that it was hardly recognized until a distance of 500 feet had been reached from the portal and connection made at that point with tunnel No. 3. A winze was also sunk about 70 feet below No. 4 tunnel, but nothing of value was discovered. From this point the tunnel advances on a somewhat regular vein, but barren of ore until the fault is reached, and at that point about 3 feet thick of vein matter occurs. It is mostly siderite, and contains scarcely any mineral of value. From the point where the vein was displaced by the fault a south cross-cut was started, which in part paralleled the fault and over-shot the other end of the vein by about 80 feet; in other words, from the point where the cross-cut intersected the vein, it extended east 80 feet and for the greater part of this distance has been stoped in the roof and partially stoped in the bottom, the remaining blocks showing a fair mineralization, principally of blende.

Passing westerly a main winze was sunk 150 feet from the point where the cross-cut intersected the vein and extended in depth to the No. 5 tunnel. From this winze at a depth of 30 feet below No. 4 tunnel an intermediate level was driven easterly 200 feet and westerly 70 feet. About two-thirds of the vein matter has been stoped out over this level for the entire distance specified (270 feet.) Some small stopes occur in the bottom of the level and show very fair zinc blende and an average of 3 inches of galena against the hanging wall.

Most of this work, I understand, was performed by contractors who stoped the ore on a basis of tons of concentrates produced in the mill. The rate, however, was for the most part based on lead concentrate, and only towards the close of the campaign was the contractor allowed anything for zinc as that mineral was then of little value; even when concentrated.

At the end of this 200-feet intermediate level there is a very nice ribbon-structure vein consisting of an irregular streak (about 2 inches) of galena, against the black slate hanging wall, then a breccia of blende cemented by siderite, and the remainder made up of alternate seams of siderite and blende with an occasional sprinkling of galena (see Plate XVII, Fig. 1). Sample No. 13 was taken at this point in two cuts across the vein; it assayed: silver 13 ozs., lead 5.5%, zinc 30.4%.

This bunch of ore is at a splice in the vein and is just, as it were, coming in, and future developments, by advancing this drift easterly, will show up its length, though the break or fault should be met with in about 30 or 40 feet. Fifty feet east of the main winze in this intermediate drift a winze has been sunk 70 feet, and a second intermediate level driven from it, connecting with the shaft and extending easterly from there 150 feet. At the collar of this winze and extending down for 15 feet on its eastern side, a vein of siderite and blende occurs, almost a duplicate of that taken in sample No. 13, and in various places between this point and the bottom of the winze very solid kidneys of blende occur in the vein, which would average about 18 inches of rather solid ore.

At a depth of 60 feet, and about midway between the shaft and the winze, sample No. 14 was taken from the vein, which showed solid blende and a little galena, in two cuts averaging 18 inches in thickness. The vein here would have to be stoped to a width of 4 feet. The sample assayed: silver 20.9 ozs., lead 13.1%, zinc 31.7%.

Descending to the second intermediate level, which has been advanced 150 feet from the shaft easterly, the vein east of the stope is, as a whole, rather poor. In the eastern face there is about 3 inches of galena and blende coming in, and the ground appears to me favourable for the production of mineral. I rather expect that another bunch, or lens, will be found between the present face of the drift and the fault. Blende also occurs in one or two places along the back of the level, but in very small bunches. The ground, however, is, on the whole, poor, and the drift has apparently passed in very close to the bottom of the lens. Yet, as we have seen, the upper portion of the block contains a good vein, the winze proves its continuity, so the block as a whole will no doubt pay well to stope out regularly.

The main winze continues below the second intermediate level for 110 feet on the dip of the vein and there connects with No. 5 tunnel. As this winze (or raise from No. 5) was full of mill ore it was impossible to examine it below the intermediate level, much as I would have liked to have done so on account of the depth to which the vein was exposed at this point, and moreover because the vein has not been located with any certainty in the No. 5 tunnel workings, immediately adjoining this winze.

Returning to No. 4 tunnel, the lens on which the main winze was sunk continues westerly along the vein for about 150 feet, at which point a stope was put up about 30 feet, opening a vein 3 feet wide, mostly silicious siderite containing bunches of blende. At about 40 feet further west along the level this lens pinches out, and for some little distance the vein was barren, but at the time of my visit the face showed a seam of 6 inches of fine-grain galena ore, containing a little blende. The rock is soft black shale standing almost vertical and showing fault movement. The advance made in this drift, since the examination, traced the vein into a faulted zone, as shown on the plan (Plate XVI). This work, however, I have not seen.

No. 5 tunnel is, according to aneroid reading, 240 feet below No. 4. It is wide enough for a double track almost to the fault, and is a very fine piece of tunnel work. I was informed that No. 5 tunnel was almost completed before the faulting in the upper levels had been developed, and no doubt it resulted in considerable disappointment.

A careful examination of the tunnel shows a well defined, though small, vein 350 feet east of the fault, while at 300 feet a small lens of blende and siderite can be seen along the roof of the tunnel for about 40 feet; in places it is 18 inches wide and shows some good seams of galena. This lens looks very promising and occurring, as it does, in line with an ore shoot above No. 4, there is good reason to expect paying results from the development of the ground between these tunnels; and perhaps an intermediate drift easterly (toward surface) would prove up good ore at a shallow depth.

At the point where the tunnel crosses the Ruth-Lone Star line a strong vein can be seen, consisting of quartz and slate breccia, containing some siderite and small crystals of zinc blende. This is, I believe, the Ruth vein west of the fault. There are some water courses adjacent to this vein from which, and the vein proper, considerable water is issuing, draining no doubt a wide area of the country.

I could not detect any ore or even a regular vein in the workings west of the Ruth-Lone Star line on No. 5 tunnel.

The Ruth mine contains some large blocks of zinc-lead concentrating ore. Further development of the lower intermediate, between Nos. 4 and 5 tunnels, easterly towards the big fault, should open up another run of good stoping ground, while the lens east of the big fault on No. 5 gives promise of opening out into another important ore shoot. Lastly, the old workings might, under the leasing system, pay for re-working on a blende basis.

PAYNE MINE.

The Payne mine, owned by the Payne Consolidated Mining Company, Limited, is situated about 2 miles from Sandon. The mill is located on the Kaslo & Slocan Railway, and is connected with No. 5 tunnel by an inclined surface-tramway. The property consists of six mining claims—about 79 acres. The Payne vein was discovered in 1892 and has been worked ever since for silver-lead ore; zinc ore has also been produced since the completion of the concentration mill in 1903.

The vein has been developed by six tunnels, the first four of which are worked out and abandoned. No. 5 is now the principal working tunnel, and near its face a main winze connects with No. 8 tunnel, driven in from the east side of the mountain, opening up and draining the vein to a depth of 850 feet, vertical, below the outcrop. With the exception of No. 8, the tunnels are mostly on the vein, while between the latter and No. 5 tunnel, Nos. 6 and 7 levels are opened on the vein from the main winze. The Payne vein varies in thickness from 1 to 10 feet and would average about four feet. The pay streak of galena, it is estimated, would average about four inches, though in some places it has exceeded 8 feet in thickness.

This property has produced 50,000 tons of silver-lead ore, averaging about 68% lead and 120 ozs. of silver per ton, together with 6,000 tons of zinc blende, the aggregate value of which amounts to $5,000,000. The dividends paid reached the grand total of $1,438,000, or 28.76%, which stands, so far as I can determine, the highest rate of profit and largest aggregate dividend of any property in the Slocan.

Mill.—The Payne mill was erected in 1902, at a cost of about $30,000, for a capacity of about 10 tons per hour. It is operated by water power with steam-auxiliary to tide over the dry seasons. The mill does not vary much from the average Slocan concentrator, except that the first magnetic machines for zinc separation in British Columbia were installed in this plant. For various reasons they did not prove much of a success.

The separation plant consisted of a modified Spence roaster, a Knowles and a Cleveland-Knowles magnetic separator. The roasting furnace is a very poor one and ill adapted for the light roast necessary to change siderite to the magnetic state. The cost of roasting is said to have averaged $3. per ton, and to have resulted in considerable silver losses (probably as dust). The Knowles separator consists of a fixed electro-magnet over which passes a leather belt thickly studded with iron rivets, which on passing over the magnet pick up the magnetized portion of the ore and remove it from the magnetic field; this machine was not successful. The Cleveland-Knowles machine, however, gave fairly satisfactory results on well roasted ore. Through various causes the separating plant was not a commercial success and is not now in operation.

The mill was treating ore from No. 5 dump at the time of my visit, and in addition to the silver-lead concentrate, a good zinc product (averaging 45% zinc and 15 ozs., silver) was being produced and shipped direct to the smelters without further treatment.

The Payne mill has treated 180,000 tons and produced 5,186 tons of concentrates, or one ton of concentrates from 34.7 tons of crude ore, the material treated coming largely from stope fillings in the mine and the surface dumps. The milling cost varies considerably. In 1903 and 1904 it was about 90 cents per ton, and during 1905 about one half this figure; in addition to which is the cost of repairs which is said to vary from 2 to 10 cents per ton of ore.

The gravity tram connecting the mill with the mine is about 5,000 feet in length. The head house is about 2,300 feet in elevation above the mill. The cars hold 3 tons each and the cost of delivering ore in the mill bins is 4 cents per ton.

Mine.—I went in through No. 5 tunnel to its intersection with the Payne vein, noticing 2 dykes on the way, then passed easterly to the face of the level. The main ore shoot appears to have been stoped out from this tunnel to surface, and in places as far down as the 6th level. Near the east face of No. 5 a main winze communicates with No. 8 level, opened from a tunnel entering the mountain on the side opposite to that entered by the other tunnel. This now forms the main drainage adit of the mine. The winze is equipped with an electric hoist to raise the ore to No. 5, and a chute to convey the waste to No. 8 to be trammed out through that tunnel, while the third compartment is used for a ladder-way. The sixth and seventh levels are opened from this main winze and have no surface outlets, as all the other levels, from one to eight, have.

The sixth level is opened from the winze, and extends westerly on the vein, which here contains considerable siderite, and would average for the greater part of the distance 3 feet in width. One stope is now being operated, from which I took sample No. 37, which assayed: silver 12.6 ozs., lead 5.2%, zinc 42.2%. This appears a fair zinc ore and has distributed through it some small seams of galena, alternating with siderite and blende, forming a very pretty vein structure (See Plate XV. Fig. 3). West of the stope the vein is slightly faulted to the north, and was followed for about 50 feet beyond, with indifferent results. For the last 30 feet a small dyke occupies the vein fissure and next to it is about 12 inches of siderite with a little quartz, as indicated (Plate XVII. Fig. 3).

No. 7 level extends westerly from the winze about 400 feet and terminates at the fault (Plate XVII. Fig. 2). This drift also shows the dyke observed on the level above, and in several places a vein averaging 2 feet in thickness, composed of alternating seams of siderite and blende of about 2 inches each, with considerable quartz development on the foot wall.

No. 8 level extends from the main winze and connects with No. 8 adit. The drifts appear badly mixed up, making several folds and right angles (See Plate XIX for plan of lower workings). The vein, however, was finally intersected at a point 850 feet below its outcrop and shows at this place some mineral. A stope was opened out east of the intersecting cross-cut, near which preparations were made to sink a main winze below No. 8. The vein was, to some extent, stoped above this point and appeared to be of normal composition and value.

I sampled a 2 inch lead and zinc streak (Sample No. 38) on the floor of the level east of the winze and probably about the center of the vein, as there was evidence of a streak 4 feet further north, which would probably be the main hanging wall of the vein. The sample assayed: silver 113 ozs., lead 67.9%, zinc 12.1%. This was from the deepest point of development on the vein. On the west side of the cross-cut the vein opened out to 10 feet, is stoped above the level, and produced, I am informed, a fair tonnage of lead ore. The end of the stope is mostly siderite, though there is quite a little blende also. (See Plate XVII. Fig. 4).

Beyond this point the water was too deep to permit of examination, but I was informed the level was abandoned on reaching the fault shown on the 6th and 7th levels. Near this fault a very large stream of water issues, perhaps draining the entire vein above. I suspect it comes from a water course following the wall of the dyke which occupies part of the vein fissure above this level, as previously described and illustrated in Figs. 2 and 3. (Plate XVII).

It appears to me that the enclosing slate rock is much harder on the 8th level than on the 5th for example, and that it gradually increases in hardness between the 5th and 8th levels. The vein, neglecting the pay ore, is more regular and better defined on the 7th level than on the 6th, and best of all on the 8th level. The galena is apparently less plentiful in depth. A little is showing on the 6th and 7th levels, though on the 8th, I understand a good bunch has been taken out of the stopes and some remains visible at this date. The siderite, however, undoubtedly increases with depth, as does also the amount of quartz in the vein, but not to the same extent as the siderite.

I have previously pointed out that the vein fissure, on the sixth and seventh levels, followed for some distance the course of a porphyry dyke which forms the hanging wall of the vein; near the eighth level, however, the dyke apparently flattens, disappearing in the hanging wall, while the vein continues on its normal dip. It is worth noting that the heavy decrease of galena in the vein is, to some extent, co-extensive with the porphyry hanging wall, but not exclusively so; hence it may be assumed that a slate hanging is more favourable for the deposition of galena, and therefore, that below the eighth level where this condition again obtains, normal deposits of galena should be found. However, I attach more importance to the structure of the vein itself, which on the eighth level is one of the best defined and strongest veins in the Sandon district, at least that portion of it extending westerly from a point slightly east of the intersecting cross-cut. The vein here gives the appearance of opening out into one or more lens-like deposits at moderate depth. The whole showing on the eighth level fully warrants proving the veins to greater depth, and I would add, I have confidence that pay values will be found in the wider portions of the Payne vein below the eighth level.

On referring to Plate XIX—reduced from the working map—it will be noticed that east of the intersecting cross-cut the eighth level, following a vein, diverges widely from the normal course of the Payne vein, which on the seventh level from a point 200 feet west of the main winze bears North 30° East, while the eighth level vein for a practically similar distance bears North 70° East; but for the fact that a slight fault in the Payne vein occurs, near the main winze, one would be led to the conclusion that the eighth level east is on a branch, or spur, from the main vein. My examination was not made in sufficient detail to justify an opinion, but the circuitous drifts and cross-cuts on the eighth level, evidently in search of the vein, show not only that something is amiss, but also that the position of the winze to develop the Payne vein below the eighth level is perhaps too far east. Consequently drifts should be extended west from this winze, at least to the fault, before the Payne vein can be assumed to have had satisfactory development in depth.

The ore reserves in the Payne mine are so nearly worked out that an estimate of the available tonnage of milling ore is not justified. This state of affairs is, in my opinion, due as much to lack of development as to changes in the vein. The Nos. 6 and 8 levels should have been driven westerly to the limits of the ore shoot as outlined by the stopes above No. 5. While, perhaps, of greater importance, the opening of at least one more level below No. 8, should have long since been completed. The shallower portions of the mine would probably yield a further tonnage of high grade ore if leased, and work by lessees would undoubtedly lead to discoveries of mill ore in the majority of the workings, but in depth further development must precede future production.

IVANHOE AND ELGIN MINES.

These mines, owned by the Minnesota Silver Company, Limited, consist of two full claims situated near the summit of the mountain, overlooking Sandon to the west. The Ivanhoe vein was discovered in 1893, and has been worked for silver-lead ore almost ever since. Zinc ore has been produced during the last 3 years.

The mines are developed by two main tunnels, aggregating 2,880 feet in length, and a similar amount of drifts on the vein, together with about 1,000 feet of raises and winzes. The greatest depth reached on the vein is 540 feet below the outcrop.

Further details of the vein were not available, nor could I obtain any records of the production of the mine. Inasmuch as the "developed ore" had been stoped out, and work in search of other shoots had not opened up pay ore, the property was not examined.

Mill.—The Ivanhoe mill is situated on the C.P.R. Railway, about ¼ mile below the town of Sandon, and connected with the mines by a wire tram 8,400 feet in length. The mill is well arranged, well lighted and modern in its equipment. The machinery consists of one 10 inch by 20 inch rock breaker, two 14 inch by 30 inch and one 6 inch by 42 inch rolls, an automatic sampler, one two-compartment roughing jig, one 3 compartment and six 4 compartment jigs treating 1 in., ⅝ in., 10 mm., 6 mm., 4 mm., and 2.5 mm. ore from screens. Eight Wilfley tables treat the material from the hydraulic classifiers. A picking belt, 45 feet centers, between rock breakers and the storage bins, is a good feature in this mill, but it should not be lost sight of that it is invariably necessary to wash the ore before effective picking can be hoped for.

This mill was erected in 1900, at a cost of $50,000. It has a capacity of 6 tons per hour and is operated by water power. The milling cost is about 57 cents per ton. Delivery of ore from mine to mill over 8,400 feet of wire tram cost 12 cents per ton. (See Plate XXV).

NOBLE FIVE AND GOODENOUGH MINES.

The Noble Five mine and mill, and the Goodenough mine are described in the report by Mr. A. C. Gardé, which is to be found in a subsequent portion of this volume.

MONITOR MINE.

The Monitor mine is situated at Three Forks in the West Kootenay district, the lower tunnel being practically on the line of the C.P.R. Railway. The property consists of 8 claims and is owned by the Monitor & Ajax Fraction, Limited.

But one vein has been developed—the Monitor—discovered in 1895 and worked intermittently for silver-lead ore. The vein is opened by five tunnels varying from 810 to 1,200 feet in length and aggregating 4,300 feet, together with about 1,500 feet of drifts, winzes, etc., on the vein. The greatest depth reached is about 490 feet below the outcrop. The mine has been closed down for a year or so pending the completion of the new mill at Roseberry, and was not visited by the Commission. I understand, however, that the property has been a fair shipper of high grade hand sorted lead ore running from 35% to 40% lead, 18% zinc, about 120 ozs. silver, and $10 per ton in gold. There is, it is claimed, a large tonnage of zinc concentrating ore in the old stulls, on the dump, and in some of the stopes, that will be treated at the new mill.

Mill.—The new concentrating mill is situated on the C.P.R. Railway at Brockman, near Roseberry, on the shore of Slocan Lake. The site is a level one, and the mill, as a whole, extremely well built. The general design is, however, not of the best, though the level site offered all possible facilities for the construction of the most approved design in concentrators. I am indebted to Mr. M. Gintzberger for an excellent flow sheet of the mill. This is shown in Plate XXX, a reduction from Mr. Gintzberger's drawing, which clearly indicates the operations of the concentrator. It is said that the intention of the company is to treat the ore from its own mines at Three Forks, and also to do a custom business, or, at least purchase concentrating ores and treat them in this new mill at Brockman.

A magnetic separating plant will, it is said, be added to the mill equipment at some future date. The ore will be unloaded from the C.P.R. cars by shoveling into a chute leading directly to a Hadfield all-steel crusher, thence (without washing,) the ore passes to a picking belt and is delivered to an elevator which takes it up to a Snider sampler, consisting of three cast iron discs having two orifices each, through which the sample passes, while the face of the disc deflects the main stream into the stock bins below. There is no crushing or even mixing devices between the cutters or samplers, the second simply cutting the stream taken out by the first one, and the third cutting after the second. The design and arrangement of this sampler do not conform to the generally accepted principles and its results are hardly to be relied upon.

The sample leaving the last cutter passes through a sample grinder, and thence directly to a Jones sampler. This arrangement is also faulty, inasmuch as the ore should be mixed after crushing and dumped in some quantity, at intervals, over the Jones sampler. The ore is fed from the stock bin to a Hadfield 14 inch x 24 inch rolls operated by two driving pulleys, 36 inch x 8 inch, with a flange next to the roll. This is a very poor machine for coarse crushing as the belts and pulleys are entirely too small, incapable of transmitting the power necessary to crush rough ore to the full capacity of such rolls, and such small wheels cannot, of course, contain the momentum that is required when coarse ore or uncrushable material enters the rolls. In such a case it would mean a choke-down and throwing off of the belts. On the same floor there is another rolls of the same pattern and size for crushing middlings. One elevator takes the crushed ore to the top of the mill building, where it passes through four revolving screens of 12, 8, 4 and 2 mm. apertures. The screened ore passes to the jigs and the finer material to three Culver hydraulic classifiers, which give three jigging products. The stream leaving these classifiers passes into a V shaped settling tank, 60 feet long, about 2 feet wide at the smaller end and 10 feet at the larger, making eight products, which pass to nine Luhrig type vanners immediately below, the tailings from which pass to waste, while the headings are re-worked on nine other similar vanners on the lower floor. The tailings from the finishing vanners are elevated in four sizes and again passed over the upper, or first, vanners. Below the vanners there are the usual settling pits for the concentrates. There are also water tight bins below the jigs for receiving the jig concentrates. A 24 inch Pelton wheel is being placed on the lower vanner floor to operate a dynamo for lighting the mill, and probably to operate also a low tension machine for the magnetic separators, although this part of the plant had not been decided upon. The concentrating machinery is operated by a 60 inch Pelton wheel on the screen floor. The discharge water from this wheel furnishes the supply of wash water for the jigs and vanners.

A very well arranged assay office has been built in connection with the mill, and, as previously noted, the mill is very well constructed, and it is well lighted, as, of course, all mills on flat sites are, and conveniently arranged. The roll capacity is, however, far too small for treating 4 tons per hour, to do good work. At least two other rolls should be added for grinding middlings.

The modified Luhrig vanners are probably the principal problems to be solved in this mill. The regular Luhrig vanners have done good work on the Pierrefittes ores in the French Pyrenees and at other points in Europe, but so far as I know they are new in this country, and for slimes, at least, cannot compare with the Frue vanner, and on fine sands are inferior to the Wilfley type of table. The modified vanners in the Brockman mill, however, are operated by eccentrics and have neither spring, bump, nor any sort of concussion.

When mixed zinc-lead-siderite ores occur, 5 compartment jigs are essential to a close saving of zinc, and none should be less than 4-compartment. I was, therefore, disappointed to find no 5-compartment jigs in this new (1905) mill, but very few 4-compartment jigs, and practically no fine grinding machinery.

LUCKY JIM MINE.

This property, situated at Bear Lake is owned by Mr. G. W. Hughes and associate. It consists of 12 mining claims and fractions, aggregating about 350 acres. The Lucky Jim vein was discovered in 1892 and worked irregularly for some years for silver-lead ore; during the years 1896–1899 concentrating ore amounting to 5,641 tons was produced, from which 1,600 tons of zinc blende averaging 50% zinc was sorted out. The remainder, a zinc-lead product was sold to the owners of the Pilot Bay concentrating and smelting works. The 1,600 tons of zinc ore assaying about 6 ozs. of silver per ton, 3% lead, and 50% zinc, was shipped partly to Antwerp and partly to the Fry process works on the Manchester Ship Canal, England, a special freight of $14.50 per ton having been secured from the mine to those works, but unfortunately the works, the process and its inventor all came to grief about the time the ore arrived in England and the shippers gained nothing but experience by the transaction.

During 1901–2 the property was shut down, but it was reopened in 1903 by Mr. G. W. Hughes, the present owner, who has declared dividends of $100,000 as the result of zinc ore shipments during 1904 and 1905.

The production under Mr. Hughes' management up to the end of 1905 amounted to 5345 tons of zinc blende averaging 54% zinc, a small portion of the tonnage being concentrates from a trial shipment to the Payne concentrator at Sandon.

Development.—The property is developed by five tunnels, the uppermost worked exclusively for silver-lead ore and now abandoned, being situated at an elevation of 4,551 feet above sea level. The Slide tunnel, so called because its portal is situated on the side of the gulch, in the track of a large snow slide, is at an elevation of 4,474 feet. A second tunnel (Safety tunnel) was driven in at this same elevation from a sheltered point on the side of the mountain and connected with the workings of the Slide tunnel, so that the men can enter or leave the mine in safety when slides are running in the gulch.

Two tunnels, one on the east and one on the west side of the gulch, have their portals at an elevation of 4,366 feet. These tunnels (called No.2) are connected with the Kaslo & Slocan Railway by a gravity tram 1,300 feet long, with a fall of 830 feet. These tunnels together with the branch levels, aggregate 3,000 feet of drifts.

Geological.—The Lucky Jim ore deposit differs so much from the general run of the Slocan vein series, that a short note on the geological conditions is necessary to a clear understanding of the ore occurrence at this very interesting mine. In a word the ore is found in a zone of limestone and calcareous slate where penetrated by fissures, and invariably in the purer crystalline limestone of the zone. The foot wall of the limestone zone is a hard dark-green fissile slate, more or less pyritiferous near the plane of contact with the limestone. Impure quartzite beds occur in the foot wall slates, but not in the vicinity of the ore deposits. The hanging wall country, as seen in No. 2 tunnel, appears to be the average graphitic slate of the Slocan series. The foot and hanging country is separated by about 100 feet of calcareous slates, limestone, etc., that make up what I shall call the limestone zone. The pay ore occurs in chimney-like columns in the purer limestone, invariably along some line of fissuring, or extending along the fissure in vein-like form, where limestone forms one or both of the fissure walls. The fissuring, like most of the Slocan series, is greatest at the present surface and becomes less in depth, the minor fissures often disappearing in less than 100 feet from surface. These latter, however, are confined to the limestone zone, and are more properly called incipient fissures. The key to this ore deposit is, however, the east-west fissures crossing the strike of the limestone zone and this fact should not be lost sight of in prospecting at the Lucky Jim, or other properties along the strike of the limestone zone to the south.

The mine.—Referring to the plan of the workings (Plate XXXI), the Safety tunnel follows very closely the strike of the slates from its portal to the turn. The main fissure of the mine was intersected at this turn and followed westerly through the slates until the contact plane was reached. This fissure is a clear break in the slates, averages about 2 feet in width and stands vertical. I could not observe any mineralization in the slates, the fissure being for the most part open, though here and there blocked by crushed slate. The drift followed the open fissure westerly until the limestone zone was reached, where ore was at once discovered in chimney-like mass which was followed up to surface along the contact and stoped out, producing considerable lead ore. This upper stope, it will be noticed on the plan, extends near the surface from the main fissure across the Slide tunnel, and connects with one of four subsidiary fissures that occur in the west drift of this tunnel.

The Slide tunnel enters slate at its portal and continues in same until the limestone is reached. At this point a drift runs back in a northwesterly direction along the foot wall contact plane, intersecting 4 parallel fissures in the limestone. These are small and usually tight, incipient fissures, which do not extend into the slate foot wall, although they carry ore in the limestone, and in some places quite good bunches of ore. The first fissure is small and tight on the levels, but has been followed up by a stope which connects with the surface stope previously noted. The second fissure is also very tight on the drift, but as followed west, opened out, and from 20 to 32 feet west of the drift contained good concentrating ore for a width of 12 feet, mostly a high class blende. The third fissure intersected in this side drift is very small and apparently unimportant, while the fourth and last is the strongest of all. It has been followed through limestone 60 feet from the foot wall, at which point the hanging wall slates are met with. Galena with some zinc blende occurs in this fissure for a length of 50 feet, and has been stoped in one place up to surface. These four fissures occur at intervals of 15 feet making a definite though incipient fissuring or sheeted zone, confined however to the limestone which here has a proven thickness of 50 feet. A fifth fissure occurs at the junction of this drift with the main Slide tunnel and the sixth and main fissure, previously traced through the Safety tunnel, is intersected at a distance of 110 feet from the portal of the Slide tunnel.

The main ore chimney on the tunnel horizon has an elliptical shape measuring 50 feet along the fissure by about 30 feet greatest width. The sides of the stope show some galena and considerable blende disseminated in limestone, all of which would pay well to concentrate. A winze connects with No. 2 tunnel 100 feet below, and some stoping has been conducted around the winze. The workings were, however, filled with concentrating ore and inaccessible. It looks, as if the high grade lead and zinc ore had been stoped, in part, leaving the concentrating ore to be removed later. Referring to section A B (Plate XXXI) it will be seen that this ore chimney, as represented by the stope, commences at surface with a thickness of about 6 feet, swelling to 30 feet on the Slide tunnel level. The limestone is also thin at surface, about 20 feet, while at the Slide tunnel it is 50 feet and on No. 2 tunnel about 30 feet. This ore deposit is undoubtedly a replacement of the pure semi-crystalline limestone, occurring near the foot wall of the limestone zone, with high grade lead ore near the surface, followed by lead and zinc ore of considerable purity, and, as will presently be shown, a considerable development of pyrites in depth, associated with a very fair grade of zinc ore.

This fissure I have previously referred to as the main fissure, because of its great length in the slate and limestone, and furthermore because a winze was sunk on it to the No. 4 tunnel, where it is very well defined and carries ore on its walls in the limestone. The rich ore has probably been very carefully stoped out from this big chimney, though there are no doubt, many thousand tons of good concentrating ore to be obtained by further working around the periphery. At least, very good blende and galena ore, mixed with limestone and a little slate, can be seen around the workings on and above No. 2 tunnel level, while below that point the chimney is filled with broken ore, which is stored there, until arrangements can be made for its concentration.

There are two adits on the horizon of No. 2 tunnel. I shall first describe the one driven to intersect the ore chimney, last described.

This tunnel starts in on the southern side of the gulch and intersects the limestone-slate contact 200 feet from its portal, after passing through slate for that distance. Near the contact the slate contains quite a large development of scattered pyrite crystals, which may be said to extend for about 20 feet back from the contact. The pyrites is in form of cubes, and is best developed close to the contact. Passing inward along the tunnel, at a distance of 50 feet from the contact the first fissure occurs. A drift has been opened 15 feet to the east and a raise put up some distance. This fissure is well defined here, and would appear to correspond with the most northern of the Slide tunnel series, which has been there drifted on for a length of 60 feet. Where cut on No. 4 tunnel, several good bunches of blende occur in the fissure and iron pyrites is somewhat plentiful in the limey rock adjacent to it.

On the west side of the tunnel at this point a drift was pushed in 10 feet on a tight fissure. The limestone in this neighbourhood is somewhat massive and dark coloured, while 18 feet further in along the tunnel a very distinct open fissure crosses, on the walls of which about an inch of calcite is found. The space between the calcite on the walls varies from one-half to two inches. It is quite probable that this is a post-mineralization fissure. There is no change from this point until the main fissure of the mine is intersected at 380 feet from the tunnel portal, along which the tunnel turns to the east and connects with the raise which passes up through the big ore chimney now stored with broken ore, as previously referred to. This main fissure has been opened for about 15 feet east into the slate foot wall, on which the raise was started, and, so far as I can determine, it closely follows this slate foot wall until the winze is reached (Section A B, Plate XXXI). In the face of the east cross-cut in the slate foot wall on No. 2, the fissure appears small and tight, and would scarcely be recognized as the main fissure which passes so persistently through the slates on the Safety tunnel horizon. In the westerly direction, however, the fissure is very strong and has been followed for a distance of 160 feet, for much of the way through calcareous slate and bands of impure limestone, ending in the typical black Slocan slates, where the fissure is again practically closed. Ore occurs for a considerable distance between the points indicated, but to be more precise, at the foot of the main raise on the foot wall there is 3 feet of very fair blende next the fissure, with 5 feet of rather light coloured pyrites, associated with a little blende, followed by strong developments of calcite which continue southerly in the limestone to the next fissure. A third parallel fissure, small and tight, occurs at the bend of the level, along which no particular mineralization can be noted.

Passing westerly along the main fissure, the No. 2 tunnel was driven through the fissure and extended on its last course 40 feet beyond it, ending in black Slocan slates. A raise was also put up in the main fissure, at this place, but was in no condition for examination. The ore in the fissure, however, gives the same section as at the foot wall raise, 3 feet of blende and about 5 feet of mixed pyrites and blende. The pyrites appears rather massive, and some of it looks like pyrrhotite, but on the whole, it would make good concentrating ore. Continuing westward along the main fissure, toward the hanging wall of the limestone zone, the drift passes mostly through slate from the point of intersection of the main tunnel to the winze, though in this slate there are a few small beds of impure limestone. The left hand side of the drift is deeply marked with striae, showing movement toward the hanging wall at about 40 degrees dip. Limestone again occurs at the winze for a width of 15 feet, and in it there is a very fair development of pyrites, zinc blende and galena, the blende being more abundant toward the hanging, and indeed extends back along the northern side of the drift for 14 feet from the winze, the blende being of the usual high grade character of the Lucky Jim ore. This winze has not been unwatered since Mr. Hughes purchased the property, but he informed me the drift inside of the winze was full of good concentrating ore when he took hold of the property and he believes this ore was taken out of the winze, though it is just as likely to have been obtained from the raise. The ore (he stated) contained considerable galena and was a first class concentrating ore. Beyond the winze the drift is entirely in slate, and ends as previously noted, in dark Slocan slates, softer considerably than those passed through in other portions of the tunnel. Returning to the main raise on the foot wall, it will be noted that the level after passing through the sheeted zone, previously described, makes a southerly bend and enters the slate. Passing onward for considerable distance, the level was very hurriedly examined and the face found to be in slate, but near the face some impure limestone occurs. A westerly cross-cut passes through this band in the slate, and one to the east also ends in slate. One or two cross fissures occur in this drift, but no mineralization of moment was observed. From one fissure quite a stream of water was issuing, giving a black deposit, probably manganiferous.

While this drift south of the main fissure was not carefully examined, yet the irregular occurrence of the limestone and the fact that No. 2 tunnel, after intersecting the main fissure, passed through the limestone and into the slate, that between this point and the winze in the main fissure another slate mass exists, it becomes evident that the limestone is of irregular occurrence and is probably best understood, in part at least, as an accretion or segregation of limestone in the slates, the larger mass following closely one particular stratum of slate now represented as the foot wall; in this limestone segregation mineral has been deposited in the main fissures of certain sheeted zones. The strike of the limestone zone at the surface can be followed for about 250 feet, when it passes under wash and debris and is lost from sight, but again crops on the point of the hill almost opposite the portal of No. 2 tunnel east. Here an open cut was made and considerable lead and zinc ore shipped therefrom. In the sides of this open cut three well pronounced fissures are observable, each one well mineralized.

Tunnel No. 2 west passed in through hard slate, which on nearing the ore deposit is pyritiferous and the slate greenish, but on the tunnel horizon I was not able to observe any of the fissuring, the ore having been completely stoped out, one stope taken out below the tunnel and the place filled with water. I was informed, however, that a winze was sunk about 70 feet and followed ore for the entire distance, but at the bottom the ore was narrowing and following a distinct fissure. I believe the principal shipments from the Lucky Jim, during 1905, came from this chimney, which roughly measures 65 feet in length by 35 feet greatest width, having the shape of a flat ellipse. The ore is very clean zinc blende, containing only a few stringers of iron pyrites, easily sorted out. Some of the crystalline blende has a resinous colour and occurs in very large crystals. Very little hand sorting is required, as the ore is singularly free from waste or impurity, whole car-loads running 54% zinc as broken down in the stope.

No. 2 tunnel west, on passing through the slate-limestone contact plane, broke into solid zinc ore and followed along the basset edge of a very rough jagged slate, forming the northern boundary of the limestone and the ore chimney, which are practically coterminous (see Plate XXXI, Plan and Section C D), while the face of the drift is in slate which terminates the limestone in that direction.* The slate is of dark green colour and pyritiferous. Here we find the curious phenomena of a pure crystalline limestone apparently interbedded with the slate, suddenly and abruptly terminating against a jagged slate wall, being as it were cut off on its strike. This could be more easily explained as due to faulting, but no fault fissure could be observed along the northern boundary of the ore deposit, while the limestone 2 inches from the slate assayed 53% CaO with no silica or magnesia present. On the south side of the chimney the ore has been stoped back into the crystal-

*The hatching indicating limestone in Plate XXXI is purposely shown unconformable with the slate, but not necessarily at any particular angle.

line limestone, while a drift passing almost at right angles to the general strike of the formation, shows the limestone zone to be 60 feet thick at a point 70 feet south of the center of the ore chimney.

The shipping ore on the tunnel horizon and above is stoped out, and the workings in places are filled with concentrating ore, while the stope below the level was full of water, hence no thorough or satisfactory examination of the ore chimney could be made. However, it looks to me as if the limestone in which this ore chimney occurs is a sort of segregated deposit, or possibly a cavity in the slates filled with pure crystalline limestone. The limestone enclosing the other large chimney in No. 2 tunnel east appears to conform to the strike of the slates, at least adjacent to the chimney. It is also crystalline and a sample taken at the foot of the main raise, south side, gave on analysis CaO 53.4%, SiO_2 0.8%, MgO nil.

The Lucky Jim ore deposits occur in a limestone zone in part interbedded with the slates. This zone consists of crystalline limestone of great purity, calcareous slates and dark impure limestone bands which in some places follow the strike of the slates and in other places are of irregular form, more particularly the crystalline limestone in which the two developed ore chimneys occur. The ore deposits occur in association with a system of vertical fissures crossing the zone at about right angles to its strike. The fissuring is best and largest near the surface and does not continue very strong in depth. Future prospecting should therefore take the form of developing the cross fissures at a shallow or moderate depth in the limestone zone. Moreover the shallow deposits carry galena and very clean blende, while in the deepest workings a fine grained pyrites and pyrrhotite occur in quantity, mixed with the blende, and hence the deep ores will require a concentration mill and magnetic separator to handle them effectively. The Lucky Jim mine should produce large quantities of concentrating ore as well as the high grade zinc for which it is famous. Further development along the strike of the limestone will in the fissured zones undoubtedly result in the discovery of other deposits of high grade blende.

MOUNTAIN CHIEF GROUP.

The Mountain Chief group of mining claims is situated on the south side of Carpenter creek, about 1½ miles from New Denver, over an excellent wagon road to within ¼ mile of the principal tunnels. A good packtrail connects the tunnels with the wagon road. The property has not been worked for several years, and no data could be obtained regarding its output or extent.

The first point visited was an open cut in which the vein had been exposed for a length of 50 feet, bearing north 25 deg. east, and dipping southerly, or into the mountain. The vein was 3 feet wide for this distance, and contained from one to 1½ feet of zinc blende, mixed with quartz on the hanging wall, and mostly quartz material on the foot. The vein exhibited ribbon structure, and the short lens made a very fair showing for a surface outcrop. The enclosing rock, however, is an extremely hard, light coloured slate, not, I think, very favourable for the production of ore. About 60 feet below the open cut a cross-cut was driven in to intersect this vein, which is exposed in the cross-cut about 18 inches wide, consisting of a stringer of quartz on the hanging and foot walls; in the central portion it is hard slate, the whole highly silicified and extremely hard. A raise was put up at the face of the tunnel for about 20 feet, but no pay ore was anywhere visible.

Proceeding eastward from this tunnel and passing into a side gulch, or draw, another tunnel (No. 3, on the plan) is found, which intersects the vein at 60 feet from its portal. This vein has been stoped west of the tunnel for a distance of about 80 feet, continuing to surface, or rather to a short tunnel (No. 4) about 60 feet above. No ore could be observed in the face of the workings, the productive vein matter having been very thoroughly stoped out. The mouth of this tunnel exposes a fault, showing considerable movement. The gulch follows the strike of this fault. East of the lens, which has been stoped out on the 60 foot tunnel (No. 3) above described, the ore lens pinches up and the vein is probably cut off by the fault, but as the workings at the fault were not accessible to complete examination, this fact could not be determined. About 80 feet below No. 3 tunnel and on the east side of the gulch or draw, a tunnel (No. 5) was started in massive quartzite and extended for 250 feet into the mountain to intersect the vein showing on the north side of the gulch and fault. This tunnel penetrated extremely hard rock, and a few irregular fissures were encountered, but nothing indicative of a mineral vein. As near as I could judge, this tunnel should have intersected the vein previously mentioned. About 150 feet above this tunnel a vein has been developed by No. 6 tunnel and drifts, and quite a little stoping has been done. About 30 tons of ore was sacked on one of the dumps from which a sample (No. 42) was taken by Mr. Gardé, which assayed: silver 26.1 ozs., lead 5.1%, zinc 31%. The vein, in these workings, according to Mr. Gardé, appears quite regular but small. Sample No. 41 was taken in No. 7 tunnel. It assayed: silver 58.5 ozs., lead 10.7%, zinc 26.6%. I believe the same condition of affairs exists here as that described at the first tunnel examined; at any rate, the 250-feet quartzite tunnel (No. 5) below it should have intersected this vein, but it has given negative results. The two samples taken will give some idea of the grade of the ore. If this vein can be traced into the darker and softer slates of the Slocan series, as it probably can along its strike, further up the valley, I would consider it a good prospect, but as the vein extends in depth into the extremely silicious slates and quartzite, previously described, it becomes barren and practically disappears.

It looks to me as if tunnels 2, 3 and 4 exposed merely the roots of an ore vein, the upper portions of which had been eroded during the formation of the valley of Carpenter creek.

HARTNEY GROUP OF MINES.

These mines are situated on Silver Mountain, about 4 miles by good wagon road from New Denver, and at an elevation of 2,320 feet above Slocan Lake.

The group consists of 8 mining claims, 242 acres, but only one vein, the Hartney, has been developed. It was discovered in 1897 and worked for silver-lead ore about 5 years. During the latter part of 1905 an effort was made to market zinc ores and to ship the crude zinc-lead ore of this property, but not with much success, as the price offered was too low.

The Hartney vein has been developed by 6 tunnels, mostly on the vein, varying from 110 to 570 feet in length and aggregating 1,500 feet, with 510 feet of connecting raises and winzes. The vein averages about 18 inches, the pay streak about 4 inches. The lowest working has reached a depth of 510 feet below the outcrop.

The property has produced 180 tons of high grade silver-lead ore, and there was 160 tons of mixed zinc-lead ore, rich in silver, ready for shipment at the time of the examination (Sept. 27, 1905), and about 300 tons of similar ore stored in the stopes. The average grade of the ore as mined is said to be 40 ozs. silver per ton, 15% lead, and 30% zinc. The ore haul from the mine to the wharf at New Denver, costs $2.50 per ton.

The mine is developed by tunnels. No. 1, or the lowest, has not been started. No. 2, started as a cross-cut, had not reached the vein. No. 3 tunnel intersects the vein at the end of a cross-cut 105 feet long, from which point the drift extends on the vein 570 feet. The enclosing rock is an extremely hard slate, which becomes for the last 100 feet of the drift quite silicious, and the vein slightly changes its strike in this extremely hard ground, while the dip very nearly approaches the vertical; in other portions of the vein the dip is about 60 degrees. At 106 feet from the face a small stope shows the vein to be 10 inches wide, with very regular walls, and a pay streak of but 3 inches of zinc and galena, from which sample No. 43 was taken. This assayed: silver 77.7 ozs., lead 39.7%, zinc 17.9%. This small lens did not appear on the level below, and it looks as if it might open out still wider as the stope is advanced upward. Between this latter stope and the mouth of the tunnel, another stope has been carried up to the level above.

The face of No. 4 tunnel is similar to No. 3, the vein being reduced to a simple 3 inch gouge of crushed slate between very regular and extremely hard walls, showing evidence of movement. The vein here is also practically vertical for a length of 100 feet, as in the level below. So far as I can see, all the ore that would pay to mine has been stoped out above this level. The property has been worked for some time under lease. During part of the term two men produced 120 tons of lead ore in six months, equal to 10 tons of ore per man per month.

The vein looks best at its outcrop and at its shallower depth in the tunnels. All the levels gradually enter harder rock in which the vein becomes both tighter and poorer. The average thickness of the vein matter would not exceed 2 feet, and the pay streak, neglecting the last 100 feet in the two tunnels previously described, would not average over 4 inches. The ore now being mined is almost an intimate mixture of zinc blende and galena, with a little siderite and quartz, though from the shallower workings the ore is said to have been almost exclusively galena.

Owing to its high silver value, it is very probable that the Hartney ore will be difficult to handle, more especially as there is very little gangue associated with the zinc and lead in the ore, and no doubt considerable silver will be associated with the zinc, which, under the present conditions, is practically valueless.

The best pay ore appears to follow the contour of the mountain, extending in the vein only to a moderate depth from surface, or until the silicious slates are met with. Unless a change to a more favourable rock occurs at greater depths, deep development of the vein does not look encouraging and attention is perhaps best given to tracing up and working the vein along the shallow productive zone.

BOSUN MINE.

This property is situated on the east shore of Slocan Lake about one and one half miles south of New Denver. It is owned by the Bosun Mines, Limited, and consists of 8 mining claims and fractions, about 200 acres of mineral lands.

The Bosun, the only developed vein, was discovered in 1896, and was worked for silver-lead ore from 1898 to 1903, and for zinc and lead ore from 1899 to 1903. The mine is developed by 6 tunnels, varying from 300 to 1,000 feet in length, and aggregating 3,600 feet of drifts, and 1,100 feet of connecting winzes, etc. The vein averages about 5 feet in thickness. The pay streak varies from 3 feet down to a mere film and would average about 6 inches. One-third of the area developed on the vein has been stoped, which, however, covers the principal ore shoots. The lower tunnel (No. 1) is in good ore, 250 feet vertical below the outcrop of the vein.

The mine has not been worked for some time and complete figures regarding the production could not be obtained. It appears, however, that the first zinc shipment from the mine went to Antwerp in April 1899. About 20 tons assayed: silver 81.9 ozs., lead 1.8%, zinc 45.6%. The freight on this ore amounted to $21 per ton, afterward reduced to $16. The several shipments to Antwerp amounted to 620 tons. Mr. W. H. Sanford gives the total zinc shipments from the mine up to February 1904, as 1,440 tons averaging: silver 71.3 ozs., lead 1.8%, iron 6.4%, zinc 41.8%, silica 13%, lime 3%, and sulphur 20%. The highest return received on a carload lot was: silver 111 ozs., lead 1.2%, iron 6%, zinc 51.7%. The lowest was: silver 53.3 ozs., lead 2.4%, iron 6%, and zinc 43.4%. The above returns were taken from the books of the company, after mining operations were suspended.

An excellent wagon road extends to the second and third tunnels, while No. 1 tunnel is on the main Denver and Silverton road. Good office buildings and bunk houses exist, and the mine was apparently worked at one time with considerable vigour. Operations were suspended in July, 1903, on account (it is said) of difficulty in marketing the zinc and silver ores produced from the property. Most of the ore shipped from Nos. 1 and 2 tunnels was screened and hand sorted; the waste culled out from the ore in the sorting houses contains a high percentage of zinc, mixed of course with considerable slate, but would pay well to concentrate. There is probably 1,000 tons of such mixed ore on the dumps.

The examination of the mine dumps shows considerable zinc blende and some of the richer portions of them will doubtless pay to turn over and hand sort before concentrating, when a good market is obtained for zinc ore carrying silver.

The lower tunnel is opened out on the Lake Shore road about 150 feet above high water mark, and the ores could be very conveniently delivered on board barges for transfer to concentration or reduction works.

I am informed that the owners of the new concentrator near Roseberry have secured control of the Bosun property, and intend shipping the ore for treatment to its new plant at Brockman, in the near future.

The mine.—On referring to the section (Plate XXXII) it will be seen that the vein over No. 3 tunnel is practically stoped out up to surface, for a distance of 420 feet from the tunnel portal; at 400 feet the stull timbers had fallen out and the level was choked, so that my examination terminated at that point. The mine section, however, shows a solid block extending easterly to the last winze connecting with tunnel No. 4, probably a lean place in the vein. I was accompanied in this examination by a man who held the position of foreman when the mine was in operation, and he informed me that the last 200 feet of No. 3 tunnel showed a good vein along the bottom of the level, averaging 30 inches in width, including an 8 inch streak of galena mixed with considerable zinc blende. The mine section shows that a roof stope extends for 200 feet westerly from the face of the tunnel, which goes to confirm the foreman's statement. This stope is evidently on an ore shoot separate from the main one followed from surface to No. 1 tunnel, and for distinction I will call it the eastern ore shoot.

Some few small pillars are left over the drift in the stoped out zone extending 400 feet from the portal. These pillars show a very irregular distribution of zinc throughout, the fractured slate constituting the vein. The stopes average about 5 feet wide, while the pay streak as seen in the bottom of the level is about 12 inches in width, mostly blende, with one to two inches of galena next the hanging wall. These minerals, however, occur as small lenses in the vein, commencing with say a ½ inch streak, swelling out to 10 or 12 inches in thickness and gradually thinning down to a mere film. The lenses do not occur at regular intervals, and occasionally there is a long stretch of barren vein between them. From the 300 to 400 foot mark I took one large sample in 5 cuts across the pay streak. This is No. 45, which assayed : silver 44.9 ozs., lead 1%, zinc 36.2%. The average thickness sampled was 6 inches.

In going up through the old stopes to No. 4 tunnel, the latter was found to be caved at 300 feet from its portal. For about 200 feet westward from the cave the level is stoped in the roof. For about 100 feet a seam of zinc similar to that shown on the level below occurs close to the hanging wall along the bottom of No. 4 level. It comes in at about 2 inches and in places is about 8 inches wide, showing considerable galena and would average about 3 inches for the entire distance. The walls are very firm and have not yet put any weight of moment on the stulls, which have been so badly placed by the timbermen that many of them have fallen down in the level.

I am informed that the vein in the bottom of the drift runs in for 140 to 150 feet beyond the choke, and is said to average better than outside the choke, and to contain more lead. The lead ore appears to be coming in at the choke and in one place shows about 2½ inches of apparently high grade galena. This level has been stoped close to the mouth and almost up to surface. There are two other tunnels, No. 5 and 6, on the Bosun vein, but they are short and unimportant.

Tunnel No. 2 is stoped near its portal, where an extremely rich ore shoot is said to have been discovered in the early days of the property. From that stope eastward the vein is low grade, until at 300 feet a slight fault heaves it 30 feet south, and then a spur from the vein runs back westerly 50 feet. At this fault and split in the vein the main ore shoot begins and a winze follows it in ore to the intermediate level between No. 1 and No. 2 tunnels and has been stoped in the ends. A raise communicates with No. 3 tunnel 20 feet east of the last mentioned winze. The ground around this raise is mostly stoped, but I sampled some pillars, obtaining sample No. 44, which assayed: silver 66.4 ozs., lead 10.3%, zinc 35.4%. The width sampled was 8 inches. The stope continues for 200 feet east of the raise and up to the third level, being, as previously noted, the main stope and ore shoot of the mine. The bottom of the level has been stoped under-hand for 80 feet east of the fault. At 120 feet one lens cuts out and another occurs 6 feet nearer the foot wall, which has been stoped for a length of 80 feet, making the stopes here 200 feet in length. They terminate easterly in a tight vein, showing scarcely any mineralization. Two or perhaps three small lenses of zinc blende can be seen on the floor of the level east of the under-hand stope, the largest 6 feet long and 18 inches at the thickest place; the blende is of good quality. From the east end of the roof stope to the face of No. 2, the vein is somewhat wavey and enclosed in very hard slate; the vein itself, however, will average 4 feet in width of a crushed and softer slate than the enclosing walls. For the last 100 feet a quartz seam forms the principal portion of the vein and at the face is 10 inches wide, showing a little pyrites and films of siderite. The ground is hard and wet, letting out considerable water. The vein on this level, as a whole, consists of a series of very short lenses, and by far the greater portion of it outside the stopes, is practically unproductive.

Reference to the section (Plate XXXII) will show that No. 2 level should be advanced about 100 feet to reach the centre of the eastern ore shoot developed on No. 3. This is a very important point, for if that shoot extends in depth it will greatly enhance the value of the property.

No. 1 tunnel passes through a long stretch of sand and wash, but on entering solid rock did not pick up the vein until a distance of 750 feet from the portal had been reached. The vein forks at this place (See Plate XXXII). The south fork so far as developed did not show up any pay ore. It probably extends 150 feet further than shown on the plan, ending in a crosscut toward the north vein. The north fork struck pay ore at once, which continued almost to the face and has been partially stoped in the roof, while the floor of the drift for a length of 50 feet, shows the best run of ore I have seen in the mine, and this at a depth of 250 feet below the outcrop (vertical). Moreover the ground is moderately soft and quite favourable for the production of mineral. The outlook here for development in depth is certainly very good.

The roof of the stope and its east face is found 30 feet above the level and 20 feet west of the face of No. 1 tunnel, north branch. The vein here consists of 12 inches of zinc blende, containing galena against the hanging wall and 4 inches of zinc blende in the foot wall, separated by 3 feet of softish slate rock. I sampled this vein at a point where a roll in the hanging wall gave another streak of zinc 4 inches wide, making 20 inches of zinc blende. This was No. 46, which assayed: silver 62.3 ozs., lead 21.2%, zinc 32.8%. This sample did not include any of the slate in the centre of the vein, but was simply the three mineral streaks. Six feet from the face of the stope this lens terminates, but others come in at irregular intervals. Sixty feet above the level and 30 feet west of the main raise the vein is 3 feet wide, mostly crushed slate and soft talc with a 3 inch blende-galena streak on the hanging.

Proceeding up the raise to the intermediate level, the vein in the west face is reduced to a gouge varying from 6 inches to 12 inches thick, but at 100 feet east from the raise, a stope goes up in the roof, averaging 50 feet in length. It continues for 140 feet northerly and ends 15 feet from the north face of the intermediate level. The roof of the stope is said to average 3 inches to 6 inches of zinc on the walls, but was inaccessible. Sample No. 47 taken from the bottom of intermediate level, in two cuts averaging 6 inches of pay streak, assayed: 53.4 ozs. silver, 5.1% lead and 41% zinc.

The vein in the face of No. 1 tunnel, north branch, consists of a 12 inch gouge of soft crushed slate and a little talc on the hanging wall, and is well defined. Water issues from various cracks and joints, depositing quite a little ferric oxide. In the intermediate level lime carbonate is deposited in several places from drop water. The stopes over No. 1 level extend, from within 15 feet of the face, practically to the turn of the main tunnel where the vein forks, as previously noted. Zinc and lead can be seen on the bottom of the drift in several places, and I am informed that it makes out in places to 4 feet in thickness. In all probability one or more lenses extending for a distance of 100 feet will carry pay ore for considerable depth below this tunnel. For the greater part of this distance the hanging wall is extremely regular and stands steeper than in the stope above. There is very little siderite in the vein and what there is occurs mostly in fractures and on joint faces, and seems to be a recent addition to the vein filling.

The Bosun zinc ores contain high silver value; the lead is also high in silver. The vein is small but very regular in width, and fairly so in mineral contents; consequently with a reasonable market for the silver-zinc ore, as well as the silver-lead ore, this mine should make good profits, if properly opened up and economically managed.

LIPTON GROUP.

This prospect is situated on Trout Creek, which empties into Slocan Lake about one mile from its head. The Lipton claim is 7 miles from the lake and is said by one of the owners to contain a strong vein averaging 12 feet in width. The pay streak is 16 inches wide from which about 100 lbs. of ore was taken by the owners. Sample No. 48, from this lot was taken at New Denver, which assayed: silver 18.2 ozs., lead 36.7%, zinc 32.6%. The property was not visited by the Commission.

STANDARD GROUP.

This property is situated on Four Mile Creek about two miles from the town of Silverton, and about midway between the Emily Edith and the Alpha claims. The lower tunnel (No. 3) is about 2,000 feet above New Denver, or 3,700 feet above sea level. This tunnel is now being advanced under contract and has reached a distance of about 250 feet from its portal (October 2, 1905). The first 100 feet of this distance is in wash and where the rock first is found it is extremely soft and considerably faulted. The enclosing rock of the vein is black slate. The vein structure is of the usual lenticular type and quite irregular as to the length and thickness of the lenses. The ores present in the vein are galena, usually adjacent to the hanging wall, blende, calcite and quartz toward the foot wall. About 40 feet from the face considerable quartz occurs in the vein toward the foot wall side and continues to the face of the tunnel, averaging about 3 feet in thickness. Resting on the quartz is calcite and a little galena, 4 inches to 6 inches in thickness, adjoining a soft gouge on the hanging wall, which is here crushed and contorted slate.

The whole vein in No. 3 tunnel shows evidence of considerable movement since the deposition of the ore, blocks of which occur in the vein in an almost rounded condition, showing the striae of movement under pressure. Hence it is a mooted question if this vein is in place, or if the tunnel has reached the solid mountain formation.

South of the face 100 feet a raise connects with No. 2 tunnel, 81 feet vertical above, or 140 feet on the dip of the vein. Going up this raise the pay streak was found rather thin at the roof of the level and continued so until the intermediate level was reached about 35 feet above the lower tunnel. At this place the vein made almost a right angle fold, extending back flat into the foot wall about 35 feet (Plate XXXIII, Fig. 4). The raise continuing on its course and passing into the overlying black slate was finally connected with a winze sunk in the vein from No. 2 tunnel. The cross-cut, however, at the intermediate level passing into the foot, followed the vein and opened up a very good deposit of argentiferous galena, mixed with a little blende. The drift along the hanging at the commencement of the bend in the vein, now called the intermediate, level has advanced about 20 feet in a very loose and open vein showing little, if any, mineral. From the openings and cavities in the vein it is evident that the principal movement occurred along the plane of the present drift, and it is interesting to note that descending solutions have carried down lime, depositing calcite in mammillary form in the cavities of the vein. From the point where the vein commences to straighten up above the fold, to No. 2 tunnel there is a very good showing of argentiferous galena averaging about 8 inches in width, visible in both ends of the winze. There is also considerable zinc lying under the galena, probably a streak averaging 12 inches in width, but much more regular than the galena.

I should judge from appearances that there is a very good bunch of ore between the intermediate level and No. 2 tunnel, but the zinc is very much mixed with quartz, calcite and porphyritic material and will require concentration. The lead ore can, however, be hand sorted.

There appears to be a porphyry dike crossing between the head of the winze and the portal of No. 2 tunnel, while the drift after passing northerly through the dike immediately reaches pay ore. The level was advanced northerly on the vein for about 300 feet, and for the last 100 feet there is a silicious vein of zinky ore that would probably contain 20% blende and average one foot in width. The vein in the face is, however, pinched and consists of about 2 inches of porphyritic vein filling, lying on a silicious slate, the hanging being the polished black slate seen in several places to form the hanging wall of this vein both on No. 2 and on the lower tunnels. About 80 feet from the face of the tunnel a cross-cut into the hanging shows up a zinc seam, opening in places to a width of 2 feet, but as it was only visible in the bottom of the level, it was very difficult to examine. On the dump, however, about 60 tons of zinky lead ore had been picked out from the galena shipping ore. The owners had recently taken a large sample from this heap, and from the rejected portion of this sample No. 50 was taken, which assayed: silver 19.4 ozs.; lead 6.1%, zinc 34.8%.

The Standard vein is said to be the same as the Alpha, or a continuation of the Alpha, the workings of which can be seen on the mountain above. The Alpha is said to have been a large shipper for this district. The very steep mountain road from Four Mile Creek was built by the Alpha company in 1893, and it is said about 3,000 tons has been shipped over it. The end of the road is connected with the lower tunnel of the Alpha by a 1,700 feet gravity tram.

As the Alpha property has not been operated for several years, it was not examined. I am, however, credibly informed that the vein continues good in the two main tunnels, and that work was suspended on account of personal disagreement between the owners. The Emily Edith, a well developed property, is situated immediately below the Standard group.

The high grade lead of the Standard mine carries some grey copper. The zinc-lead ore rejected from the shipments now lying on the dump of No. 2 shows in places a little chalcopyrite. The Standard mine, while only in the prospect stage, gives promise of developing into a good shipping property, both for lead and zinc ores.

WAKEFIELD MINES.

These mines are located on Four Mile Creek, about 3 miles from Silverton by good wagon road to the mill, and then by very steep mountain trail to the mine, situated at an altitude of 6,000 feet above sea level.

The property consists of 8 mining claims owned by the Wakefield Mines, Limited, of London, England. This company ceased operations in 1902 and no complete record of the early history or production of the mine is, at this time available. I am indebted to Messrs. Cross & Co., of Silverton, for the following statistics:

Prior to 1902 the owners worked the property, and received in net smelter returns for ores shipped $93,031.69—no weights or assay values obtainable. From May 1st, 1902, to November 1904, the property was leased to the Anglo-Slocan Syndicate, Ltd., during which time about 1,400 tons of lead concentrates and 400 tons of zinc ore were shipped, but no records are available of the exact weight or assay value of the shipments. I quote the following from a letter of Mr. Reginald Leake, dated Feb. 20, 1904,—"During the four months we ran last summer, we made about 250 tons of zinc concentrates, running on the average 45% zinc, 44 ozs. silver, 5% lead, and 3% iron, the average feed carrying about 14% zinc....... We occasionally run across (always near the surface) rich argentiferous galena intermixed with blende, in which the blende carries most of the silver value. Nearly all the calcite at the mine carries zinc in small quantities (which it would pay to concentrate under more propitious circumstances). We expect this year to make at least three times the quantity we did last, as everything is ready for an early start."

At the time of my visit (Sept. 30, 1905) the Wakefield mines were being worked under lease by Mr. William Hunter, who had shipped 23.6 tons of ore assaying: silver 85.1 ozs. per ton, lead 52%, zinc 10.9%, which netted about $54 per ton.

The Mine.—The mine is developed by seven tunnels aggregating about 10,000 feet of drifting on the vein (see Plate XXXIV for plan of the workings). Nos. 6 and 7 tunnels are connected with the head house of the main aerial tram by a short tramway, while No. 2, the principal working tunnel, is connected with the head of this tramway by a short aerial tram spanning a deep draw on the west side of the mountain. The facilities for getting ore to the mill are therefore exceptionally good.

No. 2 tunnel is about 6,000 feet above sea level. All the upper workings are now operated through No. 2 tunnel, which appears to have opened up the vein just below its most productive zone. The Wakefield vein differs from anything I had previously seen in the Slocan region. It has the general appearance of a bedded deposit, the slate foot and hanging conforming to the dip and strike of the normal vein, at least in the mine workings. Very little could be seen on surface on account of a recent snow fall and bad weather.

Galena where present in the vein is invariably associated with calcite, which occurs in very large lenses, sometimes reaching a thickness of 20 feet. Fig. 1, Plate XXXIII, shows the vein of normal composition on No. 2 tunnel consisting of four distinct bands of white calcite separated by films of black clay, which also in places slightly penetrates the cleavage planes of the crystals or coats their faces with a dark film (See Plate XXXVIII). Some galena occurs at the hanging wall and in other places zinc blende. These minerals follow the bedding planes and in part replace the calcite when they occur in quantity. It would thus appear that the pay ores are almost exclusively confined to the calcite lenses in the vein. Between the calcite lenses the vein consists of crushed slate showing considerable movement and having calcite seams, veinlets and irregular stringers throughout the mass. On No. 2 tunnel for the first 200 feet the vein is probably not over 3 feet wide. It is almost entirely crushed slate and gouge; then calcite comes in and a very large stope occurs, in the top portions of which 700 tons of argentiferous galena was taken out in the early days. The vein here is rather flat, the dip averaging about 10 degrees. The average dip of the whole vein I should estimate at about 15 degrees, with an average thickness of 6 feet, from No. 3 tunnel up.

The face of No. 2 tunnel could not be examined on account of foul air, but I am informed by Mr. Hunter that there is a good stope of zinc ore about 800 feet from the portal. Going up through a raise to No. 1 tunnel there are several stopes, some of which showed considerable zinc blende associated with a sprinkling of galena. Several of the pillars left in these stopes would pay to work and are now being operated in part by the lessee of the property. I took sample No. 49, here, this being one foot of zinc ore in three cuts, two on one side and one on the other side of a cross-cut going through a pillar toward the hanging wall. It assayed: silver 10.6 ozs., lead 7.5%, zinc 38%, but owing to the irregularity of the pillars and the prevalence of stopes not shown on the map, it is impossible to make any accurate estimate of the amount of ore available without constructing a stope map. Passing up through another raise to a tunnel above No. 1, the vein was opened in several places for a width of about 8 feet, mostly calcite with a parting of galena near the hanging. A little blende could be seen in many places, also close to the hanging wall.

It was distinctly noted that as the levels advanced into the mountain the vein became smaller, the enclosing rock and vein filling much harder, and the calcite more silicious than in the shallower workings.

The main galena stope was carried down to the level of No. 3 tunnel, but the shoot split a short distance below No. 2. There is, however, a little galena showing in the direct course of the shoot, and considerable calcite. Some work was done here during the early part of 1905, and one stope, at least, can be profitably operated.

Figure 2 shows a calcite deposit with a little zinc blende in a roll in the vein.* These rolls are very common, and where they occur in the shallower levels, the vein is usually of great width and calcite the predominant mineral. The pay ores occur almost exclusively in the calcite, filling in these rolls or lenses, 700 tons of lead ore being found in a calcite lens on No. 1 level as previously noted.

No. 4 tunnel is not over 100 feet long, and is unimportant.

No. 5 tunnel is 250 feet long. The vein averages about 4 feet in width mostly calcite, which, however, is mixed with quartz and toward the face is extremely silicious. There are no stopes on this level and it is quite apparent the vein is narrowing up and becoming tighter. Near the mouth of the tunnel a raise connects from No. 6 tunnel. About midway in this raise a stope has been opened out in a vein about 3 feet wide, showing zinc blende with a slight sprinkling of galena, for a thickness of 6 inches. At the bottom of this raise, on No. 6 tunnel, some little stoping was also done without showing up anything of value. The vein near the end of No. 6 tunnel, about 600 feet from its portal, and in direct line with the main ore shoot in Nos. 1 and 2 tunnels, is pinched down to 2 feet, and in the face consists almost entirely of quartz with a tight gouge. The cross-cut into the vein and a raise extending up through the vein near its face have not shown up anything of value. At this point the enclosing rock is a dark blocky slate, very hard and silicious, comparing unfavourably with the soft carbonaceous argillites enclosing the vein in the productive parts of the shallower levels.

No. 7 tunnel must have been a great disappointment as it was connected with the head house of the main aerial tram to the mill by a short mine

*The hatching indicating calcite in Fig 1 shows the bedding planes correctly. I have on Fig 2 simply indicated calcite, as the bedding structure is for the most part folded or obliterated.

tramway, and I am informed the management had expected to stope all the vein above this level and send it through the mill. The mountain near the mouth of No. 7 tunnel stands out in a steep ridge, at the northern flank of which the tunnel entered. Here the vein is about 4 feet wide and contained on the hanging wall (judging from the pillars left) from 1 inch to 2 inches of galena of high grade. The first raise goes through to surface, and the ground has been about two-thirds stoped out. Lessees are now (Sept. 30, 1905) gophering the remainder in search of high grade silver-lead ore. About 300 feet from its portal a branch extending south from the main tunnel came through to surface, the vein flattening up at this place. Some little stoping has recently been done, near this second portal of No. 7 tunnel, on the southern flank of the ridge previously referred to. Reference to the map (Plate XXXIV) will show that this branch tunnel is directly below the portal of No. 1 tunnel and here we have the very unique feature of a mineral vein with its outcrop above No. 1 tunnel, dipping out of the mountain about 1,000 feet below its outcrop measured on the dip of the vein.

This vein between this south portal of No. 7 and the first raise has been stoped for a height of probably 60 feet. Proceeding along the tunnel, easterly from the south portal some lead ore of high grade occurs in places, and the lessees are now opening it up. This lead ore can also be traced to the right angle turn in the drift, also a little zinc ore, but both in small quantities. The enclosing rock for this distance is a rather softish black carbonaceous slate with the vein matter in places oxidized. After the turn was made, the drift continued in a northerly direction, and it is difficult to determine if the vein is present in the drift. At the face, however, a raise was put up 50 feet, showing a vein of calcite with slight mineralization coming in at that height. The face of the drift shows a 2 inch gouge and a few inches of mixed quartz and slate.

It will thus be seen that at every level of the mine the vein pinches up or becomes silicious as it is followed toward the heart of the mountain. Near the surface, or say from No. 2 tunnel up, the vein is one of the finest I have seen in the Slocan. It certainly contains the greatest thickness and the purest deposits of calcite that I have previously observed in any vein. The deposits of ore in the shallow levels, although very irregular, appear to have been on the whole of good paying character; hence, the pinching up of this vein as it is followed in depth, or even into the mountain, must, (in part at least,) be due to the fact of its being, to some extent, a bedded vein, possibly limestone recrystallized. It contains good mineralization only at the point of greatest movement. There is abundant evidence of considerable movement along the plane of the vein, both in dip and strike, the former perhaps more easily explained as the descent of the hanging wall side. Numerous veinlets of calcite extend up into the shattered hanging wall, but this movement can scarcely be distinguished in the deeper levels, which is not perhaps remarkable in a slate country where the yielding and crushing walls would, at a shallow depth, absorb all movement.

Active prospecting in the shallow levels of the Wakefield should open up a fair tonnage of concentrating lead-zinc ore, as well as an occasional bunch of high grade ore.

WAKEFIELD MILL.

This mill is situated on the bank of Four Mile Creek, and connected with No. 7 tunnel (3,000 feet vertical above) by an aerial tramway 1 ½ miles long.

The mill is well built and could be easily converted into a first class concentrating plant for the treatment of zinc-lead ores. It is operated by water power and has a capacity of 3 tons per hour.

The machinery consists of one 9 inch x 15 inch rock breaker, 3 sets of rolls, one Huntingdon mill, one 2 compartment jig, three 2 compartment jigs, and two 4 compartment jigs; also 7 Wilfley tables. At the time of my visit the mill had just shut down after a trial test on the Hewitt and Emily Edith zinc-lead ores. Prior to this work,.it was, so far as I can learn, idle about a year.

HEWITT MINES.

This property is situated about three and one-half miles from Silverton. An excellent waggon road leads from Silverton to the lower terminal of an aerial tramway, which connects with No. 3 tunnel. The ore is delivered f.o.b. steamer at Silverton for $2 per ton from the tram terminal.

The Hewitt was discovered in 1892 and was worked vigourously for several years. After remaining idle for some time it was leased to Mr. M. S. Davys of Nelson, who is now operating the property with very satisfactory results. There are two east and west veins and two cross veins, one of which, however, the M vein, is merely a connecting branch between the two principal vein systems. The east and west veins occur chiefly in the Slocan slates, and are most productive in and adjacent to a granitic intrusion which has shattered the slates, opening wide and extensive fissures. The cross veins occur chiefly in the intrusive granite. The ores are silicious, though containing a small percentage of lead in the granite area, and often considerable galena in the shattered slates away from the granite. Quartz is everywhere the predominant mineral, carrying grey copper, ruby silver, galena and zinc blende. The high grades of silver ore are relatively more abundant in the upper workings, while on the lower levels zinc blende is the more abundant ore.

From May 30, 1900 to December 31, 1902, the ore shipments from the property amounted to 2,752.117 tons which averaged silver 77.1 oz., lead 4.8%, zinc 11.6%. This ore realized at the smelters $69,869.70 net. The property was leased to Mr. Davys in January 1904, and up to the time of my visit, (October 20-21, 1905) the shipments under the lease had reached thirty-seven lots, which for their uniform silver tenor, low lead and high zinc, are, I believe of sufficient interest to be covered in detail as typical of the "dry ores" of the Slocan. I might almost say, the ores from the granite as against those from the slate, which I have hereinbefore dealt with exclusively.

ORE SHIPPED FROM HEWITT MINE, SILVERTON, BY M. S. DAVYS.

Lot No.	Weight lbs.	Ozs. Silver per ton	% Lead	% Zinc	Remarks
1	37,356	176.6	8.4	13.1	
2	40,576	127.3	4.8	13.9	
3	38,124	119.2	5.0	14.0	
4	37,647	122.2	3.5	14.8	
5	39,045	107.2	3.5	14.8	
6	39,161	137.2	3.1	15.2	
7	40,710	162.4	2.6	14.0	
8	40,441	131.2	4.0	13.0	
9	40,449	148.1	5.2	15.4	
10	42,727	140.2	5.4	14.8	
11	41,405	110.3	3.0	12.4	
12	47,946	137.2	3.5	10.2	
13	45,134	107.5	3.0	9.5	
14	43,223	135.0	3.4	11.3	
15	40,856	143.2	3.1	9.8	
16	43,743	182.3	3.9	13.3	
17	41,662	169.3	4.3	12.2	
18	40,407	118.2	3.3	9.8	
19	40,199	111.2	3.8	10.1	
20	41,069	187.2	5.1	13.4	
21	42,432	225.4	10.9	14.6	
22	30,987	133.8	13.6	14.8	
23	41,224	78.6	11.9	14.8	No. 4 Level
24	42,099	70.8	11.7	14.7	"
25	42,102	64.2	15.4	12.2	"
26	29,031	70.6	11.7	12.9	"
27	39,640	77.0	14.7	10.3	"
28a	26,429	320.0	10.0	16.4	
28b	11,103	118.2	11.0	12.3	
29	27,348	139.8	7.3	14.1	
30	22,301	165.3	4.7	11.1	
31	39,462	175.3	5.7	14.1	
32	41,353	183.2	6.0	13.1	
33	40,605	135.0	6.8	12.7	
34	41,143	132.2	3.6	13.0	
35	35,452	157.5	4.5	13.9	
36	40,107	183.5	5.3	12.0	
37	38,632	129.2	3.6	9.3	

FROM LORNA DOONE.

Lot No.	Weight lbs.	Ozs. Silver per ton	% Lead	% Zinc	Remarks
1	39,859	36.2	4.5	16.3	Without Sorting.
2	10,962	100.2	10.3	15.5	Sorted.
3	18,233	110.2	11.1	15.0	Sorted.
4	4,189	257.9	14.2	16.0	

The mine.—Perhaps the more prominent and important feature in the mine is the fact that the granite area is the most productive of high grade ores, from which it is a fair inference that the granitic intrusion was, at least, one of the determining causes of the deposition of pay ore. For example all the drifts from No. 1 to No. 5 extend for some distance in a poor and even barren vein in the slates, but where granite forms one or both walls of the vein, pay ore is invariably present; in some cases, however, pay ore occurs in the vicinity of the granite area, while the vein is yet in slate.

The workings of No. 1 tunnel were not visited, as no zinc ore of moment had been discovered therein.

No. 2 tunnel follows an unproductive vein through the slates until a granite foot wall is met with, and here the first stopes are found. Referring to Plate XXXV, illustrating the vein plan, two approximately east and west veins will be observed, called the "North Vein" and the "Main Vein", a vein with southwesterly strike, called the "South Vein", and a short cross vein connecting the south vein with the two east and west veins. The main vein is stoped in the roof, from the point where the granite foot wall begins, up to the cross-cut east of the south vein, with the exception of a few small pillars. From this cross-cut to the next easterly cross-cut, the vein is small and irregular, but is opening out again in the drift going east, and gives promise of an early discovery of pay ore on the main vein. The cross-cut north from this place intersected the north vein on cutting through the granite, but it was poor and continued so in the drift easterly until the granite terminated, when the pay ore came in, and at the time I visited this place the vein consisted of two feet of high grade quartzose ore showing some zinc blende and galena on the hanging and two feet of less mineralized quartz on the foot, the whole face being high grade dry silver ore, said to average about 120 oz. per ton.

Returning to the cross-cut immediately east of the south vein, this passes through granite thirty feet, then turns west forty feet before reaching the north vein, which is deflected somewhat from its regular course. The vein, however, where met with was quite rich and is stoped in the roof for a height of 40 feet. Sample No. 66 was taken across the roof of the stope for a width of four feet. It assayed: silver 36.3 ozs., lead 2.6%, zinc 15.4%.

On leaving the granite wall the vein became poor; the drift continuing on a cross slip, leaving the north vein behind, communicated with the drift on the main vein.

The south vein is developed for two hundred feet southwesterly on its strike, one hundred feet being through granite where it is very productive, particularly from its junction with the M vein to the point where the slate is first met with. From this point to the face the granite formed the hanging wall, and the slate the foot wall of the vein, which dips 65° westerly. The vein in the face consists of fifteen inches of crushed quartz, iron stained and showing considerable oxidized minerals resting on a gouge of crushed black slate ten inches thick, and though it did not contain pay value (Nov. 4, 1905) it looked to me a very promising prospect, more especially as the drift was following up a regular and well defined vein towards the main granite mass to the southwest, in which, or along the flank of which, good deposits of ore should be met with. The south vein for eighty feet northerly from the point where the slate foot wall comes in was stoped for an average height of twenty five feet above the level, the ore being almost four feet wide, carrying high silver value, in a quartz gangue. Some galena occurs with the quartz; also grey copper and ruby silver, with quite a little oxidized matter in the joints and fracture planes of the vein rock.

The M vein connects the south with the main and north veins, and is said to have been very rich. It is now stoped out. Between the main and the north vein, the M vein appears to consist of a network of rich stringers penetrating a decomposed granite, enclosing irregular slate masses. All the granite adjacent to the veins on No. 2 horizon is partially decomposed, the feldspars kaolinized and considerable secondary minerals formed in the granite mass.

The south and M vein on the level of No. 3 tunnel were stoped out and the drifts caved. On referring to the plan of the vein (Plate XXXV) it will be seen that the dip of the south vein has carried it sixty feet further west than on No. 2 level; that the fourth level would probably find this vein on the margin of the granite, and the fifth should reach it entirely in the slates, providing these cross veins extend to that depth, which I believe is rather doubtful.

The main vein in No. 3 level terminates where intersected by the south vein, and the main level extends on the course of the vein sixty feet easterly in hard granite, showing no decomposition; the feldspars are apparently unaltered and the ground mass solid.

A cross-cut is driven northerly through this granite, intersecting the north vein in a lean place, but on drifting easterly on its strike a good ore deposit was met with in the slates, just as in the level above; but while on the second level zinc blende was not plentiful in the vein, on the third level it occurs in quantity mixed with quartz and siderite, making a first class concentrating ore.

In the east face of the drift on the north vein (third level) the vein is eighteen inches wide of brecciated material, mostly zinc blende, cemented by quartz enclosed in rather softish slaty rock. Forty feet west of the face galena and silver ores occur in connection with the brecciated vein material. Seventy-five feet west of the face a raise has been pushed up fifty feet in a vein five feet wide, in which quartz, zinc blende, and siderite occur in the order given, as to their relative abundance. The vein in part presents a brecciated appearance, angular pieces of zinc blende being surrounded by pure white quartz, or again blende and siderite are deposited in irregular wavy layers, with a subsequent development of white quartz. (Plate XXXVI). One hundred feet west of the face the vein is opened out in the foot wall, exposing nine feet of milling ore. Sample No. 68 was taken at this place in two cuts across the vein. It assayed: silver 13.3 ozs., lead 5%, zinc 11.4%. Siderite is at this place more plentiful than the blende. This ore shoot averages about four feet in width for a length of 120 feet. We have previously seen that the vein on the level above contains rich silver ore, and the question arises, where does the zinc come in, and the high silver value falls off. Sample No. 67, taken in the roof of a stope on the main vein, twenty feet below No. 2 tunnel, assayed: silver 190 ozs. lead 12.5%, zinc 24.7%, and would probably indicate that the zinc ore would extend for a height of sixty feet above No. 3.

A portion of the granite-slate area between the M and the south vein on No. 3 level and extending from the main to the north vein, is, as on the level above, heavily mineralized. The ground is very heavy at this place and closely lagged, so the ore could only be seen in the floor of the level, where it is a high grade dry ore, showing but little zinc, and therefore was not sampled. No. 3 tunnel, like No. 2, followed the main vein through slate from surface, but pay ore was not discovered till the slate-granite contact was reached, as shown on plate XXXV.

A cursory examination of the vein plan on No. 3 will show that what is called the "North Vein" is really the main fissure, while the 4th, 5th, and 6th levels show this fact still more clearly.

Opposite the M vein on No. 3 tunnel a winze was sunk from which No. 4 level was opened up by a cross-cut direct to the north vein from the bottom of the winze. This cross-cut passes mostly through crushed slate and soft heavy ground, containing some small veinlets of blende and siderite. There is no apparent development on the main vein, which was not definitely located at any place below the 4th level. Opposite the winze on the 4th level, a raise communicates from No. 6 tunnel on the north vein, draining and ventilating this portion of the mine. No. 5 level is opened from this raise, hereafter called the "Main Raise."

The principal development on the fourth level is on the north vein, which averages about eight feet in width, six feet of which can be stoped for concentrating ore. The vein structure is entirely breccia, country rock enclosed in and cemented by milk white quartz, ramifying through which galena occurs as veinlets, associated with siderite and zinc blende; galena, however, is plentiful. At thirty feet east of the main raise, a raise exposes this vein for a height of fifty feet above No. 4, showing the same vein filling and minerals. The total drifting on the north vein amounts to sixty feet east of the main raise, and forty-five feet west of same. The face of the west drift terminates at a cross course, which if followed southerly should connect with the main vein, as in the levels above. I took a sample of the zinky portion of the vein near the west face, No. 69, which assayed: silver 4.3 ozs., lead 1%, zinc 13%. The vein exposed in No. 4 level is far too irregularly mineralized, and too loose to permit one to obtain more than a rough idea of its value by hand sampling. I therefore took but one sample of the zinky portion of it. No. 5 level extends thirty feet east of the main raise in a vein of brecciated material ten feet wide, showing considerable zinc distributed irregularly through the vein, as well as siderite; galena, however, is less plentiful than in the level above. The minerals occur in the order of relative abundance, as quartz, siderite, zinc blende and galena, quartz being the cementing material of the vein breccia. Sample No. 70 was taken near the east face of the drift, two feet on the foot and two feet on the hanging wall, separated by five feet of poor vein material, not sampled. The sample assayed: silver 16.7 ozs. lead, 4.9%, zinc 12%, which represents the average value of the vein east of the main raise. The west drift extends twenty-seven feet from the main raise, and is rather poor in the face, mostly crushed slate fragments cemented by quartz, but it is doubtful if the full width of the vein is exposed. I believe a portion of it is still in the foot. Considerable soft clay gouge matter occurs in the vein on this level and particularly west of the main raise.

The sixth level is the lowest working in the mine. Opened as a main tunnel, it followed an unproductive vein through the slate until a point was reached sixty feet west of the main raise, when the edge of the ore shoot was met with, showing a good development of zinc blende. Sample No. 73 was taken from two feet of milling ore at this point, assaying: silver 37.1 ozs., lead 8%, zinc 13.6%. Twenty-five feet further east sample No. 72 was taken across four feet of milling ore, assaying silver 23.4 ozs. lead 6.1%, zinc 16.1%. Sample No. 71 was taken twenty-five feet west of the main raise, the pay streak only, averaging 18 inches in thickness and showing considerable galena. It assayed: silver 102.6 ozs., lead 19.4%, zinc 17.2%. The vein in this shoot on the 6th level consists of a breccia of slate, zinc blende, etc., cemented by milk white quartz; siderite, so plentiful in the third and fourth levels, is here in small proportions, and quartz forms three-fourths of the vein filling. The shoot appears to be at least sixty feet long. The drift east of the main raise is closely lagged, and owing to the heavy ground the sets are also spragged, thus it was impossible to examine the vein at any point east of the main raise. Cross-cuts into the foot and hanging near this raise show the mineralized zone to be thirty feet wide, giving the following section on the foot wall: 18 inches of soft clay gouge, next three feet of silicious slate with siderite in the joints and cleavages, next four feet of zinc blende and quartz breccia—the central vein, in which the main raise went up—next following this is four feet of less mineralized vein matter, beyond which extending into the hanging country is crushed slate, clay seams etc., making in all a mineralized and fault zone thirty feet in width. The granite has not been discovered in connection with the vein on the sixth level; in fact the main vein with its granite foot wall has not been found in any of the levels below No. 4, and very little can be seen of it there.

It would therefore look like good, or at least a safe, practice, to trace up the main vein in the granite on No. 4 level, and with the information so obtained locate this vein on No. 5 level. Should the granite prove to be merely an intrusion sheet, not extending deeper than the 4th level, the main vein would in all probability not extend to a great depth, or if so, probably not as a pay vein, because it is dipping away from the granite intrusion. On the other hand, if the granite intrusion continues in depth, the main vein in its vicinity should carry pay ore fully as deep as the north vein, and possibly of better grade.

The Hewitt mine has, practically blocked out, a large tonnage of zinc ore. While this ore is not high in zinc, it must not be forgotten that the gangue is mostly quartz, easily separated by concentration. The high silver value in the veins will probably be shown in high silver in the zinc blende concentrate. When equipped with a modern concentrating plant, this property should make a large and steady output of zinc blende carrying good value in silver.

ENTERPRISE MINE.

Ten Mile Creek empties into Slocan Lake about ten miles south of New Denver. The Enterprise mine is situated on Ten Mile Creek, about eight miles from the lake; the wagon haul from mine to steamer costs $3.50 per ton.

The Enterprise property is owned by the Enterprise B. C. Mines Company, Limited, and consists of two full claims and a fraction, about 97 acres of mining land.

The Enterprise mine was discovered in 1894, and is developed by five tunnels, varying from 120 to 1,050 feet in length, and aggregating 2,920 feet of drifts. The vertical depth from the outcrop to the lowest development on the vein is 750 feet. The vein is in granite, and averages about ten inches in thickness; the greatest observed width is about 36 inches.

The property was operated by the owners for several years, and a concentrating mill of 1½ tons per hour capacity has been erected; for some years, however, the Enterprise mine has been worked under lease. It was not visited by the Commission.

The Enterprise mine has produced 8,215 tons of shipping ore, 2,466 tons being concentrates from the mill and 5,749 tons hand sorted ore.

Included in the tonnage of concentrates is a middling product sold as silver ore, though containing 27.98% zinc, 71.6 ozs. silver and 2% to 4% lead. The general shipments from the property average about 127 ozs. of silver per ton, 19.2% lead, and for the last few years 23.77% zinc.

A sample of the Enterprise .zinc concentrates were obtained and forwarded to Denver for magnetic separation, it assayed: silver 115 ozs., lead 4.8%, zinc 43.70%.

EAST KOOTENAY.

In the East Kootenay only the principal producing mines were visited, namely the Saint Eugene, and the Sullivan Group. These mines are chiefly noted as producers of lead ore, in which respect they rank among the most important of British Columbia, and up to date have not made any shipments of zinc ore. Certain mines of this district which are in the development stage were visited by Dr. Barlow.

SAINT EUGENE MINES.

This group of mining claims embracing an area of 520 acres is situated at Moyie on the Crows Nest branch of the Canadian Pacific Railway. It is now, and has been for some time, the largest lead producing property in British Columbia. It was formerly operated as three separate mines, but they are now owned by the Saint Eugene Consolidated Mining Company (Limited).

There are two almost parallel veins, the main and the south vein, connected at irregular intervals by a series of interlacing cross fissures, extending from the foot wall of the south vein to the hanging wall of the main vein; these cross fissures form the principal ore deposits of the mine.

The Saint Eugene vein was discovered in 1895 and has been worked ever since for silver-lead ore. The group is developed by fourteen tunnels, practically all on the veins, varying from 250 feet to 2,000 feet in length and aggregating about 14,000 feet of drifts. There is a shaft on the Lake Shore mine 125 feet deep from which the 1,900 foot level has been opened. This is the only shaft on the property.

The main veins would average about six feet in thickness, the pay ore about five feet. The cross fissures, however, and the points of their junction with the main fissures are the depositories of great ore masses, reaching in some cases a thickness of thirty and, I believe, in one instance, sixty-three feet.

A plan of the 1,500 and the 1,800 foot levels is reproduced on plate XXXVII to show this extremely interesting vein formation. The property was practically closed down at the time of my visit, (November 8, 1905), on account of the recent fire destroying the shaft-house, hoisting plant, and mouth of the 1,800 foot level. I was, however, enabled to pay a hasty visit to part of the 1,700 foot and 1,800 foot levels. My observations during this brief trip through the workings is to the effect that the main and the south veins occupy irregularly fissured and slightly crushed zones which have been impregnated with galena, and a little zinc blende occasionally associated with garnets, and in which there is little sign of faulting and not much slickenside, or gouge. The idea conveyed on examining the vein structure is that of an irregularly fissured and broken zone, the fractures in which have been filled principally with galena; towards the ends of the galena shoots, zinc blende predominates, but where the galena is at its best, there is but little zinc blende.

An interesting feature is the interlacing veins the drifts on which are designated as avenues. The first avenue on the 1,800 foot level appears to have been fairly productive, while the second avenue is perhaps best described as a bonanza deposit. In one stope between the 1,700 foot and 1,800 foot level on this cross vein, I noticed four feet of almost solid galena on the foot-wall and a three foot seam on the hanging wall of similar material, with streaks and veins of galena occupying the crushed zone of country rock between these ore seams, the whole of this cross vein or avenue being at this place twenty-five feet in width: a very remarkable showing.

The country rock may be an altered slate, but it exhibits at this time no slaty structure. It is very silicious, and in places almost a quartzite. This rock cannot be said to be softened or altered in the vein. In places quartz occurs in the vein formation, more particularly where there is impoverishment. I could see but little quartz in the second avenue stope previously noted. The northern vein is probably the main one and carries in places a little gouge on the foot-wall and irregular seams of vein quartz, but the rock fractures and vein structure suggested a shattering rather than a shearing or faulting movement indicative of deep fissuring. The longitudinal section on the veins show that the stoping conforms fairly well to the contour of the hill, and that a line drawn parallel with the latter at a depth of 200 feet would possibly include all the rich stopes in the veins. The ore deposit will thus appear to be somewhat superficial. No doubt in places where the fissuring is more profound and the shattered zone large and well mineralized, pay ore will continue below the 200 foot contour and in other places where the fissuring is less intense it may not reach this depth.

The country rock is thinly bedded, and the beds have a southerly dip of about ten degrees. Pyrites is common in the rock joints away from the vein and many of the bedding planes near the surface appear to have been water courses and are now partially filled with iron oxide. The vertical development from the outcrop of the vein on the mountain top to the bottom of the Lake Shore shaft is about 2,000 feet. The management claims to have about 150,000 tons blocked out in this property and the records show that from April 1900 to September 5, 1905, the mill produced one ton of lead ore from 4.5 tons of mill feed. During this period 69,162 tons of lead concentrates were produced from about 320,000 tons of ore. The average output of late has been 2,700 tons a month of concentrates containing 65% lead and 32 ozs. of silver, produced from about 13,000 tons of crude ore, which is said to average about 18% lead.

Three hundred men are employed when the mine is in full operation. The mill is located on the hillside near the shore of Moyie Lake and is connected with the Saint Eugene mines near the summit of the mountain by means of a wire tram, and by a surface tram to the Lake Shore and Moyie mines.

The mill is operated by water power about three months in the year; at other times by steam. The capacity of the plant is twenty tons per hour. It consists of two Blake breakers, 12 inches by 15 inches; three rolls, 10 by 30 inches; one roll, 12 by 36 inches; one 5 ft. Huntington mill;

eighteen jigs of two compartments, five of three compartments, and three of four compartments, jigging 25 mm, 15 mm, 10 mm, 8 mm, 6 mm, and 3 mm, and also three hydraulic sized products. The mill also contains five single and four double-decked Wilfley tables, Frue vanners, etc.

A special re-concentration process is in use on the fine tailings, and with satisfactory results. The method was devised by the mill manager.

The milling cost is in the neighbourhood of 70c. per ton, the repairs average about 10c. per ton for material used, exclusive of labor. Ten men are employed in the mill, each shift of twelve hours. The bosses are paid $4.50, jig men $3.50, table men $3.50, engineers $3.75, roustabouts $3. Coal costs $3.80 per ton; wood $3.50 per cord.

Zinc blende was saved for some time, at the rate of twenty-five tons per day containing 20% zinc and 4% lead, but owing to the zinc being mixed with garnets, and so intimately associated with fine pyrites, no market could be obtained for it, nor yet any process that would raise the tenor of the zinc to the market requirements of 45 to 50% metal. Consequently the zinc is allowed to pass off with the tailings into Moyie Lake. A sample of this zinc concentrate was obtained for experimental treatment at Denver, Colorado.

SULLIVAN GROUP.

This property is situated near Kimberly in the Fort Steele Mining District. It consists of three claims about fifty-seven acres, owned by the Sullivan Group Mining Company.

The Sullivan vein was discovered in 1895 and with certain intermissions has been worked ever since for silver-lead ore. While zinc occurs in considerable quantities it is not of commercial grade and none has been shipped. The Sullivan Group has been developed by five shafts and one tunnel about 400 feet in length. The shafts are all shallow, the deepest being 122 feet. The ore is an irregular occurrence and would average about sixteen feet in thickness in the present workings; the greatest thickness observed is thirty-five feet—this, however, for lead-ore, almost free from gangue. If the zinc ores were included the thickness would be much greater.

The mine produced 7,000 tons when operated by the old company, and 18,000 by the present owners, up to the end of October, 1905. The average grade of the ore mined in 1905 is: silver 11 ozs., lead 27%, zinc 12%. The average production is 2,500 tons per month. About forty-five men are employed at the mine to produce this tonnage and keep up the development work.

A branch of the Canadian Pacific Railway runs from Cranbrook on the Crow's Nest Line to Marysville, where the Sullivan Group smelter is situated, and from there to Kimberly. From this latter place a wire tramway extends to the principal shaft (No. 5) of the group, connecting the mine with the railway, affording cheap and rapid transportation between the mine and the smelter at Marysville.

The Mine.—The development consists of a few shallow shafts and drifts in an extremely large and irregular deposit. None of the workings (November 10, 1905) had been sufficiently extended to show the length of the ore along the strike of the lode or deposit, and only the 100-foot level had with any definiteness shown the thickness of the ore deposit. Preparation had, however, been made to test the property and prove the extent of the ore occurrence by diamond drilling.

Referring to Plate XXXVII, it will be seen that near the outcrop the deposit is very flat, while below the 60-foot level it assumes a vein-like form with a regular dip. In the flatter portion, the full width between the assumed walls is perhaps nowhere continuous ore; in fact one shaft was sunk clear through the zone, without meeting a trace of ore. However, on the line of the section it will be seen that the workings show up a very considerable thickness of solid zinc-lead ore.

This main shaft (No. 5) is 100 feet deep. From the bottom a drift has been extended south 38° west to open up the vein, which is here very well defined, showing an oxidized hanging wall of softish slate. The main wall is, however, apparently a thinly bedded quartzite. Next the hanging wall is, as previously noted, an oxidized slaty seam two feet thick, which can be seen in three places on this bottom level. In the upper workings I saw it in three other places, the cleavage paralleling the vein wall, which structure, however, may have been developed by pressure. The hanging wall of the deposit is thus fairly well defined, at least on the 100 and 60 foot levels, between which the dip is about 60°. Above the 60-foot level, however, the deposit appears to flatten, with possibly a very irregular and difficultly determined hanging-wall, for the reason that it has not been exposed with any regularity. The foot wall is not by any means so clearly defined either. Irregular impregnation and lenses of sulphide occur in the supposed foot wall even on the 100-foot level, while in the upper workings, certainly in those above the 60-foot level, no one can say that the true foot wall has been anywhere found. I suspect the deposit will on development prove to be a large fissure vein, that from its size and persistency will doubtless continue to considerable depth. On the 60-foot level two well marked vertical fissures occur in the ore; containing in places considerable oxidized matter; but they do not seem to extend to any considerable length in the deposit. It will thus appear that this fissure vein has on the 100-foot level a quartzite hanging wall and in all probability a quartzite foot wall, though on the upper levels, for the reasons given, the deposit appears more like a bed and will probably average double the thickness of the ore proved on the 100-foot level; in fact good lead ore in one place comes up to the wash and is now being stoped immediately below it in an open trench near the compressor building. In places in the upper workings the ore deposit looks very much like a replacement, though it is difficult to conceive a replacement of quartzite by lead and zinc ores; yet, a full examination will, I have no doubt, prove up a considerable portion of this flattish deposit above the 50-foot level as a replacement deposit of some sort.

The ore is an exact counterpart of that known in Wales as "blue-stone" occurring in some quantity in the Mona mines in Anglesea, and also in the Connorree mines in Ireland. It is an intimate mixture of galena, zinc blende and iron pyrites, presenting a fine grained structure with fracture resembling that of the cast steel. In places the lead predominates and zinc does not exceed 10%, while in other places zinc predominates. The present mining is to a large extent the selection of the leady portion of the vein. Future development, however, will undoubtedly prove up an extremely large tonnage of zinc ore, which on account of its intimate mixture with galena and pyrites will prove a very difficult problem in separation. I am of the opinion that the Sullivan mine has blocked out more zinc ore, such as it is, than any other mine examined in British Columbia, yet sufficient work has not been done to prove the nature of the deposit or to give any line on its ultimate production. The diamond drilling recently started will, no doubt throw a flood of light on this extremely interesting and very valuable zinc-lead ore deposit.

If the Sullivan deposit should on development prove up to be a lens-like mass, then the central portion of the lens, contains a very large tonnage running, say about 11 ozs. silver, 20% to 25% lead and 8% to 16% zinc, while the edges of the lens will probably consist of 20% to 25% leady zinc ore, and similar zinc ore will doubtless be found in places associated with the lead ore near the central (thicker) portion of the lens.

ROCKS AND ORES OF THE SLOCAN.

DYKE ROCKS.

The slates and limestones have been described elsewhere. The dykes have been classified as basic dykes, and porphyry dykes in the field, the usual miner's classification; samples of some of the principal dykes in the Slocan and Ainsworth Districts were, however, collected and submitted to Dr. A. E. Barlow, of the Canadian Geological Survey for microscopic examination. Dr. Barlow reported on these dyke rocks as follows:—

"The following are the numbers and localities as well as the rough field determinations by Mr. Argall:

No. 1. Basic dyke, Goodenough mine, Sandon.
No. 2. Basic dyke, Highlander mine, Ainsworth.
No. 3. Basic dyke, Blue Bell mine, Ainsworth.
No. 4. Porphyry dyke, Slocan Star mine, 5th level east, Sandon.
No. 5. Basic dyke, Whitewater Deep mine, Whitewater.
No. 6. Dyke in tunnel, Whitewater Deep mine, Whitewater.
No. 7. Porphyry dyke, Jackson mine, (Sept. 8th, 1905).
No. 8. Porphyry dyke, (cut by Slocan Star vein), Sandon.

"Nos. 1, 2, 5 and 6 are all related types of rocks. They belong to the general group of dioritic lamprophyres, and are closely allied, if not identical with the kersantites.

No. 1.—Goodenough mine, Sandon, B. C. The hand specimen shows a dark, greenish grey, rather coarse-grained, porphyritic rock. The porphyritic constituents are chiefly biotite and olivine, the comparatively large and rounded individuals of the latter being now almost wholly replaced by an aggregate of carbonates and talc. Augite is also abundant in stout columnar, almost colourless, crystals. The pale brownish, feldspathic, ground-mass, with abundant inclusions, shows the rudely radiating arrangement so frequently noticed in these rocks. Apatite is the most abundant of the accessory constituents. Magnetite and pyrite are both disseminated through the rock.

No. 2.—Basic dyke, Highlander mine, Ainsworth, B. C. The hand specimen shows a dark green, almost black, brightly glistening, porphyritic rock. Under the microscope the rock has been identified as a kersantite, closely related to No. 1. Augite is the most abundant of the porphyritic constituents, although biotite approaches it very closely in point of abundance. It occurs in the usual short, stout, almost colourless crystals. The olivine is not so abundant and some of the individuals are comparatively fresh, while others are decomposed to talc and serpentine. Irregular areas of chlorite are of frequent occurrence. A thin vein of calcite traverses the section. Much of the original plagioclase ground-mass is decomposed to calcite.

No. 3.—Basic dyke, Blue Bell mine, Ainsworth, B. C. The hand specimen shows a dark grey porphyritic rock with amygdaloidal cavities occupied chiefly by calcite. The thin section shows porphyrite. The groundmass is made up of small interlacing laths of plagioclase and abundant deep brown biotite, in which are developed phenocrysts of plagioclase and occasionally quartz. The amygdaloidal cavities in the thin sections are filled up with calcite together with some zeolite.

No. 4.—Porphyry dyke, Slocan Star, 5th level east. The hand specimen shows a pale, greyish, fine-grained, massive rock. The thin section is probably that of a porphyrite, but only suggestions remain of the original structure. It is now made up of a confused aggregate of felspar, bleached and often chloritized biotite, sericite and quartz.

No. 5.—Basic dyke, Whitewater Deep mine. Only a small piece of rock was available for the thin section and no portion of it was returned.

The examination of the thin section under the microscope shows the rock to be a kersantite, although the true nature of the felspathic groundmass cannot be determined satisfactorily. The porphyritic constituents are biotite, olivine (decomposed to talc) and augite. The scales and plates of biotite show the characteristic deep brown borders, tinged with green, with comparatively pale coloured bleached interiors. The olivine originally present is now replaced by rounded areas made up of brilliantly polarizing scales of talc. The augite is in the usual short, stout and almost colourless tabular crystals. The groundmass is pale brownish in colour, showing in places comparatively fresh, well developed, blade-like crystals of plagioclase, usually with well-defined radial arrangement. A very little quartz is present, occupying the irregular interspaces left by the other constituents. A small amount of magnetite and pyrite were also noticed.

No. 6.—Dyke in tunnel, Whitewater. The hand specimen shows a comparatively pale greyish porphyritic rock, with very numerous small, but rather faintly glistening, phenocrystals of biotite. Abundant, more or less rounded, yellowish green patches are conspicuous, which on investigation with the microscope represent olivine originally present but now wholly replaced by a brilliantly polarizing aggregate of small, pale yellow scales of talc. The rock is a dioritic lamprophyre of the kersantite type. Biotite is the most abundant of the coloured constituents, the various scales and plates showing the usual deep brown, strongly pleochroic borders with bleached interiors. The plagioclase is probably labradorite and much of it is decomposed to calcite. The blade-like crystals of plagioclase appear to have a sheaf-like grouping, which gives to the groundmass characterized by their presence a rather indefinite or wave-like extinction. The olivine originally present is now wholly replaced by talc. Magnetite is disseminated through the rock, sometimes as sharp octahedra. Pyrite is also present in occasional grains. Apatite is very abundant. Quartz fills up a few of the irregular interspaces.

No. 7.—Porphyry dyke rock, Jackson mine (September 8, 1905). The hand specimen shows a pale greyish massive rock with abundant small disseminated grains of pyrrhotite and pyrite. The thin section shows a a very much decomposed porphyrite. It is composed of a confused aggregate of calcite, sericite, felspar, quartz, pyrrhotite and pyrite. Some larger individuals made up chiefly of calcite suggest original phenocrysts of plagioclase, while in places considerable areas of the comparatively unaltered felspathic groundmass still remain.

No. 8.—Pophyry dyke, cut by Slocan Star vein, Sandon. The hand specimen is of a pale greyish rock. Under the microscope it is seen to be a porphyrite which has undergone very pronounced alteration. The outlines of the original phenocrysts are still rather sharply defined from the surrounding groundmass, although both are largely replaced by carbonates."

A few notes on the occurrence of these dykes may be added.

No. 1.—Basic dyke, Goodenough mine. Sample collected by Mr. Gardé, and said to be cut by the vein.

No. 2.—Basic dyke, Highlander mine, Ainsworth. This dyke, determined by Dr. Barlow as kersantite, cuts through the Highlander vein and the crystalline schists, and appears on surface, or at least a dyke of similar composition can be seen on the summit of the cliff above the line of the Highlander tunnel; other dykes of this rock occur in the neighbourhood. (See Plate No. I).

No. 3.—Basic dyke, Blue Bell mine. The field examination showed a close resemblance to No. 2, excepting, however, the development of calcite in the cracks and cavities of the rock. This dyke is intersected in the main tunnel of the Blue Bell mine, and is shown in section in Plate V. A band of limestone cut by this dyke is changed to calcite with the development of very large crystals, while in the basic rock of the dyke some of the calcite amygdaloids reach $\frac{3}{4}$ inch diameter.

No. 4.—This dyke cuts into the Slocan Star vein on the 5th level, east of the main cross-cut, and shows in places fine pyrite. It is typical of the porphyry dykes (porphyrite), associated with the silver-lead veins of the Slocan.

N. 5.—This is from a small basic dyke in the main tunnel of the Whitewater Deep mine.

No. 6.—This sample is from a large dyke that crosses the main tunnel of the Whitewater Deep mine, and along the course of which considerable drifting was done. In the field the rock appeared about identical with the Goodenough dyke, No. 1, but more decomposed.

No. 7.—This is a typical specimen from the porphyry dykes (porphyrite) of the Jackson mine, Jackson Basin.

No. 8.—Is from a somewhat similar dyke, cut by the Slocan Star vein in the 5th level west, which carries pay ore through the dyke.

Summing up, the typical porphyry dykes of the Slocan are represented by Nos. 4, 7, and 8, and are correctly designated as porphyrites.

The basic dykes are represented by Nos. 1, 2, 5 and 6 and may be designated as kersantites.

The granites vary from the extremely coarse crystalline Nelson type, in which the Molly Gibson vein occurs, to the fine grained hornblendic variety, occurring as intrusions in the slates, and even down to the porphyrites, which occur in small dykes, though no doubt derived from the intrusive granite magma.

THE ORES.

Blende.—Representative specimens collected in the field were forwarded to Mr. W. George Waring, of Webb City, Missouri, for special analysis of, so far as possible, the pure minerals. Mr. Waring enclosed the following note with his analysis:—

"Note to Analyses No. 28437-40.

The nearly complete analysis of the siderite and blende of No. 28439-40, sufficiently explains the composition of the others, therefore the analysis of the others was not carried further than appeared absolutely necessary.

Upon the blende of 28440 the following special tests were made:—

(A.) Three grams treated with dilute (1%) H_2SO_4 at about 90°C. showed no marked flotation. The solution was found to contain all the manganese and 0.42% Fe, and these are therefore assumed to exist as siderite.

(B.) The washed residue from (A.) was shaken for several minutes with a 1% solution of KCy and allowed to stand over night in contact with the solution, which latter, after filtration, was found to contain 4.28 mg.Ag, equal to 0.143% Ag. dissolved, which must have existed as Ag_2S.

(C.) The washed residue from (B.) was shaken for a few minutes with a globule of pure Hg without addition of any chemicals. The mercury, after separation was found to contain 1.62 mg Ag, equal to 0.054%, thus showing that a part of silver, at least, exists as native silver. The residue upon scorification yielded the exact remainder of the silver, namely 19.60 mg, which is equivalent to 0.653%.

(D.) Three grams of the blende boiled with strong HCl gave a solution containing all of the zinc, nearly all of the lead, together with a little copper and 8.92 mg Ag. The residue insoluble in HCl contained 1.14% Fe, which is assumed to be the amount existing as pyrite in combination with the copper sulphide."

A. Blende, associated with siderite, from 4th level east, Slocan Star mine. Analysis No. 28,439.

B. Blende, associated with siderite, from the 8th level, Payne mine, analysis No. 28,438.

ANALYSES OF ZINC BLENDE ORES.

	A.	B.	C.	D.
Zinc	58.10%	59.70%	54.25%	53.47%
Iron	4.39	3.12	5.68	4.12
Manganese	0.00	0.00	0.00	0.22
Lead	0.20	0.00	1.00	5.72
Copper	0.66	1.00	1.06	1.70
Silver	0.082	0.048	0.355	0.8505
Sulphur				30.37
Silica				1.00
CO_2, As, CaO MgO by difference				2.55
				100.00%

C. Blende, associated with siderite, Molly Gibson mine, analysis No. 28,439.

D. Blende, a special sample of high grade silver blende, showing native silver in the cleavage planes of the crystals. Analysis No. 28,440.

These analysis show a fairly high zinc, varying from 53.6% to 59.7%. The silver values in the blende appear to follow the lead rather than the copper mineral.

As for example:—

	A.	B.	C.	D.
Copper	0.66%	1.00%	1.06%	1.70%
Silver	0.082	0.048%	0.035%	0.8505%
Lead	0.20	0.00%	1.000%	5.75

The highest ratio between copper and silver is 30.3 to 1 in (C); the lowest, 2 to 1 in (D). The highest lead ratio is 28.6 to 1 in (C); the lowest, 2.44 to 1 in (A). The average is:—

Copper to Silver. . . . . .13.4 to 1.
Lead to Silver. . . . . .12.6 to 1.

The samples vary so much in silver values that one cannot safely theorize on them. In testing the ores and concentrates, however, it is very clearly seen that the high silver values in the zinc is invariably associated with galenite.

Mr. Waring figures the mineral composition of the almost complete analysis on the high grade blende (D), as follows:—

Zinc blende	79.78%	Sulphur equiv.	26.31
Chalcopyrite	4.57%	"	1.73
Galenite	6.61%	"	0.89
Argentite	0.88	"	0.13
Fe S	3.20%	"	1.16
Siderite	1.67		
Quartz	1.00		
	97.71		30.22%

"I found no appreciable amount of cadmium in the blende."

The almost entire absence of manganese in the blende is curious, more especially as the associated mineral siderite carries a high percentage of it.

Siderite.—This mineral is abundant in all the veins in the Slocan rock series, is invariably associated with the zinc blende and very often replaces that mineral in depth, and at still greater depth is itself replaced by quartz. The concentrate from the mills is almost invariably a mixture of zinc blende and siderite, the separation of which will in most cases give a 48 to 50% zinc ore, suitable for the manufacture of spelter. Therefore, a close examination of the siderites was taken in hand, to throw light on the distribution of the zinc and the silver in siderite ores.

The blende and siderite ores were in each case taken from the same hand specimen, and subjected to partial analysis; those of the blende have already been discussed, while the following analysis are, it must be remembered, made from adjoining seams of siderite.

	A.	B.	C.
Iron	28.40%	30.15%	17.62%
Manganese	13.14	12.97	26.55%
Magnesia	5.52	....	....
Lime	1.26	....	....
Zinc	0.00	00.00	00.00%

Corresponding to the following proximate mineral composition:—

	A.	B.	C.
$FeCO_3$	58.88	62.31%	36.53%
$MnCO_3$	27.48	26.12%	55.52%
$MgCO_3$	11.59	11.57%	7.95% by difference.
$CaCO_3$	2.05		
	100.00	100.00	100.00%

These siderites, it will be observed, are quite free from zinc. The pure specimens are also free from silver, and an examination of the silver-bearing siderite separated from the blende in the magnetic separating machines, invariably shows included zinc or lead minerals; hence the presumption is that if this magnetic siderite were ground sufficiently fine it could be entirely freed from silver, or at any rate practically separated from its associated silver-bearing minerals.

The siderite specimen from Molly Gibson mine carried quite a high percentage of manganese, and had a distinctly pinkish hue. The entire absence of zinc in the siderite analyzed is remarkable, as well as the absence of silver in this mineral.

A sack of the Slocan Star zinc concentrates, high in silver, was shipped to Mr. W. Geo. Waring to test by a wet process of zinc extraction. The concentrate as received by Mr. Waring assayed: zinc 34.95%, iron 13%, lead 2.32%, copper 0.50%, manganese 2.10%, silver 0.1924%, gold trace, insoluble 6.88%. To further test the siderite I had Mr. Waring pick out several apparently pure pieces from the general mass of the concentrate, and subject them to analysis. He reports as follows:—" Ten grams of carefully selected particles of the siderite, from 2 to 3 mm. size, as free as possible from blende and other minerals, assayed for silver gave 2.28 oz. per ton.

Six grams of the same material gave 0.12% zinc, and 2.34% silicious residue. Although the siderite grains were carefully selected, most of them showed under the microscope dark stains and often minute (microscopic) crystals of blende or tetrahedrite, probably both, imbedded in the siderite matrix. I infer, that the zinc and silver do not exist as molecular constituents of the siderite, but as constituents of other minerals associated with the carbonate."

This confirms the conclusion previously arrived at; and while there are no doubt some exceptions, yet we are safe in assuming that the Slocan siderites are practically free from combined zinc or silver.

Iron pyrites is of course a very common mineral in the Slocan veins, but it seldom carries precious metal values.

Pyrrhotite is less common than pyrites, but is invariably found in the zinc concentrates. It is usually more abundant in the neighbourhood of the eruptives.

The zinc concentrates received from the Slocan Star mine contain a large amount of pyrrhotite.

Calcite occurs in all the veins, very often associated with galena, and in some cases forming the exclusive gangue of the pay ores.

Galena and grey copper (*Freibergite*).—The galena ore in the Slocan veins carries a slight sprinkling of grey copper, particularly when the silver values are high; yellow copper is also present to a less extent, and while often found in the shallower portion of the veins, seems to be more plentiful in depth. Both of these copper minerals appear to be primary minerals deposited in an irregular way in the galena. I procured some galena specimens showing grey copper from Mr. O. V. White of the Slocan Star mine, and submitted them to Mr. Waring, witth he request that he separate the minerals and subject each to analysis. Mr. Waring states that "the galena was not quite free of grey copper nor was it possible to pick out grains of the grey copper that did not show minute (microscopic) crystals of galena imbedded in the tetrahedrite mixture. The galena was, however, quite free from adherent quartz gangue, although there was perhaps 1 or 2% of quartz, accompanied by spicules of grey copper, in the galena. The grey copper itself is impregnated with quartz. The sample obtained contained 10.66% insoluble quartz gangue."

Galena and grey copper from the Slocan Star mine, Sandon, British Columbia. Analyses Nos. 28,303 and 28,304.

	Galena.	Grey Copper.
Silver	2.472%	15.142
Copper	1.69%	11.27
Lead	74.32%	5.40
Zinc	1.63%	8.15
Iron	0.88%	5.81
Sulphur	....	20.13
Antimony	....	17.72
Arsenic	....	None
Insoluble	....	10.66

2.472% Silver = 720.98 oz. per ton.
15.142% Silver = 4416.32 oz. per ton.

Mr. Waring computes the proportions of silver carried by the copper and the lead mineral, and arrives at the conclusion that the galena carries only about 60 oz. of silver per ton, the remaining 660 oz. of silver in the galena being due to intermixed grey copper.

The grey copper appears to be of the variety freibergite; rejecting the galena in the above analysis, which is evidently an admixture with the freibergite, Mr. Waring computes the pure mineral to have the following composition.

Freibergite from Slocan Star mine:

Silver	19.57%	Computed composition.
Copper	14.57%	
Zinc	10.53%	
Iron	7.51%	
Antimony	22.91%	
Sulphur	24.91%	
	100.00%	

It is abundantly evident that the high silver value in the galena is due to the associated freibergite, while the high silver values in the zinc blende is in part due to freibergite, but more often to galena, or perhaps galena associated with freibergite, in which the latter is finely disseminated through the galena.

Ore Lot No. 7 is a case in point. This is zinc concentrate from the Enterprise mine, assaying 115 ozs. of silver per ton and 4.8% lead. When crushed to 20-mesh and concentrated a product was obtained assaying 45% lead and 298 ozs. silver per ton. Also with ore Lot No. 13, zinc concentrate assaying 80 ozs. silver and 11.5% lead, upon passing it over the Blake separator a product was thrown out assaying 198 ozs. silver and 35% lead. Other examples could be given.

Native silver occurs in both the galena and the zinc blende, but more often in the former; it is of course secondary and does not occur in quantity sufficient to cut any figure in the general tenor of the ores.

ECONOMIC CONDITIONS.

The Slocan region is well provided with railways. Sandon, the principal mining center, has two lines of railway, (1) the Canadian Pacific, running down to Lake Slocan, and connecting there by steamer and rail with Nelson, Trail and the Crow's Nest line, or by rail direct to Nakusp, thence by steamer and rail to the main line of the C.P.R. at Revelstoke; and (2) the Kaslo & Slocan, which runs from Sandon to Kaslo, where connection is made with the Kootenay Lake steamers.

The wagon roads to the principal mines are mostly well built, while excellent "raw-hiding" trails and good pack trails are numerous in the mining districts. Most of the principal mines have aerial or gravity trams connecting the main workings with mill or railway, as described elsewhere in this report. Considering the small output from Sandon and vicinity the means of transportation are superlatively abundant.

LABOR.

I was assured that the labor supply was ample, though I must add, that most of the mines visited by the Commission were either idle or working on a very small scale. The average wages in the Slocan are as follows:—

General foreman, per day of 8 hours	....	$5.00	to	$6.00
Shift bosses "	....	4.00	to	5.00
Miners, "	....	3.25	to	3.50
Trammers "	....	3.00	to	3.50
Blacksmiths "	....			4.00
Carpenters "	....	3.50	to	4.00
Machinists.. "	....	3.50	to	5.00
Stationary engineers "	....	3.50	to	4.00

General mining supplies compare favorably in price with the average Rocky Mountain mining camps; hardware, however, is from 15% to 20% dearer than in the average American mining camps. Fuel is on the whole cheap, due no doubt to the densely wooded country around the mines. Cordwood varies from $3 to $4. Coal at Sandon costs about $7 per ton. Water power is almost everywhere available and abundant for about six months in the year. Several of the mills have auxiliary steam plants, but so far as I could determine these were but seldom used; the general practice being to close down the mills when water power is not available. None of the steam plants was in operation during our visit to the various mining districts. While in the field the members of the Commission visited 12 concentrating mills, nine of which were idle. Various causes were given for this state of affairs, the chief one being "the plants were closed down for the winter; and would start up again in the spring," indicating that only a semi-annual campaign was contemplated. Other mills were awaiting the completion of magnetic separation plants to render the zinc concentrates marketable.

ORE RESERVES AND OUTPUT.

Generally speaking the mines of the Slocan have been hard pushed. Development in most cases has been neglected, the ore reserves allowed to dwindle (in some cases almost to the vanishing point), and mining generally is in an extremely precarious condition; best illustrated by the fact that notwithstanding the high prices of silver, lead and zinc, the output has not been stimulated, nor the mining industry roused from the lethargy into which it had fallen.

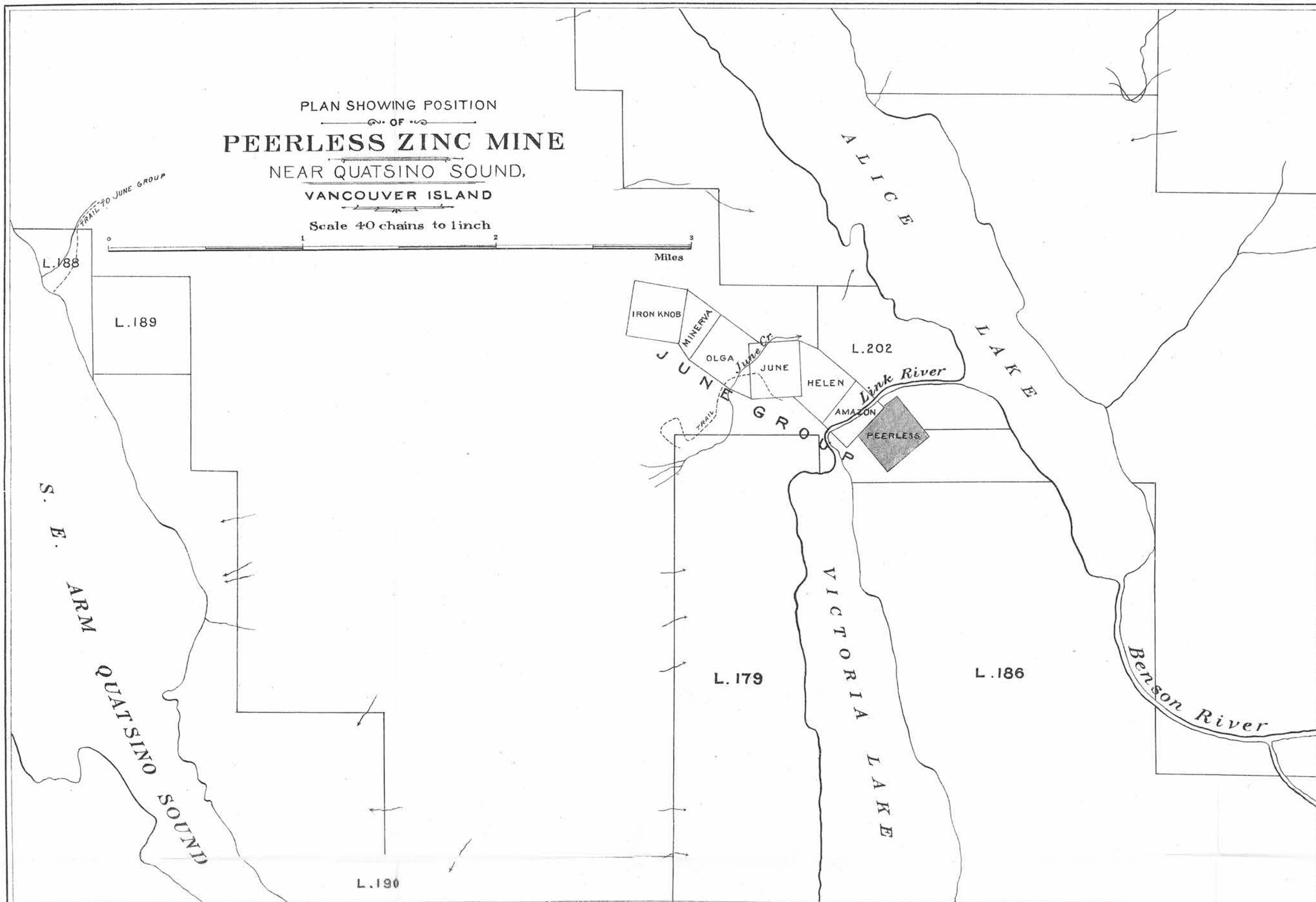
There are various reasons given for this mining torpidity in a district crowded with mineral veins, most of which carry very high grade argentiferous galena and zinc blende in paying quantities when worked in connection with the silver-lead ores. The principal reasons, however, are in my opinion: *First*, the operative system is in many cases too expensive; and *second*, proper facilities do not exist for making a close saving of values in the concentration mills, or for preparing the zinc ores for the smelters.

I shall pass over the boom period, with its lavish and irreparable waste of capital, when comparatively insignificant prospects were in some cases equipped with expensive mills and tramways, to offset, by surface display, as it were, the lack of ore in the mines. Or when small though profitable veins—if worked economically—were swamped with expensive and cumbersome mills and tramways, far and away beyond the capacity of the mines to supply them with ore. These things occur in the heyday of most mining districts, and the Slocan is no exception. While it is best to dismiss them from consideration, they will not be forgotten so long as these monuments of folly or extravagance occupy conspicuous positions in the various mining districts.

OPERATING EXPENSE.

An examination of the details of the mine reports will, I think, clearly show that the Slocan veins, though rich, are also small, the ore lenses short and uncertain, very often with long stretches of barren vein between them. Such conditions are not usually conducive to the success of an expensive management, a paid directorate, or a large investment of capital.

Leasing is the proper system to apply to the economic working of the argentiferous galena veins of the Slocan; leasing with a fair royalty and leases running for not less than one year. "The labourer is worthy of his hire," and the lessee who by hard work, sound reasoning and expenditure of money, is fortunate enough to discover a good lense of ore, is rightly entitled to enjoy for a season the fruits of his labour. Many mines, and even mining districts, in Colorado have again and again been brought from a state of torpor and decay up to a high tide of prosperity simply by the leasing system. Some rich mines at Cripple Creek are now, and have been for years, worked almost exclusively on the leasing system, for the reason that it has been proved, the cheapest and the best method of operation.


PLAN SHOWING POSITION
OF
PEERLESS ZINC MINE
NEAR QUATSINO SOUND,
VANCOUVER ISLAND
Scale 40 chains to 1 inch
0
1
2
3
Miles
TRAIL TO JUNE GROUP
L.188
L.189
L.190
L.179
L.186
L.202
IRON KNOB
MINERVA
OLGA
JUNE
HELEN
AMAZON
PEERLESS
June Cr.
TRAIL
JUNE GROUP
Link River
ALICE LAKE
VICTORIA LAKE
Benson River
S. E. ARM QUATSINO SOUND

REPORT

ON THE

OTHER OCCURRENCE OF ZINC ORE

IN

BRITISH COLUMBIA.

BY

J. E. GRIFFITH, G. C. TUNSTALL,
FRED. FRASER, J. F. ARMSTRONG,

AND

A. L. SMITH.

Gold Commissioners of British Columbia.

1906.

OTHER OCCURRENCES OF ZINC ORE IN BRITISH COLUMBIA.

In order to obtain additional information as to the occurrences of zinc ore in British Columbia, which were not examined by the Commission, a series of inquiries prepared by Mr. Ingalls was addressed to the Gold Commissioners of the Province, through the Hon. Richard McBride, Provincial Minister of Mines. The replies of the Gold Commissioners, who reported the occurrence of zinc ore, are here given. They include notes on several mines which have undergone more or less development in search of lead ore, and thereby have disclosed the existence of zinc ore, but have not yet been productive of that ore, the term "productive" being here used in the sense of furnishing ore for market. While there is reason to believe that the reports of the Gold Commissioners, together with that of Dr. Barlow, cover the principal showings of zinc in the Province upon which some work has been done, there is ground to suppose that the literally undeveloped occurrences are far more numerous.

The Gold Commissioners at Kaslo, Greenwood, and Grand Forks reported having no information on the subject.

GIANT MINE.

J. E. Griffith, Gold Commissioner at Golden, under date of January 25, 1906, transmitted the following letter from F. P. Armstrong, dated Jan. 6, 1906:—

As requested by you I beg to furnish the following information regarding the Giant mine.

1. Owner, F. P. Armstrong, Golden, B.C.

2. Locality, Spillamachene Mountain, south of Golden.

3. Forty-eight miles from main line of C. P. R., six and a half miles from steamboat navigation on Columbia River and also from located line of Kootenay Central Ry. Elevation above Columbia River 600 feet.

4. Present communication from river is by sleigh road. Good wagon road with light adverse grades can be cheaply built. Country is of park character, with large fir timber. Spillamachene River, one mile from claim, would furnish large water power.

5. The zinc ore occurs in soft black slate, which forms the hanging wall of a large galena-bearing ledge. This ledge has been considerably developed, and would appear to have an average width of 25 feet, a depth of 400 feet (to lower tunnel) and is mineralized for a length of at least 600 feet, the galena ore assaying 10 to 15 oz. silver and 60% lead.

The zinc ore does not crop out anywhere, and was found accidentally. It has been run on for 50 feet in an adit level, which apparently cut the top of a lense of ore, varying in width from 1 to 4 feet. A winze was sunk on this ore, showing a streak averaging 30 inches in width.

Another adit is being run 70 feet below that referred to above. At 50 feet from the winze, the face of this lower tunnel was in the same soft black slate; kidneys of zinc ore occurred through it. The hanging wall instead of containing lead only was full of zinc. No assays have yet been made, but it obviously carries too much rock to allow of shipping without concentration. A few feet further on the streak of zinc ore was struck, 2 feet in width, and presumably the same as that which is exposed in the winze.

The ore in the winze is perfectly clean and requires no dressing beyond knocking off pieces of slate that may stick to the side. In running the 50 feet of tunnel about 60 tons of clean ore were taken out. The crest of the ore contained about 10% lead and 45% zinc. In the winze the proportions are 2% lead and 55% zinc.

It is the intention of the owner to ship 500 tons this winter if sleighing lasts until say March 15. However, as the snow fall to date has been very light, the season is likely to be a short one.

MONARCH MINE.

H. T. Coperley, Secretary of the Vancouver Smelting and Mining Company, communicated the following letter, addressed to R. F. Tolmie, Esq., Deputy Minister of Mines, under date of Jan. 27, 1906:—

1. Owner, Vancouver Smelting and Mining Company, Limited, Vancouver.

2. Locality, about two and three-quarter miles east of Field, B.C., and from 700 to 800 ft. above main line of C. P. Ry.

3. The nearest railway station is Field, B.C., which is accessible by trail.

4. Means of communication are a gravity tram, which was formerly in operation, but now requires considerable repair, as does also the trail to the mine.

5. The development consists of a tunnel 250 ft. in length, with crosscuts and a winze 70 ft. in depth.

6. The ore is a mixture of lead and zinc sulphides, together with some lead and zinc carbonates. The lead sulphide carries a small percentage of silver. The formation is limestone. The carbonate of zinc occurs in spots, and a considerable quantity could be hand-sorted; the remainder of the ore requires concentration.

7. An excellent water power could be obtained from the Kicking-Horse river, and an abundant supply of timber for mining purposes.

LONE PROSPECTOR MINE.

G. C. Tunstall, Gold Commissioner at Kamloops, reported under date of Feb. 2, 1906, as follows:—

I have not been able to obtain information in regard to the occurrence of zinc in the Lone Prospector mineral claim, which is situated on the west bank of the North Thompson River, over seventy miles north of Kamloops. The claim was abandoned many years ago. It is reported that the zinc averaged from 20 to 40 per cent. I believe it was associated with galena.

DONALD MINE.

Fred. Fraser, Gold Commissioner at Revelstoke, reported under date of Feb. 26, 1906, as follows:—

1. Owners, David Woolsey and John Caldwell.

2. Locality, Revelstoke Mining Division, about 2½ miles from Flat Creek Siding, on the Canadian Pacific Railway, on north side of said railway, about 32 miles east of Revelstoke.

3. Communication by trail, not in good condition at present, but could be made passable for horses at very little expense. This will be done in the early part of next summer, as the property is likely to be operated in the near future. Free from all snow slides.

4. Development consists of shaft and tunnel. Shaft on vein about 80 feet. Tunnel, 800 feet. Tunnel is being run to tap vein at a depth of 325 feet. Breast of tunnel estimated to be within 30 feet of vein. Width of vein at shaft 9 to 10 feet, showing solid ore full width. Character of ore pyrrhotite, blende and galena. Sample sent to Pellew, Harvey, Bryant & Gilman, assayers of Vancouver, showed 30% zinc. The gangue is quartzose. The association of minerals is coarse. A great deal of hand picking could be done.

5. Plenty of good water power in close proximity to the property; also plenty of timber (spruce and balsam).

Mr. Fraser also reported that zinc ore is found at Keystone Mountain, Big Bend, Jordan Pass, at Revelstoke; and Beatrice and Mammoth mines at Camborne.

FORT STEELE MINING DIVISION.

J. F. Armstrong, Gold Commissioner at Cranbrook, reported under date of Feb. 16, 1906, the following occurrences of zinc ore in the Fort Steele Mining Division:—

VICTOR.—This group of claims is owned by C. W. Freeman and W. Van Arsdalen of Fort Steele and is situated near the head waters of Mans

Creek, a tributary of the Kootenay River, about eight miles in an easterly direction from the town of Fort Steele. Fort Steele Junction, fifteen miles distant, is the nearest railway station. There is a wagon road from the town of Fort Steele to the foot of the mountain, and from that point there is a trail, by the way of Horse Shoe Gulch, to the property at an altitude of 7,000 feet.

On the Victor there is a tunnel giving a depth of 25 feet. The vein crosses a mountain ridge at right angles. Near the base of the ridge there is a vein, two to eight feet wide, stripped for a distance of three hundred feet, being well defined and highly mineralized, containing clean shipping ore.

The ore is argentiferous galena and zinc blende, carrying some gold. It is claimed that the zinc ore occurs in places as blende of a light color. assaying from 55% to 63% in zinc and carrying silver. In other places it is associated with galena. The gangue of the vein filling is white massive quartz. On the edges of the vein the quartz is mixed with country rock, sprinkled with bunches of ore. A considerable quantity of the ore as mined can be sorted ready for shipping.

On the Silver Leaf claim, in the same group, there is an ample supply of timber and water for mining and camp purposes.

KING OF KOOTENAY.—This claim is situated at the head of Victoria Gulch, a tributary of Wild Horse Creek, and is owned by William Voss of Fort Steele. There is a fairly good trail leading to this claim, which is about eight miles from Fort Steele in a north-easterly direction.

BILL NYE.—This claim is situated on Trail Creek, a tributary of Wild Horse Creek, and is distant about 12 miles from Fort Steele, in a north-easterly direction, and is owned by C. Reynolds of Fort Steele. There is a fairly good trail leading to this claim.

ESTELLA.—I am informed the Estella group on Tracey Creek, owned by Alexander Polson, et al, carries a large percentage of zinc ore, but I have been unable to get into direct communication with the owners.

There have been many other locations made in this vicinity between Wild Horse and Tracey Creeks, but allowed to lapse, owing it is stated to the large percentage of zinc they contain.

AURORA.—This group is situated on the west side of Lower Moyie Lake, and is owned by Thomas Rader, Morrissey Mines, Ole J. Johnson, Moyie, and Irwin B. Sanburn, Portland, Oregon. These claims are ½ mile distant from the station across the lake, and 4 miles distant by trail to Aldrich Siding on the C. P. Ry. On the Aurora there is a tunnel 450 feet long connecting with the surface and several other smaller openings. The ore is blende associated with galena. There is an abundance of wood and water in the immediate vicinity of these claims.

HESQUOIT LAKE, VANCOUVER ISLAND.

A. L. Smith, Gold Commissioner at Alberni, reported under date of February 3, 1906, with respect to a group of zinc-bearing claims near Hesquoit Lake, as follows:—

1. Owner, Arthur Norris, Clayoquot, B.C.

2. Locality, south side of Hesquoit Lake, West Coast, V.I. The claims are on the shore of the lake.

3. Distance from steamboat landing at Hesquoit, 6 miles.

4. Means of transportation are by small boat. The country is rough and steep.

5. The property has been developed by open cuts and a shaft.

6. The ore is blende, associated with galena, pyrites, pyrrhotite, chalcopyrite, and barytes. The association of minerals is mostly coarse. The blende occurs in fairly clean streaks, and considerable could be picked by hand.

REPORT

ON

METHODS FOR THE CONCENTRATION

OF

ZINC ORES OF BRITISH COLUMBIA

BY

PHILIP ARGALL.

Member of the Royal Irish Academy. Member of the American Institute of Mining Engineers. Member of the Institution of Mining and Metallurgy. Fellow of the Geological Society of America. President of the Colorado Scientific Society, Etc.

1906.

sulphides are covered by three or four feet of gossan. The rocks near by are also decomposed to a rusty product, in cavities of which iron stalactites of fantastic form are present.

A tunnel had been started and for its entire length of 20 feet was in solid sulphides. The width of the ledge was, at the time of my visit, impossible to determine. It was said to be traceable throughout the length of 19 claims.

The iron sulphides formed the bulk of the "ore" at the points where the ledge was seen, but zinc blende ranked next in abundance, and it is not impossible, if the ledge has the dimensions popularly attributed to it, that at some points the zinc blende may assume commercial proportions."

(f). *Illecillewaet Mining Division.*—Two mineral claims situated in this mining division have been brought to the notice of the Commission by the Revelstoke Board of Trade as containing zinc in such quantities as to be commercially valuable. These claims are known as the Donald and Round Hill, and are situated at Flat Creek about one and one-half miles directly north of the Canadian Pacific Railway at an altitude of about 2,900 feet above the track. They are reached by a trail about three miles in length. The Donald claim is owned by David Woolsey, William Elson, and John Lawson, all of Revelstoke, while the Round Hill claim is owned by A. G. H. Howard-Potts, of Victoria, B.C.

The Round Hill adjoins the Donald to the northeast. The ore occurs in two parallel veins which run through both properties in a northeasterly and southwesterly direction. The vein on which most of the mining development work on the Donald has been done is stated to be 9 feet 8 inches in width with a dip, at the surface, of 71°, although in going down this dip flattens out to about 40°. The second parallel vein is situated 104 feet lower down the mountain and has a width of about 12 feet. The development work consists of a shaft 84 feet in depth and a tunnel 800 feet in length, which should reach the vein at a depth of 325 feet. The buildings include a cabin and a blacksmith's shop. The information from which the above description has been prepared was supplied by Mr. David Woolsey, of Revelstoke.

The following are the results of analyses of ore from the Donald mineral claim:—I. Analysis of a very pure specimen of zinc blende by Mr. M. F. Connor of the Geological Survey Department. The sample was received from an independent source. II. and III. are partial analyses by Mr. George C. Robbins of specimens of ore sent in 1903 to Mr. A. C. Gardé, then manager of the Payne mine.

	I.	II.	III.
Lead	0.91%	5.10%	——
Copper	None	——	——
Zinc	54.27	29.60	46.30
Iron	9.77	18.40	7.10
Insoluble	0.08	——	——
Gold	None	Trace	Trace
Silver	0.30 oz. per ton.	4.00 oz. per ton.	2.30 oz. per ton.

(g). *Lardeau Mining Division.*—The mining recorder of this division writes from Camborne that two of the silver-lead properties show a considerable percentage of zinc. The Sirdar mineral claim on Goat Mountain has 4 feet of zinc blende, assaying 52 per cent., while at the Beatrice mine on Mohawk Creek, where considerable mining operations have been carried on, there are 3,000 tons of zinc ore available, which will show an average assay of 33 per cent. of zinc. It is stated that most of the ore from this mine contains from 14 to 20 per cent. of zinc, some of the ore occasionally running as high as 35 per cent.

The mining recorder of Trout Lake Mining District writes that the Silver Cup and Nettie L. mines, owned by the Ferguson Mines Co., Limited, of which Mr. George Alexander of Kaslo is manager, are shipping ore which contains about 10 per cent. of zinc.

The ore of the Triune mine of which Mr. John Morton of Ferguson is manager, is said to contain about 15 per cent. of zinc. A very small percentage of zinc occurs in the Lucky Boy mine.

On the Old Gold, situated on the west fork of Duncan River and on which considerable development work has been done, a large percentage of zinc is said to occur.

An undeveloped property known as the "J.C.," situated at the head of Lake Creek, is also said to contain considerable zinc ore.

Mr. William Simpson writes from Howser that the Irene mine on which between six and seven hundred feet of tunnelling has been done, has a considerable body of ore made up of galena and zinc blende. This mine is the property of the Irene Mining Co., of Wallace, Idaho, and is situated on the east side of Duncan River about 10 miles above Healy's.

(h). *Steele Mining Division.*—Zinc occurs as an important constituent of most of the silver-lead ores of the East Kootenay district, but only in a few cases is it present in such proportions as to be worthy of consideration from a commercial point of view. One of the most promising of this class of deposits in which the zinc constitutes such a high percentage of the total metallic contents of the ore as to be economically separable, if any large amount of ore can be mined, is that covered by the Aurora group of mineral claims. These are situated on the west side of Lower Moyie Lake, opposite the town of Moyie on the Crow's Nest Branch of the Canadian Pacific Railway. They are owned by T. Rader et al., of Morrissey Mines, B.C. and during last summer were being operated under bond by a syndicate represented by Richard Wilson and Walter Mackay, of Portland, Oregon. This property consists of the Horseshoe, Portland, Etna, Durango and Aurora mineral claims (Ann. Rep. Min. of Mines of B.C., 1898, pp. 1010-11).

The rocks underlying these claims and the country adjoining are certain greyish shales and quartzites which have been described by Mr. James McEvoy as of Cambrian age. (Ann. Rep. Geol. Surv. Can., Vol. XII, 1899, pp. 91 and 97 A.)

In the immediate vicinity of the deposit, these measures dip to the west at an angle of nearly 25°. Throughout the neighbouring district these rocks are cut through by certain basic igneous rocks occurring both as dykes and masses.

Some of these intrusions seem to be directly connected with the segregation of the ore bodies, although certain dykes of basic igneous material cut across the lodes without apparently influencing them in any way. The vein on which the mining development work on the Aurora is proceeding has a strike of nearly east and west, with a dip to the south at a very high angle. This vein can be followed up the steep mountain side as far as it has been uncovered, but in places the walls come together with little or no vein matter separating them. At other places the fissure widens out and is seen to be made up almost wholly of galena and zinc blende with a varying proportion of quartz and a trifling amount of pyrite. No large or continuous ore body has yet been discovered, but a considerable amount of ore has been taken out and has been piled up ready for use. This ore has been roughly hand-picked, the pure galena being kept in one pile, while that portion of the ore in which zinc blende is a prominent constituent is laid aside for future consideration. The vein is regarded by many as a continuation across the lake of the St. Eugene vein, but considerably more development work would be necessary before this could be confirmed or denied. Some parts of the vein are made up of very solid and pure galena, with very little or no gangue material or blende, while other portions of the ore body are almost wholly composed of zinc blende with scarcely any traces of galena or quartz. Another type of ore, intermediate between these two, shows a very intimate admixture of galena and zinc blende.

Of the specimens brought to Ottawa, two were selected for purposes of analysis. These were handed to Mr. M. F. Connor of the Geological Survey Department, who furnishes the following results: I. Analysis of zinc blende containing a comparatively large admixture of galena. II. Analysis of massive and apparently pure zinc blende.

	I.	II.
Lead	41.61%	—
Zinc	23.12	—
Iron	4.49	—
Insoluble	8.70	57.28%
Gold	0.01 oz. per ton.	7.30
Silver	9.30 "	1.74

Zinc is always an important constituent of the ore met with on the Estella group of claims, situated about 18 miles northeast of Steele. This property belongs to Mr. Alex. Polson, president of the Polson Logging Company, of Hoquiam, Wash. His Attorney, Mr. J. A. Harvey, of Cranbrook

B. C., reports that about 3,000 feet of work has been done on this group, and that one ore chute, about 300 feet long, runs about 12 per cent. zinc. The mine is reached by a waggon road from Steele.

According to A. C. Nelson, Acting Gold Commissioner at Cranbrook, the Watson and Kootenay King on Wild Horse creek, owned by William Myers and William Voss, also carry a considerable amount of zinc. A good deal of development work has been done on this property, but it is still in the prospect stage.

(i). *Golden Mining Division.*—The presence of zinc in commercial quantities has been reported from the Monarch mine and the Giant mineral claim in this mining division.

The Monarch mine is situated on Mount Stephen about three miles east of Field, at an elevation of 850 feet above the main line of the Canadian Pacific Railway. It is one of the older mining propositions in the Province of British Columbia, having been discovered in 1884. As early as 1890 it had undergone extensive development, as noted in the report of the British Columbia Department of Mines (Ann. Rep. Min. of Mines of B.C., 1890, 372-373). The workings at that time included about 450 feet of tunnelling, with larger chambers opened up in places where the ore was more abundant. A tramway was galleried out of the face of the mountain leading to the principal ore-bins and a gravity road thence to the bins on the railroad. It is also reported that as a result of these mining operations about 1,500 tons of ore were shipped. An arrangement was also made to supply the smelter at Revelstoke with 200 tons of ore per month.

The ore is reported as occurring in somewhat irregular chambers, pockets and other deposits in limestones which are considered by geologists as of Middle Cambrian age. The ore is mainly galena varying from fine to coarsely crystalline, with a considerable amount of zinc blende and a very little gangue matter, made up chiefly of quartz and calcite. Small quantities of iron and copper pyrites are also present. The galena is low grade in silver containing, according to analyses made in the laboratory of the Geological Survey; from 3.281 oz. to 11.302 oz. to the ton of 2,000 lbs., with only traces of gold. (See Ann. Rep. Geol. Surv. Can., 1885, p. 24 M. Assays 19-22 both inclusive). W. Pellew-Harvey, of Vancouver, in 1891, obtained from $4 to $5 in gold from some of the samples assayed by him.

Mr. R. G. McConnell of the Geological Survey Department, who visited the locality in 1885, gives the following description (Ann. Rep. Geol. Surv. Can., 1886., p. 41 D.) "Near Field, Messrs. Coffman and Weitman have opened up the Monarch and Cornucopia claims on Mount Stephen and the former, especially, now presents a very favourable appearance, showing over six feet of solid galena. The ore here is deposited in what miners call a 'blanket-lode' and appears to impregnate a zone of interbedded calcareous rocks. It has been traced along the face of the mountain for several hundred yards and since I was there Mr. Pattie, of Carleton Place, by blasting out a trail around an almost vertical cliff has been enabled to explore it still further and reports the discovery of a nine-foot deposit."

At present the Monarch mine is owned by a Vancouver syndicate, represented by John Hendry as president and H. T. Coperley as secretary Mr. George Turner, writing to the Zinc Commission at the request of Mr. Coperley, states that the work done on the property consists of a tunne. 250 feet in length in which are a number of cross-cuts; an upraise started about 25 feet below and to one side of the mouth of the tunnel, taps it some 50 feet within. From these workings between 5,000 and 6,000 tons of ore have been taken, as nearly as I can learn, mainly from the tunnel. There is certainly a large amount of ore in sight, but its extent can only be told by development. The former owners of the property worked it as a smelting lead and silver proposition, sorting out all the zinc possible, but failed, as did a smelter at Revelstoke which had a lease from the present owners.

The following are analyses of Monarch mine ore,—I, II, and III were done by W. Pellew-Harvey, of Vancouver; IV by F. Roeser, of the Kootenay Smelting and Trading Syndicate, Ltd., of Revelstoke, in 1890; V and VI are analyses by F. Roeser of ore run through the Revelstoke smelter in July and August, 1891:—

	I.	II.	III.	IV.	V.	VI.
Lead	50.99%	50.80%	55.00%	60.00%	48.00%	47.00%
Zinc	7.20	—	6.98	10.10	15.80	15.90
Antimony	22.00	—	—	—	—	—
Iron	7.15	—	14.00	1.60	3.10	2.30
Sulphur	8.31	—	—	—	—	—
Silica	3.75	—	—	2.40	2.00	1.50
Lime	0.35	—	—	—	—	—
Water & loss	0.22	—	—	—	—	—
Gold	—	a$5.00	—	—	—	—
Silver	0.03	a 9.50	—	a3.00	a3.00	a3.00
	100.00					

(a) ounces per ton.

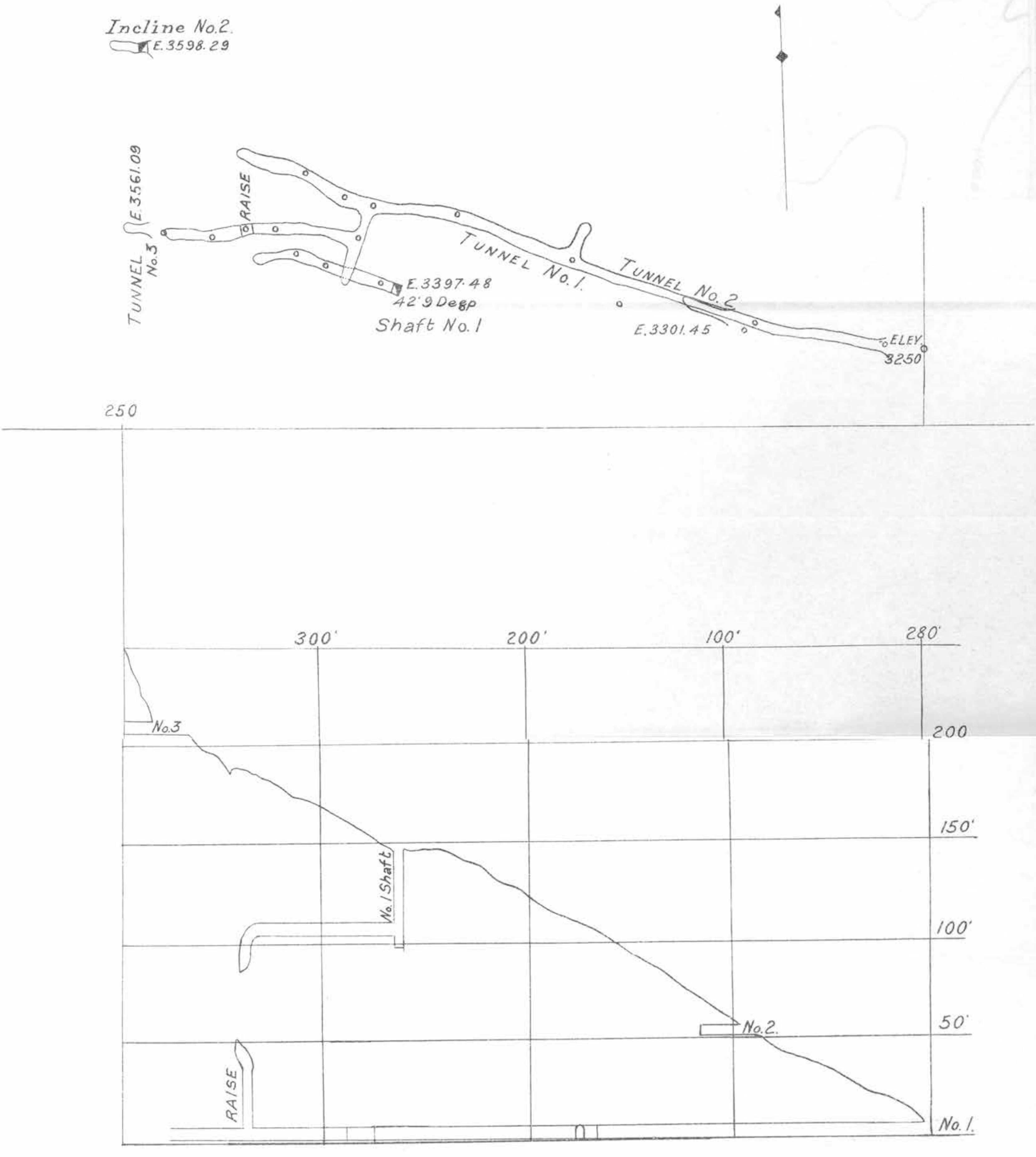
The Commission is indebted to Dr. H. M. Ami, of the Geological Survey Department, for a copy of the analyses of the Monarch mine above quoted.

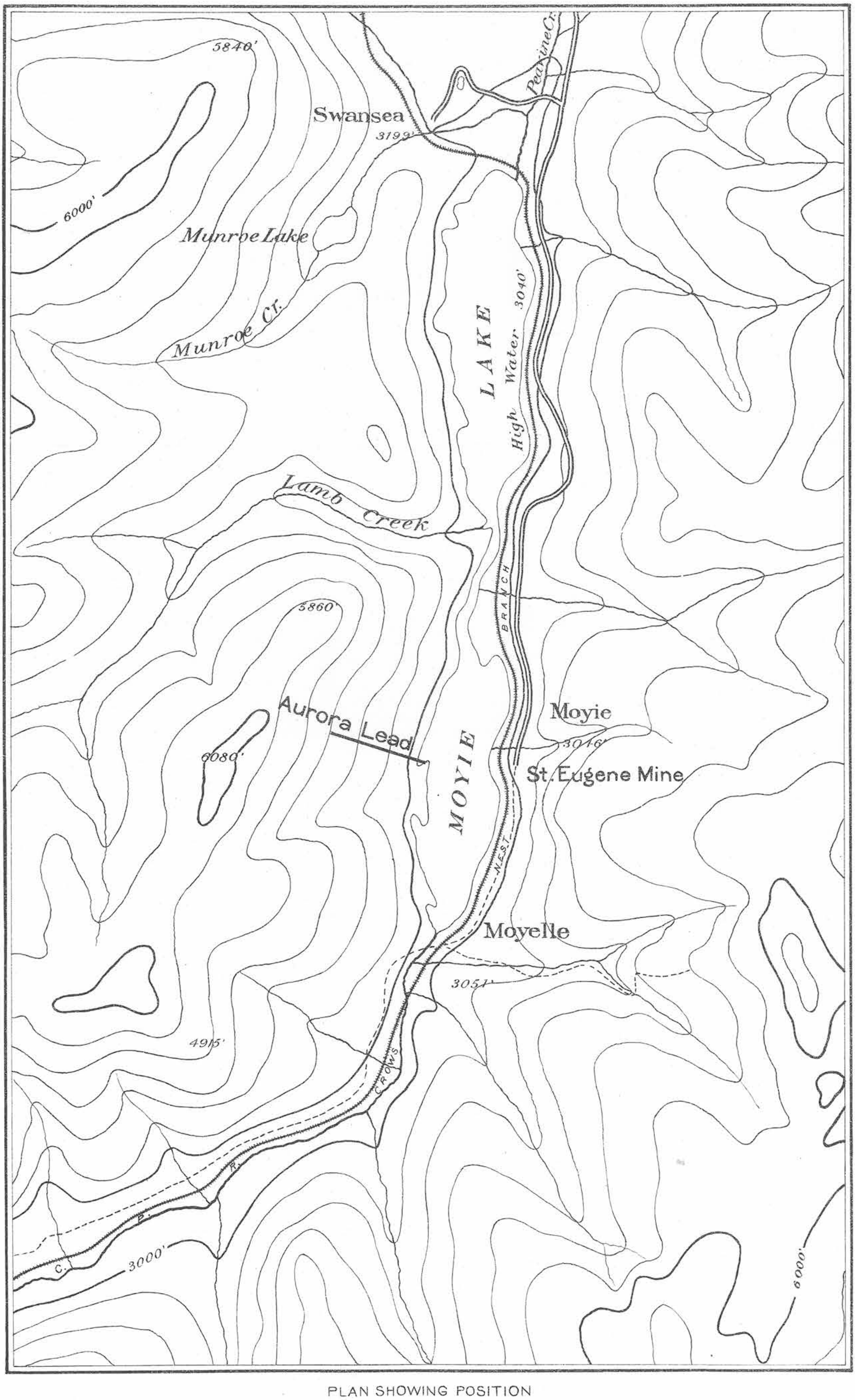
The Giant mineral claim is situated on Spillimacheen Mountain, west of the Columbia River, and 45 miles from Golden. It is readily accessible, the landing place being only about 6 miles from the workings, which are at an altitude of 500 feet above the river. The property belongs to Captain F. P. Armstrong, of Golden. Captain Armstrong states that the development work consists of a tunnel about 50 feet in zinc ore which averages 2½ feet in thickness. The assays show it to contain about 55.2 per cent. of zinc.

PLATE I

PLAN
OF THE
AURORA MINE
MOYIE, B.C.

Scale 60 feet to 1 inch





PLAN SHOWING POSITION

OF

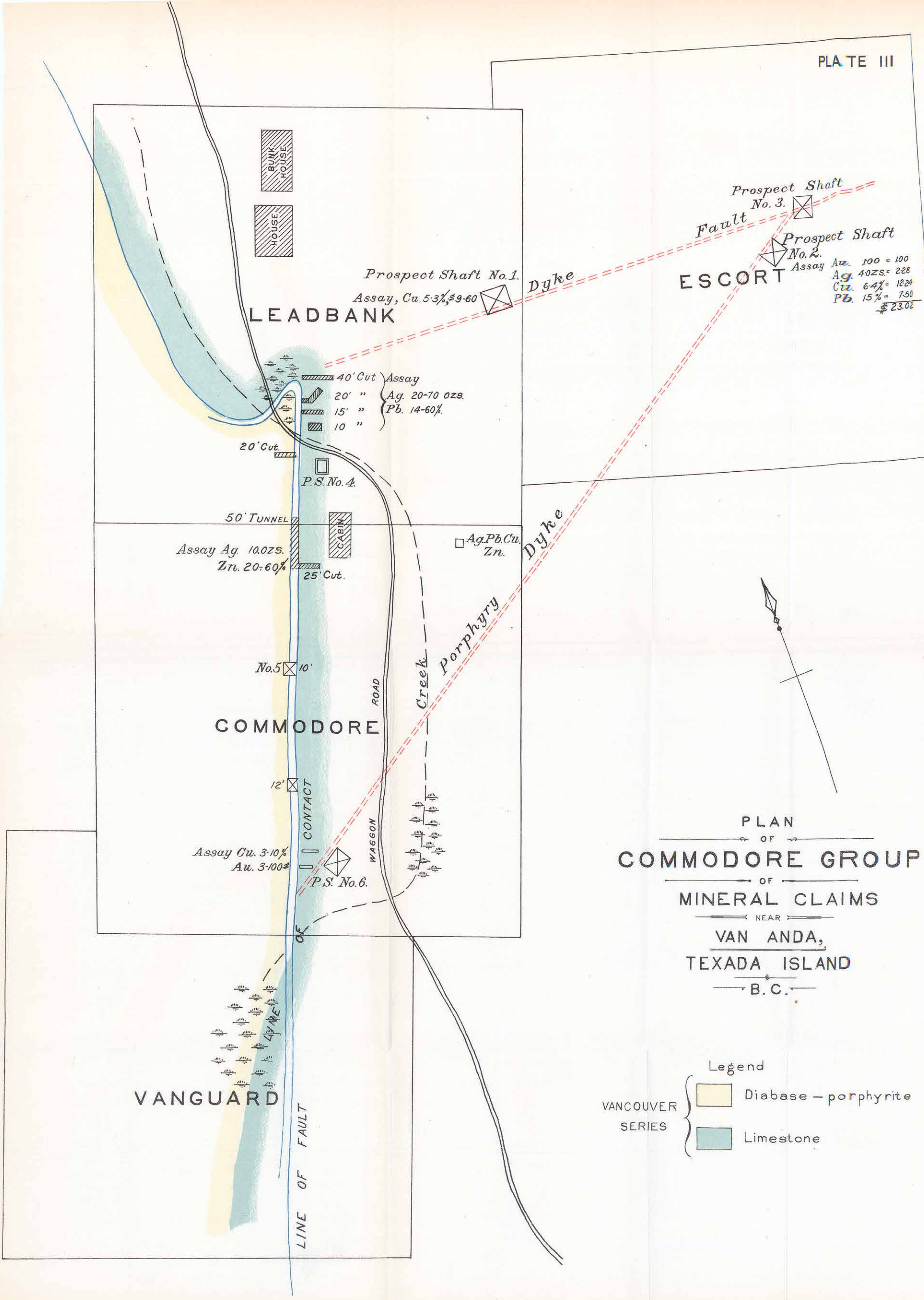
AURORA MINE

NEAR MOYIE, B.C.

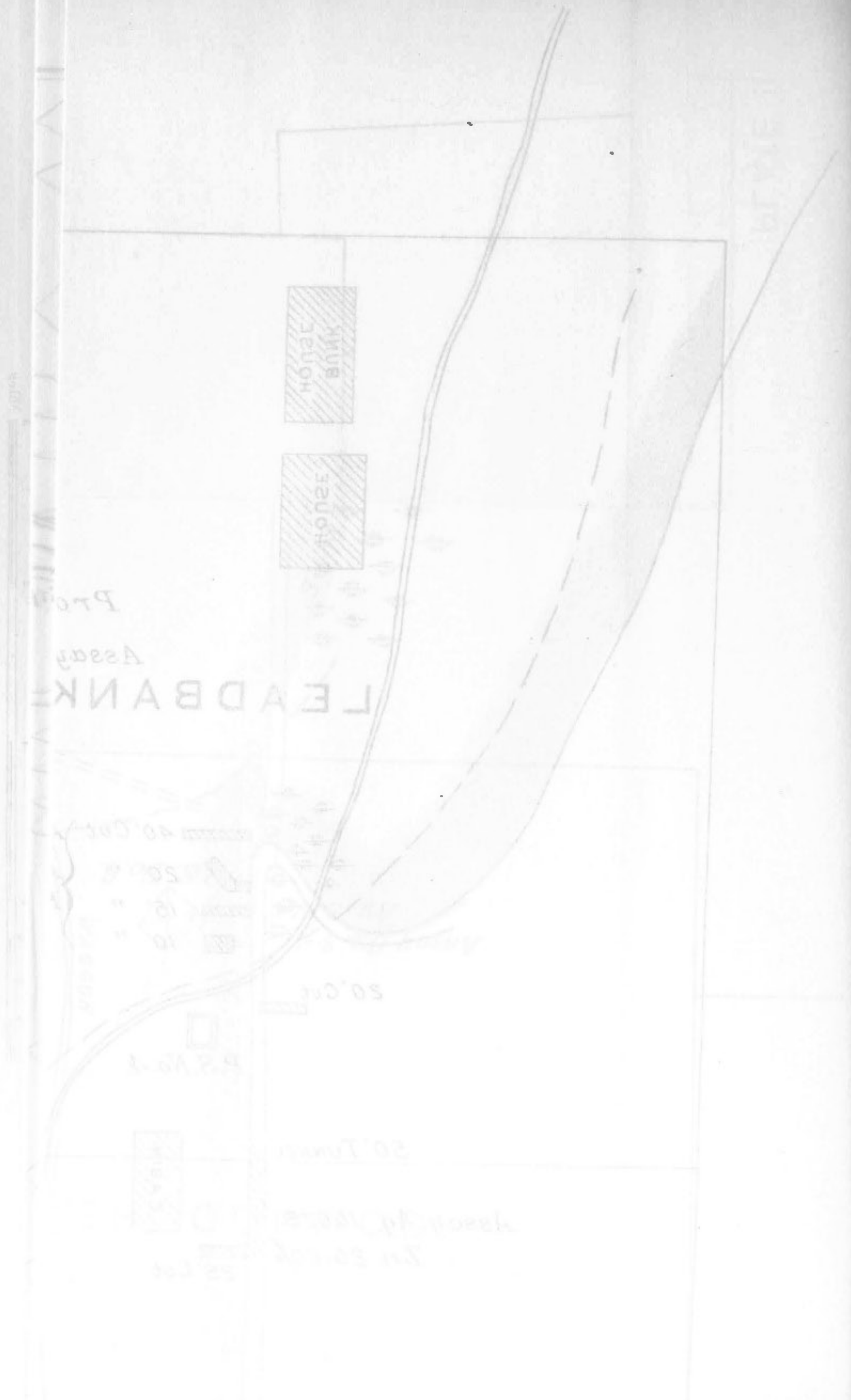
Scale 1 mile to 1 inch

1 ¾ ½ ¼ 0 1 2 3 Miles

PLATE III



BUNK HOUSE
HOUSE
LEADBANK
Prospect Shaft No. 1.
Assay, Cu. 5·3%, $9·60
Dyke
Fault
Prospect Shaft No. 3.
Prospect Shaft No. 2.
Assay Au. 1·00 = 1·00
Ag. 4·0 ozs. = 2·28
Cu. 6·4% = 12·24
Pb. 15% = 7·50
$ 23·02
ESCORT
40' Cut
20' "
15' "
10' "
Assay
Ag. 20-70 ozs.
Pb. 14-60%.
20' Cut.
P. S. No. 4.
50' Tunnel
CABIN
Assay Ag. 10.ozs.
Zn. 20-60%
25' Cut.
Ag. Pb. Cu. Zn.
Porphyry Dyke
Creek
No. 5
10'
ROAD
WAGGON
COMMODORE
12'
CONTACT
Assay Cu. 3-10%
Au. 3-100$
P. S. No. 6.
LINE OF FAULT
LINE OF CONTACT
VANGUARD
PLAN OF COMMODORE GROUP OF MINERAL CLAIMS NEAR VAN ANDA, TEXADA ISLAND B. C.
Legend
VANCOUVER SERIES
Diabase – porphyrite
Limestone



REPORT

ON

SOME OF THE UNDEVELOPED ZINC DEPOSITS

OF

BRITISH COLUMBIA

BY

ALFRED ERNEST BARLOW, M.A., D.Sc.

1906.

INTRODUCTION.

For many years it has been known that zinc was present in appreciable quantities in many of the silver-lead deposits of British Columbia, especially those of the West Kootenay district. In addition, as mining development proceeded, it was shown that in many of these mines the silver-lead contents were gradually replaced in depth by zinc until, finally, in some cases this latter metal became the chief constituent. As, however, its presence in excess of the smelter's limit of 10% subjected such ores to a penalty of 50 cents per unit, it can scarcely be a matter of wonder that in cases where at all possible no mention was made of this metal as a constituent, especially in those deposits which had, at best, only reached the preliminary development stage and were waiting for more capital before doing further mining work. This reticence on the part of prospectors, and even mine owners, has resulted in the fact that zinc has received very scant notice or attention on the part of Government and other reports which have treated authoritatively of the mineral resources of the Province.

In the earlier years of the silver-lead mining in the Kootenays, the zinc product obtained in the concentrating of these ores was allowed to escape down the creeks but latterly more attention has been given to the saving of this zinc, and, at many of the mines, such as the Slocan Star, the Ruth, the Jackson, and the Monitor, the mills were so constructed and remodelled as to produce both a zinc and a lead concentrate. In most instances, however, this concentrate assayed only about 37% of zinc, with comparatively low silver value, but even with this enrichment the product was too low in grade to permit of transportation to the far distant smelting establishments.

In view of the widespread occurrence of zinc in British Columbia, reported by various authorities, it was considered well within the scope of the work of the Zinc Commission to give descriptions and estimates, not only of the principal working mines which were believed to contain economically valuable supplies of zinc ore, but also to inquire as to those deposits which had undergone preliminary development work only, or were even of the nature of "prospects," and to secure such information as would enable an intelligent judgment to be formed of the general extent of the zinc resources of the Province. This portion of the work of the Commission was entrusted to the writer, but owing to the lateness of the season most of these "prospects" were inaccessible and their examination had to be postponed. The opportunity was taken, however, to travel over most of the field, in order to gather the information in regard to localities, facilities for transport, ownership of properties, etc., which is of prime importance in making an inquiry of this sort.

The so-called "prospects," as applied to the zinc deposits of British Columbia, may, from the standpoint of their geographical distribution, be considered as follows:—

1. VANCOUVER ISLAND.

(a) *Quatsino Mining Division.*
Peerless, Elk and I.X.L. mineral claims near Quatsino Sound.

(b) *Clayoquot Mining Division.*
Brown Jug mine on Hesquoit Lake, near Hesquoit Harbour.

2. TEXADA ISLAND.

(a) *Nanaimo Mining Division.*
Commodore group of mineral claims near Van Anda.

3. MAIN COAST, BRITISH COLUMBIA.

(a) *New Westminster Mining Division.*
Mineral claims at Lynn Creek camp, near Vancouver, B.C.

(b) *Nanaimo Mining Division.*
Broken Hill group of mineral claims, near Theodosia Arm, near Lund P.O.

4. INTERIOR, BRITISH COLUMBIA.

(a) *Similkameen Mining Division.*
Summit City camp, on Tulameen river.

(b) *Kamloops Mining Division.*
Lone Prospector and Iron Clad mineral claims, near North Thompson river.

(c) *Kettle River Mining Division.*
Silver Reef mineral claim, near Greenwood.

The Providence and Elkhorn mines both contain a small proportion of zinc, probably not more than 5-6%. The Last Chance, Crescent, and Don Pedro also contain about 4% of zinc, while the E.P.U. contains about 2%. Such small proportions of zinc however, could not, be economically saved.

Many of the mining claims on the west fork of Kettle river contain a considerable proportion of zinc and some of the shipments from Sally Group to the Trail smelter contained as high as 16% of zinc.

(d) *Revelstoke Mining Division.*
Some of the claims in the "Big Bend" (Columbia River) mineral district contain considerable zinc. The Silver Belle, one of the Silver Shield group on Keystone mountain, near Five Mile Creek, may be mentioned as one of these. Zinc also occurs on Downie Creek, but no claims have been staked. The J. & L.

group on Carnes Creek, and the Jordan group in the Jordan Pass, about twelve miles northwest of Revelstoke, are also stated to show considerable zinc.

(e) *Arrow Lake Mining Division.*

"Big Ledge," Bald Mountain, Pingston Creek, (opposite Halcyon Hot Springs).

According to R. W. Brock nearly all of the wet and dry ores of the Fire Valley region, as well as those of the area between Burton city and Slocan Lake, contain considerable amounts of zinc blende, but no deposits of commercial importance have been discovered.

On the Trio claim on Goat Canyon Creek, east of Burton, a vein two feet wide, containing zinc blende, arsenopyrite, together with some chalcopyrite, pyrite and galena was exposed for a short distance, but unless it develops unexpected strength along the strike it would have little value as a source of zinc.

(f) *Illecillewaet Mining Division.*

Donald and Round Hill mineral claims, near Flat Creek station.

(g) *Lardeau Mining Division.*

Zinc blende is a frequent accompaniment of the silver-lead ores of this division, and a limited amount might be obtained as a by-product, but no large deposit has yet been opened up.

The Beatrice mine, near Camborne, and the Sirdar on Goat Mountain may be specially mentioned in this particular.

The Ferguson mines, in the Trout Lake mining division, and the Mother Lode at Poplar, in the Ainsworth mining division, also contain a comparatively large zinc percentage.

(h) *Fort Steele Mining Division.*

Aurora group of mineral claims on the west side of Moyie Lake, opposite the St. Eugene mine.

Estella Group on Tracey Creek, eighteen miles north of Steele. The Watson and Kootenay King on Wild Horse Creek, twelve miles northeast of Steele.

(i) *Golden Mining Division.*

Monarch mine, near Field.

Giant mine on Spillimacheen mountain.

1. VANCOUVER ISLAND.

Reports of the occurrence of appreciable quantities of zinc blende on the western, or outer, coast of Vancouver Island have, of late years, attracted considerable attention, but very little was known in regard either to the exact location or probable economic importance of any of these ore bodies until after the visit of Mr. Herbert Carmichael, of the British Columbia Department of Mines, in July, 1903 (see Ann. Rep. Minister of Mines, 1903, pp. 196-203). Two of these deposits, the Peerless mine near Quatsino Sound in the Quatsino mining division, and the Brown Jug mine, on Hesquoit lake, near Hesquoit Harbour, in the Clayoquot mining division, are regarded as distinct economic possibilities, although the amount of development work so far accomplished would not justify any very definite or positive statement in this regard.

As the Peerless mine has been visited and described by Mr. Carmichael in the publication above referred to, and as no further work has been done on the claim since his visit, it was not thought necessary to make a further examination, especially in view of the fact that the steamer makes only one trip to Quatsino each month.

Quatsino Sound, which is the most northerly of the deep inlets by which the outer coast of Vancouver Island is dissected, is also one of the longest and most complicated in outline. It penetrates the island in an easterly direction for nearly thirty miles, its eastern extremity being only about six miles in a straight line from the northeastern coast of the island. The entrance from the sea is about thirty miles southeast of Cape Scott, the northwestern extremity of Vancouver Island. The breadth at the mouth is about five miles, narrowing to less than a mile at five miles within; the Sound then runs in a northeasterly direction (magnetic) nearly straight for thirteen miles, when it branches off in two arms, one extending to the southeast (Southeast Arm) for twelve miles; the other lies to the northward of, and is connected with the Sound by a straight narrow pass (Quatsino Narrows) about two miles long; the length of the part within the narrows is twenty-five miles in an east and west direction. The western end of this inner part of the sound is known as the West Arm, while that to the east is called Rupert Arm.

A short distance to the east of the Southeast Arm, and parallel with it, are two large and important sheets of water known as Victoria and Alice lakes.

Means of Communication.—Quatsino is most easily reached by means of the C.P.R. steamer "Queen City" from Victoria, which makes one trip each month to this northern portion of Vancouver Island.

A more difficult route, especially if the traveller is impeded by baggage, is by way of Hardy Bay, where connection is made with the steamers which ply along the eastern side of the island, some of which call at this port. A trail, through a practically level country and about twelve miles in length, connects Hardy Bay with Coal Harbour on the West Arm of Quatsino sound. The zinc deposits are situated on the divide between Victoria and Alice lakes. They are reached by trail by way of the June Group, this trail starting from "Patterson's" on the east side of the Southeast Arm about three miles south of Yreka.

The deposits are thus but a short distance removed from tide water, with which they could be readily connected by tramway.

Geology.—By far the greater portion of the northern part of Vancouver island is occupied by rocks of volcanic origin, to which the title of Vancouver Series was originally applied by Selwyn and Dawson. They may, in general, be referred to as "traps" or "greenstones", which have undergone rather extensive alteration and decomposition. The greater part of the series is built up of extensive lava flows, chiefly of porphyrite and diabase, much of which is amygdaloidal, alternating with volcanic breccias and tuffs largely composed of fragments derived from these. Interbedded with these volcanic rocks are certain limestones, argillites and quartzites, some of which contain fossils of Triassic Age. These are unconformably overlaid in certain places by conglomerates, sandstones and fine-grained calcareous beds, which are Cretaceous in age. A few small beds of coal, some of which is of good quality occur, but no coal seam of satisfactory workable dimensions has yet been located. (See Ann. Rep. Geol. Surv. Can., 1886, pp. 81-99B.). The strata of the Vancouver Series are tilted at high angles, the dips usually varying from 30° to 45° and occasionally 60°, while those of the Cretaceous lie comparatively flat, varying from 10° to 20°.

The basic igneous rocks of the Vancouver Series are penetrated by certain grey granites, the observed relations being very closely analogous to those which obtain between the so-called Laurentian granites and gneisses and the Keewatin greenstones of Eastern Canada. This granite is, in general, very uniformly fine-grained with, in places, a well marked foliation. Under the microscope it is seen to be porphyritic, the groundmass consisting of feldspar, quartz and chlorite, the last resulting from the alteration of hornblende, some portions of which mineral, however, still remain unaltered. A subordinate amount of pyroxene is usually present. Embedded in this groundmass, which in places becomes almost microcrystalline, are numerous large individuals of feldspar, chiefly orthoclase.

Economic Minerals.—Most of the mining locations are in the area occupied by the igneous rocks belonging to the Vancouver Series, and generally at or near the contact between these and the limestones of the same series. The prevailing minerals are chalcopyrite, pyrite, pyrrhotite, zinc blende, magnetite, bornite and arsenopyrite, carrying small gold and silver values. On the June Group of mineral claims the ore, which is made up of a mixture of chalcopyrite, bornite, pyrite, galena and magnetite, is in the porphyritic hornblende granite at its contact with the limestone of the Vancouver Series.

Occurrence of Zinc Blende.—Zinc blende occurs on the Peerless, Elk and I. X. L. mineral claims, which lie immediately southeast of the June Group. The largest amount of this mineral seems to be on the Peerless claim, on the western end of the range separating Alice and Victoria lakes. It is thus described by Mr. Carmichael (see Ann. Rep. Min. of Mines for B.C., 1903, p. 202).

"After crossing Link creek (which joins Victoria and Alice lakes) from the June Group, and at the eastern end of a slight ridge, open quarry work has exposed an ore body 30 feet wide of nearly solid zinc blende, mixed with a little quartz vein matter. The lode runs northwest and southeast dipping 60° to the northeast; the hanging wall is a diabase and the foot wall limestone. The quarry shows a face of ore from 9 to 10 feet high. Some prospect holes have been sunk in the ridge 200 feet to the west of the Quarry, the mineral at this point being principally arsenical iron in a quartz, with some blende."

The claim is owned by Julian Sutro, of Quatsino. Adjoining the Peerless to the east is the "Elk" mineral claim, owned by J. L. Leeson, of Quatsino, and located in the spring of 1903. A small prospect hole shows some blende and galena occurring between lime and diorite.

The "I.X.L." mineral claim adjoins the "Elk" claim and is owned by Samuel Bryce of Quatsino. This lode is an extension of the Elk lode and shows the same mineralisation.

Under I. is an analysis by Mr. H. Harris of the ore collected by Mr. Carmichael, a sample of which was kindly furnished to the Commission. Under II. is a partial analysis of the same ore by the British Columbia Department of Mines. Under III. is an analysis by Mr. M. F. Connor, of the Geological Survey Department, of a sample of zinc blende sent to the Commission by J. L. Leeson, of Winter Harbour on Quatsino Sound. It is probably from the Elk mineral claim, although no mention is made of the locality. It shows an association of zinc blende with a small proportion of arsenopyrite and a trifling amount of pyrite.

	I.	II.	III.
Lead	nil.	—	—
Copper	—	—	—
Zinc	35.00%	49.9 %	45.45%
Iron	12.00	—	14.50
Silica	34.00	—	0.34
Lime	nil.	—	—
Gold	(a) 0.01	(a) 0.015	—
Silver	(a) 0.60	(a) 1.20	—

(a) Ounces per ton.

CLAYOQUOT MINING DIVISION.

Reports of the occurrence of zinc in this division have come only indirectly to the Commission and lack confirmation from any authoritative source. They are said to occur on the north side of Hesquoit Lake. Hesquoit Lake drains into Boat Basin (Hesquoit Harbour) through a narrow gorge a few hundred feet long, which at low tide is not more than 10 feet wide and through which, at such times, there is a rapid outflowing current of nearly fresh water, while at high water there is a slight inward flow from the basin into the lake. At low tide there is a depth of 5 to 6 feet in the passage, while at high tide a boat drawing 12 feet can pass through the gorge, but at the outlet of the lake there is only about 3 to 4 feet depth. Hesquoit Lake is about 3½ miles long by 1½ wide, with rocky shores rising precipitously on all sides. Hesquoit Harbour, which lies between Nootka and Clayoquot sounds, is a small inlet where excellent anchorage for small-sized boats may be found. To the north of Hesquoit Lake and emptying into it by a fall and short connecting stream (a few hundred yards long), is Ike Lake which is approximately circular in shape and about half a mile in diameter. Ike Lake has about 50 feet higher elevation than Hesquoit Lake and discharges over the falls mentioned about 240 cubic feet of water per minute at the lowest flow, providing a small water power which could be cheaply utilized should it be required for the working of any property on the lake. A plentiful supply of timber is also to be had for all mining purposes or for fuel.

Geology.—The rocks exposed in the vicinity of Hesquoit Lake are limestone and diabase, representing a portion of the Vancouver Series. About 2½ miles up the lake from the outlet these two rocks come in contact with one another. (See Ann. Rep. Min. of Mines for British Columbia, 1902, pp. 208-209). Along this line of weakness certain minerals of economic importance have been deposited, chiefly magnetite, but also certain sulphides. The deposits of zinc known as the Brown Jug mine are believed to occur in this vicinity on the east side of the lake.

This deposit is described by an interested party as occurring in a vein 4 to 17 feet in width, composed largely of talcose matter in which "shoots" of zinc blende, together with a little iron sulphide, are sporadically developed. The ore is also stated to contain from 20 to 30% of zinc, with from $4 to $10 in gold per ton. The mineral claims covering this deposit are known as the Brown Jug Group and are owned by Messrs. Norris and Smith, of Alberni, with Mr. A. F. Guinn as their representative at Victoria.

OTHER OCCURRENCES OF ZINC ON VANCOUVER ISLAND.

Specimens of zinc blende were also noticed in the Museum of the Bureau of Mines at Victoria, which were stated to have come from the Red Rock mine at Quatsino. Another specimen of zinc blende in association with galena and iron pyrites was labelled "3 W's" mine on Granite creek in the Alberni mining district, but there is no considerable deposit at this place, according to Mr. Carmichael. Zinc blende was also met with at the Golden Eagle mine on China creek, about 20 miles from Alberni, in association with quartz and pyrite, but this claim has also been abandoned.

2. TEXADA ISLAND.

Texada Island lies in the strait of Georgia, between Vancouver Island and the mainland of British Columbia, being separated from the latter by

Malaspina Strait. It has a length of nearly 30 miles and averages probably five miles in width. The southern end of the island is about 50 miles from the city of Vancouver. The shore of the island is, in general, bold and precipitous, and the interior rocky and mountainous, leaving few available spots for agriculture. The only considerable settlement is at Van Anda (Marble Bay) on the north east coast of the island, near its northern extremity. This port, which is practically the only reasonably safe harbour on the whole island, may be conveniently reached at frequent intervals from Vancouver by the steamers of the Union Steamship Company.

For many years it has attracted the attention of mining men on account of the occurrence of numerous and considerable deposits of minerals which were considered of great economic importance. As a necessary consequence the northern part of the island was rather thoroughly prospected, with the result that the whole area was covered by numerous claims forming an intricate network of rectangular blocks disposed at all possible angles to one another. Most of these were ultimately abandoned, although rather extensive and continuous copper mining is still carried on near Van Anda.

This island, however, is chiefly famous to mining men from the occurrence of immense deposits of magnetic iron ore situated near the southwest shore of the island almost opposite Van Anda. These were long ago described by Mr. James Richardson (see Ann. Rep. Geol. Surv. Can., 1873-74 pp. 99-100) and by Dr. G. M. Dawson (see Ann. Rep. Geol. Surv. Can., 1886, pp. 36-39 B.) and also, much later, by Mr. W. F. Robertson (Ann. Rep. Minister of Mines of B. C., 1902, pp. 225-227).

Geology.—With the exception of a small outlier of Cretaceous on the southwest coast at Gillies bay, the whole of the island, so far as known, is underlaid by rocks of the Vancouver Series, chiefly limestones and breccias with certain intercalated basic igneous rocks, mainly porphyrites and diabases. The limestones are cut through by dykes of porphyrite and both the limestones and basic igneous rocks by dykes and masses of granite.

The hand specimen of this rock, secured as fairly representative of it, showed a light coloured and highly feldspathic type. Under the microscope it is seen to consist principally of feldspar with subordinate amounts of quartz and augite. Both plagioclase and orthoclase are present, the former with good crystallographic outline. The augite is of a pale yellow colour and is considerably altered to actinolite and epidote. The quartz is hardly in sufficient amount to be considered an essential constituent and the rock has, therefore, to be classified as a syenite rather than a granite.

Economic Minerals.—Besides the deposits of iron there are numerous occurrences of sulphides, chiefly chalcopyrite, bornite, galena, pyrite and zinc blende, most of these occurring at or near the contact of the limestones with the diabases, which is often a line of fault. In certain cases such deposits are developed at or near the junction with dykes of highly decomposed porphyrite, which cut the limestone at frequent intervals. Some of the deposits of the copper sulphides are so important that they have been mined to considerable depths, these mines being at present in active operation in the vicinity of Van Anda.

Occurrence of Zinc Blende.—The deposit of zinc blende examined by the writer is situated on the Commodore group, which embraces the Vanguard, Commodore, Leadbank and Escort mineral claims. These claims are controlled by W. Thomas Newman, of Vancouver, whose foreman, Robert A. Swan, resides in a cabin built almost on the line between the Commodore and Leadbank claims, about three and a half miles from Van Anda. A good wagon road, which crosses the island from Van Anda to the Texada Iron Mine, passes through these claims, thus rendering them easily accessible. With the exception of a comparatively short distance in the vicinity of Van Anda where the road is rather steep, there is a gradual ascent to these claims, the cabin, being situated on a small hill about 650 feet above the sea.

Geology of the Commodore Group.—The rocks which are exposed in the vicinity of these deposits are the limestone and diabase-porphyrite belonging to the Vancouver Series. The limestone has undergone rather extensive recrystallization, but certain streaks and patches still retain much of the original bluish-grey color. Superimposed upon this limestone is a diabase-porphyrite which in turn is overlaid by a breccia. The porphyrite shows a dark grey, almost black, groundmass in which are developed abundant pale greyish crystals of feldspar. Under the microscope the thin section shows the usual network of plagioclase laths with intervening areas of chlorite and partially devitrified glassy matter. The larger porphyritic individuals are plagioclase, which have been generally altered to calcite and sericite.

Occasional small dykes of porphyrite were also noticed cutting the limestone in various directions. The rock composing these dykes is evidently much decomposed. It is usually of a yellowish green color with abundant small pale greenish yellow porphyritic crystals. The thin section examined under the microscope shows the rocks to be a highly altered porphyrite, the porphyritic crystals of original plagioclase being almost wholly replaced by a granular aggregate of zoisite and epidote.

The contact between the limestone and overlying porphyrite crosses the Leadbank, Commodore and Vanguard claims in a South-Southwest direction, and is evidently a line of fracture and weakness, thus affording a channel for the mineral-bearing solutions.

The ore is chiefly galena with a smaller proportion of zinc blende, chalcopyrite and pyrite. No very extensive mining development has yet been accomplished, many of the test pits and trenches having failed to reach the underlying solid rock. On the side of the hill near the cabin a short tunnel has been run along the junction between the limestone and porphyrite, but although a certain amount of ore—chiefly galena—was secured no ore body or well-defined vein was encountered. At another spot, blasting has revealed the occurrence of ore, made up largely of zinc blende, but in this case also there were no indications of the existence of this mineral in such quantities as to be economically valuable. A sample of ore representative of the better quality of the ore raised was submitted to Mr. H. Harris for analysis, with the following results:—

Lead	8.80%
Zinc	29.20
Iron	3.20
Silica	20.00
Lime	8.90
Gold	0.03 oz. per ton.
Silver	15.00 oz. per ton.

3. MAIN COAST OF BRITISH COLUMBIA.

(a). *New Westminster Mining District.*—The occurrence of zinc blende on Lynn Creek—a small stream which flows into Burrard inlet almost opposite the City of Vancouver—was reported some years ago; and a representative specimen from this locality, contributed by Dr. G. M. Dawson, is contained in the "Systematic Collection" in the Museum of the Geological Survey at Ottawa.

The deposits are situated on several of the mineral claims, which together comprise what is known as the "Lynn Creek Camp." These claims, about 40 in number, only three of which, however, are Crown-granted, are situated at the headwaters of Lynn Creek, about 8 miles in a straight line, or 12 miles by trail, from North Vancouver. The area in the immediate vicinity of these claims is exceedingly steep and rugged, with abrupt valleys and gulches occupied by the small tributaries that serve as feeders to the main stream. The elevations vary from about 1,200 feet, in the valleys, to from 3,000 to 4,000 feet above the sea. As far as could be ascertained, the claims are controlled by W. Thomas Newman, Gideon Bower and T. J. Vaughan-Rhys, all of Vancouver.

Most of the mineral present in these claims is chalcopyrite, with some pyrrhotite, containing certain values of the precious metals.

It is stated that two bodies of zinc ore are exposed on the mineral claims known as the Kemptville and its extensions belonging to Gideon Bowers. One of these is about two feet wide; the other consists of about three feet of almost solid "black jack." Both appear to occur in a vein which follows a line of fault between a "greenstone dyke" and quartzite. The course of this vein is nearly north and south with a dip to the west at an angle of 60 degrees. The only mining work undertaken so far consists of two small open cuts about six feet in depth.

Mr. A. C. Gardé, who made a thorough examination of these claims in 1902, states that the amount of development done was not sufficient to demonstrate the existence, or otherwise, of any considerable deposit of zinc blende. The composition of the ore is indicated by the following analyses:

	I.	II.	III.	IV.	V.	VI.
	%	%	%	%	%	%
Lead	—	nil	—	nil	3.00	—
Copper	nil.	—	—	—	—	—
Zinc	40.00	42.80	45.20	34.20	30.60	44.00
Iron	5.50	5.70	3.90	4.30	2.60	6.40
Silica	33.00	—	—	—	—	—
Lime	—	—	—	—	—	—
Gold	nil.	(a)0.30	traces	nil.	(a)0.02	nil
Silver	trace	(a)0.41	nil.	nil.	(a)3.00	(a)1.50

(a) Ounces per ton.

I. Analysis of zinc blende from Kemptville mineral claim supplied to the Commission from an independent source. (H. Harris, Analyst). II. Partial analysis of "black jack" from Kemptville claim. III. Partial analysis of "resin jack" from same location. IV. Partial analysis of ore from Kemptville No. 2 claim. V. Partial analysis of ore from Evening Star claim. VI. Analysis of zinc ore from Kemptville claim. Analyses II. to VI., inclusive, were made by George C. Robbins from specimens collected by Mr. A. C. Gardé in 1902.

No development work has been done on these claims since Mr. Gardé's visit, but Mr. Newman informed the writer that arrangements had been completed to carry on extensive mining operations very early in the approaching season, with a view to demonstrating the extent and value of the deposits.

(b). *Nanaimo Mining Division.*—In addition to the occurrences of zinc blende in Texada Island a report from a reliable source reached the writer of the presence of considerable deposits of this mineral near Theodosia Arm on the mainland of British Columbia. The mineral claims covering this deposit are known as the "Broken Hill Group" and are owned by J. M. Mackinnon of Vancouver. They are situated about 5 miles inland from the head of Theodosia Arm, or 18 miles by water from Lund P.O. The steamers of the Union Steamship Company call at this port several times each week and the rest of the water journey can be made in a yacht which may be chartered at Lund. The claims themselves are reached by a very rough trail requiring about 5 hours steady walking. The ore occurs in a vein at the contact between limestone and granite showing, in a small test pit, from 12 to 18 inches of ore chiefly zinc blende, with a much smaller amount of chalcopyrite, pyrrhotite and pyrite. The vein runs about north 40 degrees west magnetic and is almost vertical. The development work consists of two tunnels, each about 50 feet in length and about 800 feet apart. One tunnel cuts the vein showing a mixed ore, chiefly zinc blende, scattered through the limestone. The other tunnel has not reached the vein.

The following is an analysis by Mr. H. Harris of the ore obtained from the person who supplied the above information.

Copper	2.50%
Zinc	29.00
Iron	7.60
Silica	16.00
Gold	trace.
Silver	2.30 oz. per ton.

4. INTERIOR OF BRITISH COLUMBIA.

(a). *Similkameen Mining District.*—A considerable quantity of zinc is an almost constant constituent of the silver-lead deposits occurring at the headquarters of the Tulameen River in the Summit City Camp. At present this camp is reached either by way of Hope or Granite Creek near Princeton, but the building of the Great Northern Railway through this district will bring it into more direct communication with the rest of the Province.

The ore is stated to consist mainly of galena and zinc blende with a much smaller amount of chalcopyrite. It is also said to be high grade ore, some of which is reported to have assayed 150 ounces of silver to the ton.

(b). *Kamloops Mining Division.*—The only information relative to the presence of zinc in this mining division is contained in two letters addressed to the Commission by Wentworth F. Wood, of Kamloops, B.C. He reports as follows:—

"Zinc occurs in many of the mining properties in this district and, I believe, in paying quantities in the northern part. The silver mines of Adams Lake all carry more or less zinc, but not in large quantities.

There are two claims especially mentioned as worthy of examination for their zinc contents. These are the Lone Prospector and Iron Clad, situated about 70 miles up the North Thompson River at Mosquito Flat. These claims are situated from 850 to 1,000 feet above the river and two miles inland. These properties are easily reached during the season of navigation, about three months in the year. Only one shipment was made to the smelter, where a charge was made for something over 16 per cent. for hand picked ore. The average ore would run nearer 30 per cent. of zinc. The vein of the Lone Prospector is 3 feet wide, a narrow stringer of galena running the length of the tunnel, 46 feet. The remainder of the "vein is quartz and zinc blende, with about 30 per cent. of zinc. A shaft has been sunk at the junction of a spur 30 feet or more, some very fine galena being thus secured. This ore was also high in zinc. A second shaft is down (on the spur) a little over 20 feet, but here the ore does not contain as much zinc. From these openings, in addition to a few open cuts, about 100 tons of ore have been taken out, which will assay about 20 per cent. of zinc.

"On the Iron Clad the tunnel is about 40 feet running with the vein which is about 4 feet wide, with ore which would average over 20 per cent. of zinc."

(c) *Kettle River Mining District.*—The presence of a very small proportion of zinc in certain of the high grade ore bodies in the vicinity of Greenwood has already been mentioned in the introduction, but the occurrence of this metal in such small quantity and with such associations may, from an economic standpoint, be altogether disregarded. In addition, however, the attention of the writer was directed to a deposit of zinc blende on the Silver Reef Mineral Claim about 6 miles north of Greenwood. This location is situated on the side of a hill about 500 feet above Wallace Creek, and is reached by a trail about three quarters of a mile in length, starting from the Wallace ranch. It is owned jointly by J. W. Nelson, Thomas Edwards and William George of Greenwood.

The ore occurs in a rather ill-defined brecciated vein, at the contact between a basic igneous rock and a quartzite, dipping to the southwest at an angle of 80 degrees. Zinc blende is the most abundant mineral present, but a little chalcopyrite and pyrite also occur, with quartz as the cementing material. Besides a small test pit which shows no ore, a shaft has been sunk on the incline of the vein to a depth of about 35 feet. The development work so far accomplished has not shown the existence of any ore body which could be regarded as of commercial importance.

The following is an analysis by Mr. M. F. Connor, of the Geological Survey Department, of a specimen of what may be considered the best quality of ore that was secured. The specimen showed zinc blende with a little chalcopyrite and a considerable quantity of quartz and rocky matter:—

Lead	trace.
Copper	1.20%
Zinc.	38.75
Iron	5.62
Insoluble	27.10
Gold	traces.
Silver	0.25 oz. per ton.

(d) *Revelstoke Mining Division.*—Zinc is reported to occur in considerable quantities in some of the claims already located in the "Big Bend" (Columbia River) mineral district. This district is easily reached during the season of navigation, a steamer making frequent trips up the Columbia River as far as Death Rapids, about 45 miles above Revelstoke. A trail also runs from Revelstoke along the east bank of the river and from this main trail branch trails have been made following up the valleys of most of the principal creeks which enter the Columbia River from the east.

One of these deposits of zinc blende has been located on Downie Creek (40 miles above Revelstoke) about 9 miles from the mouth. No claim has yet been staked and no details in regard to its size or mode of occurrence are available. Under I. is a partial analysis by George C. Robbins, chemist of the Payne mine, of a specimen from this deposit sent to Mr. A. C. Gardé in September, 1903. Another specimen handed to the writer by James T. Woodrow, of Revelstoke, has been analysed by Mr. M. F. Connor of the Geological Survey Department, with the results as given under II.

	I.	II.
Lead	—	14.84%
Copper	—	—
Zinc	43.60%	31.15
Iron	9.70	11.30
Insol	—	12.40
Gold	—	none.
Silver	5.6 oz. per ton.	4.55 oz. per ton.

Zinc is also said to be an important constituent of the silver-lead ore of the Silver Belle mineral claim, one of the Shield Group on Keystone Mountain. These claims are situated to the north of Five Mile Creek, about 40 miles from Revelstoke. An analysis of a specimen sent to Mr. A. C. Gardé of the Payne mine in September, 1903, by the chemist, George C. Robbins gave the following results:—

Lead	20.50%
Zinc	36.00
Iron	16.30
Silver	14.4 oz. per ton.

(e). *Arrow Lake Mining District.*—The "Big Ledge" situated on the west side of the Upper Arrow Lake has been reported as probably the largest zinc deposit in the Province of British Columbia. According to the mining recorder—Walter Scott of Nakusp—this deposit extends the length of 23 claims. These are situated on Bald Mountain on the west side of Pingston Creek, almost opposite Halcyon Hot Springs, about 3,000 feet above the upper Arrow Lake. They are reached by a wagon road about 8 miles in length, starting from the mouth of Pingston Creek.

The claims mentioned by Mr. Scott, on which the deposit is well exposed, are the White Heather, Empress, Delanger, Anna S., Maple Leaf, Ontario, Forest Chief and Monarch, which belong to Messrs. Savage, Symons and others.

Mr. R. W. Brock, of the Geological Survey Department, visited these claims in 1898, spending, however, only a few hours in the examination of the Excelsior and Iron Cap claims. His description is as follows:—

"The rocks are crystalline schists and limestones, cut by gneissic granite. The "Big Ledge," where seen, occurred in a quartzose schist. Near the deposit the rocks are highly decomposed. On the east side of the ledge a massive-looking rock, which might be a decomposed granite, appears to cut the schists. The ledge consists of a considerable width of solid sulphides, pyrite, pyrrhotite, sphalerite, galena and chalcopyrite, with blebs of quartz and grains of a green mineral, apparently hornblende. The contact with the schists is indefinite, the sulphides gradually becoming less abundant and occurring as almond-like patches, then finely disseminated in a highly silicious matrix and finally giving out. Where massive the

SIR:—

I beg to submit herewith my report on certain mines and prospects, in the Slocan and Ainsworth Camp, examined by me independently at the request of Mr. Philip Argall, chief engineer of field work in connection with the investigation of the Zinc resources of British Columbia.

Respectfully yours,

ALFRED C. GARDE,

Ass't. Engineer in the Field.

WALTER RENTON INGALLS, ESQ.,
Chief Commissioner,
New York.

THE SLOCAN.

The group of mines on Reco Mountains includes several well-known silver-lead mines, situated at elevations ranging from 5,000 to 7,000 feet above sea level, on the southern slope of the steep mountains directly north of Sandon and Cody. Of these properties I visited the American Boy, the Noble Five, the Reco, and the Goodenough—Grey Copper. My examination was limited to the properties wherein the production of zinc blende promises to become of economic value.

Broadly speaking the veins of this district parallel each other at distances generally not over 500 feet. The formation consists of the usual Slocan slate series, rather thinly bedded and frequently interrupted by intrusive dikes older than the veins and varying in thickness from a few feet to several hundred feet. The general strike of nearly all the veins is very much the same throughout the Slocan, viz., from 30 to 60 degrees east of north, dipping at angles rarely less than 50 degrees toward east, and more frequently approaching the vertical. Bearings stated in the following refer to astronomical north. The deviation varies in the Slocan from 24 to 25 degrees to the east.

AMERICAN BOY MINE.

There are seven levels on the property, but only four of these are being worked at the present time. I went through Nos. 3 and 4; also a raise connecting them. The vein can readily be followed, having strong walls and being generally well defined. In places it widens out to a thickness of several feet, but in the two levels visited it averaged about two feet. All levels are driven along the vein, which dips at an angle of about 50 degrees to the southeast, and has a strike of about 40 degrees east of north. In common with the other mines of the same vicinity the principal mineral production has been a high grade silver-lead ore, containing some blende. Occasionally lenses of rather clean blende are met with. During 1905, 170 tons of zinc ore was mined and hand-sorted from the galena and shipped to the Prime Western Spelter Company of Iola, Kansas. Mr. Thomas McGuigan, manager of the mine, stated that the average grade of the zinc ore shipments had been from 45 to 47% zinc, with about 15 ounces in silver to the ton. The first 130 tons was fairly low in lead, and fetched $11 net per ton, while the last 40 tons contained 8 per cent. lead, on which a penalty was levied by the smelter.

In the floor of No. 3 level I took a sample (No. 34) in two cuts, showing 18 inches of blende in the floor and 21 inches above this place in a raise. This was pointed out as one of the best showings in the mine. The sample assayed 39.8% zinc and 3 ozs. silver per ton. The extent of the lens of ore disclosed at this point could not be determined with any degree of accuracy from the present developments. I was informed by the manager that in addition to the ore body referred to there were others showing in the lower levels. All shipments are brought down to Cody over the Noble Five ærial tramway, which is leased for the purpose and is only a short distance off.

While the American Boy has never been a heavy producer, its output of high grade silver-lead ore has for several years been a steady one. It is only within the last year or so that the management has taken up the zinc question, in which considerable interest is displayed.

NOBLE FIVE GROUP.

This group is owned by the Hon. James Dunsmuir of Victoria and adjoins the American Boy on the east. On account of litigation the property has not been operated for several years, but at one time it produced a considerable tonnage of high grade silver-lead ore, which in some cases changed into zinc blende. On one of the claims, called the Dead Man, three levels have been driven on a 2 to 3 ft. vein. The strike of the vein is 55 degrees east of north, dipping at an angle of 70 degrees to the southeast and flattening somewhat in a winze sunk from the second level. This level, being the only one of interest from the zinc point of view, was examined. As will be seen from the sketch (Plate I) accompanying this report, it was first driven 85 feet along a slip in the slate formation. At this point a cross-cut was made into the hanging wall toward the east for a distance of 20 feet, where the main vein was encountered, and explored in a north-easterly direction for a distance of 120 feet. Next to the hanging wall and along the floor of this drift a lens of zinc blende averaging 12 inches in thickness has been exposed for nearly the whole distance. About 20 feet from where the vein was first encountered a winze has been sunk to a depth of 100 feet, proving the existence of the ore body all the way down, but decreasing some in size at depth. About half-way down the vein flattens out to an angle of 55 degrees and here the winze fails to make connection with the lower level, No. 3.

In this winze I took a three-cut sample (No. 35) averaging 12 inches in width. It assayed 55.6% in zinc and 15.1 ozs. in silver and represented the best part of the lens. The vein itself is from 2 to 3 feet wide, and a little galena is occasionally mixed with the vein matter. The zinc, occurs, however, essentially as a clean blende. From present developments no considerable tonnage of ore can be estimated, but I believe that the showings warrant the owner in developing the property and blocking out the ground between Nos. 2 and 3. The first step should be to effect a connection between these levels, and then explore further by raising on the vein from below. Present work on the property was done some time ago by lessees who extracted all the galena in sight. They discontinued working as soon as the ore changed into zinc and left the mine in anything but a workmanlike condition. I noticed a few inches galena in one or two of the short drifts running off from the winze in a southerly direction. Such places could conveniently be developed in connection with prospecting for zinc.

The Noble Five properties are opened through the third tunnel of the Last Chance mine. This tunnel is of considerable length, and enters the portion of the Noble Five ground, which is known as the World's Fair claim. A dispute in regard to this ground has existed between the two companies, but is now being worked by them jointly. In this ground there is a stope from which the Last Chance mine recently shipped two carloads of ore assaying 52% zinc. This stope, which is immediately above the tunnel, is approximately 9 feet high and 50 feet long. It appears to have been an isolated lens of blende. Its presence was known ever since the tunnel was driven, several years ago, but the ore was not removed until zinc became of value as a shipping product. It is possible that similar lenses will be met with in depth in the ground referred to, and there are indications of such at a point further on and nearer to the face of the tunnel, where a winze is being sunk on the vein at an angle of 65 degrees. The ore uncovered here appears to be more of a concentrating nature, and is a mixture of galena, blende and slate. The discovery is of such recent date that no trustworthy opinion as to its value can yet be expressed.

Exposures of zinc ore were reported on other claims belonging to the Noble Five group, in which, however, the workings are now unsafe on account of caving in, and consequently these were not examined.

Besides having good accommodations for the men working at the mine, the property is equipped with a 100-ton concentrator at Cody, which is connected with the mine by an ærial tramway. The tramway is at the present time under lease to the operators of the American Boy mine. The concentrator, which is now idle, is not arranged for saving zinc ore, and would require remodeling, but it is conveniently situated for handling concentrating ores from various adjacent properties, and the machinery is being kept in good repair.

RECO MINE.

This property, which adjoins the Noble Five on the east, is one of the oldest and best known of the local mines. It consists of five Crown granted mineral claims, or a total of 150 acres. I was informed that the company at the present time was not attempting to hand-sort any of the zinc ore associated with the high grade galena. On account of the blende containing high silver value, it is found more profitable to leave it with the galena, even if it be at times necessary to incur a penalty on the excess of zinc. Past experience with zinc ore shipments to Swansea, Wales, was very discouraging. For a 67 ton lot of blende containing 50% zinc and 99.5 ozs. of silver to the ton the Hafod smelter refused to pay anything for the silver content. This was in 1898, but even at the present time the prices offered for high-grade silver-zinc ores in the Slocan are by no means satisfactory.

One of the Reco veins (No. 3) which is narrow but of very high grade, was worked in connection with the adjoining property (the Goodenough) for some time, on account of the vein running into the latter. The description of this vein further on will therefore pertain to both properties. The Reco has produced a considerably larger quantity of ore from it than the Goodenough, and is being operated extensively at the present time. There are three veins in the Reco property, which strike parallel to each other. About twenty-five men find steady employment. The property is one of the constant dividend payers of the Slocan and so far has distributed approximately $300,000.

Ore shipments are handled over the Reco trail to the railway siding —a distance of four miles—in a unique and cheap manner, during the winter season, this being "sliding" or "rawhiding" on the snow. A one-ton parcel of ore, consisting of about one dozen sacks, is wrapped and laced into a raw cow-hide, which is dragged by one horse down the mountain trail, which has a down-grade of about 17 per cent. Besides the raw-hides the same horses, on their return trip, pack provisions and supplies to the mine. Two men are able ordinarily to attend to twelve horses; occasionally a few more. Rough-locking is done with common log-chains. A hide commonly lasts one season, but if well taken care of and provided with wooden runners, it will last longer than that. This method of transporting ore is also employed by other mines around Sandon, located similar to the Reco. In a ruggedly mountainous country, such as this, where the snow-fall attains a depth of several feet each season, a more economical way of handling galena ores in small quantity could not be introduced. Tramming, by means of gravitation, is certainly cheaper, but it involves a considerable investment, which is seldom warranted where the tonnage is small.

GOODENOUGH MINE.

(Grey Copper Claim). This property owned by the Goodenough Mines, Limited, is situated to the east of the Noble Five group and adjoins the Reco mine to the South. It consists of the Grey Copper claim, which is of full size (600 x 1,500 feet) and two fractions, the Goodenough and the Purcell, all Crown-granted, having a total of 50 acres. It has two parallel veins, of which the upper one on the Goodenough is known as the continuation of the Reco vein No. 3, in conjunction with which it was worked from 1894 to 1902. During this period 450 tons of hand-sorted galena was shipped, averaging 45% lead, 2% zinc and 300 ounces in silver to the ton. The average thickness of this vein is only 8 inches and its greatest thickness 30 inches. Four levels, respectively, 66, 225, 600 and 775 feet long, have exposed the vein to a vertical depth of 450 feet, with a total stoping area of 3,300 square feet. Since 1902 the property has been shut down, but during the last year the company has resumed development work on its second vein, which is known as the Grey Copper. This vein, while so far not productive of the same high grade of ore as the upper vein, has the advantage of being considerably wider and more regular. It promises to become of importance as a zinc producer.

From the sketch showing the Grey Copper workings (see Plate II) it will be observed that there are two levels, of which the upper one is 50 and the lower one 120 feet long. Those levels have been driven on the vein which is from 5 to 6 feet wide, and crops plainly on the surface. The strike of the vein is north 55 degrees east. The dip is to the southeast at about 70 degrees. The vein cuts through a large porphyry dike at nearly right angles, and has in every respect the appearance of a well defined and true fissure. The porphyry dike can be followed across the Grey Copper, Texas and Deadman claims and has a width of nearly 1,000 feet. Above the dike the usual slates and shales make their appearances. They have a bedding-strike of about northwest and southeast, and can be seen on the surface as well as in the workings of the upper Reco-Goodenough vein. It is expected that an additional 50 feet of tunneling will take the Grey Copper vein into these slates, and it will be of much interest to observe what influence this change will have on it. The same grade and character of ore is found in both levels, but the pay-streak in the lower one is twice the size of that in the upper one. In the upper level the pay-streak averages 12 and in the lower level 24 inches in width. Approximately 1,000 tons of ore has been blocked out on three sides between the two levels, which are 85 feet apart, measured on the dip of the vein. A five-cut sample (No. 36) was taken in the lower tunnel. It represented an average of 24 inches in width and assayed 42.6% zinc, 18.8% lead and 33.2 ozs. in silver to the ton. As will be seen from the above analysis the ore is of a heavily mineralized character, and requires to be separated more than to be concentrated. Hand-sorting would be of little use, unless it were followed by concentration of the "sortings". A lot of 40 tons of the ore, which was extracted while developing the levels, was recently tested at the Payne concentrator, near Sandon. The following data on the results of the test were kindly furnished me by the managing director of the company, Mr. J. A. Whittier:—

	Silver.	Lead.	Zinc.
Assay of original ore..........	17 ozs.	4.6%	41%
Assay of lead product.........	100 ozs.	61.0%	13%
Assay of zinc product.........	12 ozs.	1.5%	50.4%

Mr. Whittier was unable to furnish me the exact weights of the different products, wherefore the efficiency of the process can not be calculated.

The zinc product of this mine has been contracted to the Canadian Metal Company, of Frank, Alberta.

A sample of the average ore was secured for testing at Denver, Col.

By trail, the Grey Copper camp is four miles from the nearest railway shipping point (Reco Siding). The present cost of transporting ore by means of pack horses is $3 per ton, but with a larger output the rawhiding system will no doubt be introduced during the winter season, thereby lowering the cost of transportation materially. The mine crew consists of five men, engaged in driving the two levels ahead.

From its upper vein, the Goodenough Mines, Ltd., extracted at one time $80,000 worth of galena and paid in dividends $45,000. while the second vein on the Grey Copper claim is still only in a prospecting stage, it is very promising, and is the most interesting prospect in the locality refered to.

AINSWORTH DISTRICT.

UNION MINE.

A prospect in Ainsworth, known as the Union, is located slightly to the northwest of the United and Glengarry properties. The vein on this claim appears to strike north and south, bedded between quartzite and slate, and dipping at an angle of about 45 degrees. On the foot-wall there are several inches of crystalline calcite next to the quartzite. A prospecting shaft has been sunk on the vein, which is stated to be 35 feet deep. This shaft was partly filled with water and could not be examined. The vein at the collar of the shaft shows a width of five to six feet. A fair amount of zinc blende and galena is mixed through the vein matter, and a quantity of the material excavated from the shaft can be seen on the dump. On each side of the shaft two or three open cuts have been made on the vein, showing it to be continuous for a distance of about 200 feet. From all appearances it must have been a good many years ago since the first assessment work on this claim was recorded.

The claim is owned by FREDERICK MCLEOD of Ainsworth, B.C.

BUCKEYE MINE.

This is situated at an elevation of about 2,500 feet above Kootenay Lake in the northern mineral belt, and like many of the properties in Ainsworth has not been worked for several years. It is approximately two and a half miles from the town. For the first one and one-half miles an excellent wagon road, passing by the Highland mine is followed. The remainder of the distance is by way of a fair mountain trail.

Development work on the Buckeye consists of two inclined shafts one hundred feet apart, each about 40 feet deep, and one tunnel 200 feet long driven in under the shafts. The surface showing of zinc ore is considerable, but the work done does not seem to have been carried sufficiently far to expose the ore at depth. The two shafts are located on a northeast and southwest line, while the trend of the vein appears to be more north and south. There was too much water in both shafts to permit examination of the bottom. To the south of the first one a distinct mineralization is visible on the surface. The second shaft was started outside of the vein, with a view of intersecting it at a depth of about 70 feet, but it was never sunk to that depth.

The tunnel, which is about 75 feet below the surface showings, was driven as a cross-cut for 70 feet. At that point a body of zinkiferous ore has been intersected and followed for forty-five feet. The ore body only shows in the roof and has not been raised upon. Drifting in the tunnel was continued for an additional 150 feet through country rock, when a second shoot of zinky ore was encountered at the breast, where it can be seen. This exposure appears to correspond with the principal surface showings and seems worthy of attention. In order to learn its extensions the tunnel should be continued. The work was evidently left immediately after ore was broken into, as it was considered of no value by the owners, who at that time were looking for clean silver-lead ore, and not for a matrix of zinc and iron ore with more or less galena mixed through it. A sample of the face (top and bottom), taken on the vein for width of 18 inches assayed 23 per cent. zinc, but carried less than 1 oz. silver to the ton.

The property is owned by W. C. DALGLISH of Slocan City.

GALLAGHER MINE.

This is a Crown-granted claim, also situated in the northern mineral belt, one-half a mile beyond the Buckeye and approximately 3,000 feet above Kootenay Lake. The distance from Ainsworth is about three miles. It was at one time a producer, and according to the owner, Mr. A. D. Wheeler, 300 tons of lead carbonate ore, carrying silver, were shipped during the early days of the camp. These shipments were obtained immediately from a surface deposit, and a good deal of the same class of material is still scattered around the workings.

Close to these and apparently following the mineralized zone a vertical shaft has been sunk to a depth of 60 feet. At 30 feet below the collar of this shaft a level has been driven 60 feet toward the west. At the breast of this drift a fair amount of zinc-lead-iron ore can be seen. At the bottom of the shaft another drift has been run for about the same distance, and the same ore body broken into. A sample (No. 60) was taken here for a width of two feet, representing the best portion of the mineralization. This sample assayed 22.7% zinc, 3.2% lead and 24.6 ozs. silver. On the surface the vein shows a width of 5 feet, but it is difficult from present developments to connect this surface showing with the ore sampled in the lower level. The strike of surface deposit is approximately north and south and it occurs in a limestone country rock.

NELSON DISTRICT.

MOLLY GIBSON MINE.

This property, owned by the La Plata Mines Company, is located near the divide between Nelson and Slocan mining divisions, immediately at the head water of Kokanee Creek. By wagon road the main camp is ten miles from the nearest shipping point, which is on Kootenay River, and is known as Molly Gibson Landing. The mine is tributary to Nelson; the landing is approximately 12 miles from that town by water.

The group consists of four full size mining claims and two fractions, all Crown-granted, representing a total of about 240 acres. The lower or main camp, where the office and principal buildings are, is at an elevation of 4,600 feet, while the No. 5 level of the mine is 2,400 feet higher, or approximately 7,000 feet above sea-level. These terminals are connected by an aerial tramway, 8,000 feet long, over which all shipments are handled to and from the mine. Ample accommodations for the men are found at both terminals. The wagon road from the steamer landing to the lower mining camp is well built and of uniform grade.

The mine is located entirely in a massive, crystalline grey granite, known as the "Nelson granite." which contains large crystals of feldspar. This rock is readily traced to the shores of Kootenay Lake. The veins have a general strike of 30 to 40 degrees west of north and dip to the southwest at an angle of about 75 degrees. The vertical depth from the outcrops to the lowest point of development is approximately 800 feet. There are 5 main and 3 intermediate levels, of which levels No. 4 and No. 5, and two intermediate ones are by far the most important workings and represent a vertical depth of over 300 feet. My examination therefore covered these in particular.

A total of about 3,500 feet of tunneling has been done along the veins, and in addition thereto 750 feet of raises and winzes. Two veins have been developed, viz., the Florence and the Aspen of which the former is the larger and more important. It averages from 4 to 5 feet in thickness, while the Aspen is less than one-half of that width. The policy of the management has been to develop the property in preference to making shipments, and the tonnage shipped so far has therefore not been very large. During 1905 the monthly production has been 110 tons. At the present time all the ore is hand-sorted, and averages, according to smelter returns, about 12 per cent. zinc, 8 per cent. lead and 47 ozs. silver. As it comes from the mine, before sorting, the average, according to the manager, is about 7 per cent. zinc, 4 per cent. lead, and 20 to 35 ozs. silver. It is classed as a "dry" ore. The cost of hauling the ore by wagons to the landing is $4 per ton, by contract. The cost of tramming the ore from the mine is figured at 75c. per ton, and the cost of stoping at $2. All the ore is shipped to the Hall Mining and Smelting Co., Ltd., at Nelson, B.C. by way of steamer from the Landing.

By referring to Plate III, it will be observed that the first 270 feet of level No. 4 was driven on the Aspen vein as an adit, and exposed a shoot of ore for a distance of 110 feet from the portal. This lens has been stoped to the surface above the level, but can be seen here and there for about 175 feet in the floor. At this place it is narrow and does not represent a very large tonnage. The vein itself which is correspondingly narrow was left at a point 220 feet from the portal. Here a cross-cut at nearly right angles was started and driven into the foot-wall for a distance of 60 feet. At this point the Florence vein was intersected and followed at a course of N. 35° W. There is a strong and well defined ledge from 5 to 6 feet wide, with a seam of talc on the foot-wall, but nothing of particular interest until a distance of 380 feet is reached. At that point an ore shoot was struck and followed for 330 feet. The width of this pay streak in this shoot, known as the "big stope", averages 24 inches. It has been stoped for a distance of more than 100 feet above the level, and at the present time there is a very good showing of 2½ feet of ore in the upper stope. The general assay of the ore after hand-sorting is about as stated previously, according to the smelter returns. A long raise connects this level with No. 3 level and represents a vertical distance of 250 feet.

Following the Florence vein beyond the big stope for an additional 100 feet, it appears to be lean. Here a small fault occurs and a heavy flow of water was struck, the water pouring from a talc gouge on the hanging wall. About 25 feet further on a new ore shoot is met with and pay ore begins to make its appearance again. This can readily be followed in the tunnel, and considerable zinc ore can be seen in the roof; also in the present face. This ore-shoot is from 250 to 300 feet long. Its average width is about

$2\frac{1}{2}$ feet. In places the vein narrows down to 18 inches, while in others it widens out to 6 feet. Coming back 30 feet from the fore-breast a raise has been put up above the level to a height of about 100 feet, and has proved the existence of the ore very well all the way. The intention is to connect this raise (No. 14) with an intermediate level, which is now being driven toward the same at a height of 100 feet above the level. It will be observed from the accompanying map (Plate III) that this intermediate was started from raise No. 11, which is 375 feet back from the main face.

The intermediate level follows the same general direction as the main level, and for the first 150 feet is correspondingly lean, but a change took place when it reached a point directly above the place where ore had reappeared in the main level. From here onward the ore is continuous for 60 feet, or as far as the north face had been driven up to the date of my visit (Oct. 12, 1905). It is undoubtedly the same shoot, and the vein shows a very strong and distinctly banded structure for a width of five feet, dipping slightly to the southwest. A two-cut sample (No. 63) was taken here (top and bottom) on four parallel stringers representing a total width of 30 inches of ore. It assayed 19.6% zinc; 17.4% lead and 74.4 ozs. silver to the ton.

The intermediate level will have to be continued for about 120 feet before connection is effected with raise No. 14. This work is now being done as rapidly as possible. At the same time the main level is also being driven ahead.

From the intermediate level to the next level, No. 3, the vertical distance is 150 feet, but as this level has not yet been extended beyond a point corresponding with raise No. 11, any ground here will have to be regarded as unexplored, except as to that portion which is situated immediately above the big "stope", from where it is reasonable to expect a considerable tonnage of both shipping and concentrating ore. The total length of No. 4 adit is 1,400 feet, of which 800 feet has exposed ore in paying quantity.

Level No. 5, which is the lowest working on the property, was started at a distance of 225 feet, vertically, below level No. 4, and driven as a cross-cut tunnel for about 420 feet. The object in going so far away from the vein was to bring the portal outside of the danger limits of a snow-slide that scours the mountain each spring, and has left its marks distinctly in a draw following the side hill not far from the entrance. At the end of this cross-cut the Aspen vein was struck, and here the ore-shoot, which corresponds with the first one in No. 3 level, was intersected. This shoot, following the general strike of the vein for 200 feet is not much wider than the one corresponding with it above, and cannot therefore be considered of the same importance as the ore-bodies of larger dimensions further ahead. For a distance of thirty feet above the level the ore has been stoped. The levels at the end of this shoot have been turned off at nearly right angles for 50 feet, where the Florence vein was encountered. Evidently the prospect here did not come up to expectations, for immediately afterward the level was again turned off, and within a distance of 100 feet the Aspen vein was struck for the second time. A body of ore was disclosed and was followed for 195 feet. The pay streak is of the average size and grade. A sample (No. 62) was taken along the roof in several places where short stopes have been started for a distance of 20 feet or so above the level. The sample represented a width of 2½ feet. It assayed 18% zinc, 6%lead, and 34 ozs. silver to the ton. The general average of the ore body, as stated by the manager, is 7% zinc; 4% lead and 20 to 35% ozs. silver.

The stopes sampled and described in the foregoing appeared to be quite characteristic of the vein. Their content in zinc is higher and in composition they would be better suited for concentration than any other ore shoot exposed in the mine at the present time.

Following No. 5 level through the last mentioned ore-shoot, there is a short break or fault, from which onward the Florence vein appears to come in, indicating that the two veins may unite. Fifty feet further ahead a fourth ore shoot makes its appearance and can be measured for a distance of 90 feet along the level. This ore-body is perhaps one of the most valuable in the mine, although not from a zinc point of view. A very good grade of silver ore, fairly high in lead, but less so in zinc is being extracted for a width of two feet immediately above the level. From the end of this ore body to the face of No. 5 level there is about 100 feet of lean ground, but the forebreast looked favorable, and I have since my visit to the mine, been advised by one of the owners, Mr. Bruce White, that a new shoot of ore was uncovered after driving a short distance ahead. The only up-raise of any consequence in No. 5 level is the one which has been put up into the first ore-shoot on the Aspen vein to a height of 125 feet. As will be observed from plate III, an intermediate level has been started 100 feet above the main level by running two short drifts off in a northerly and southerly direction. A fair quantity of ore has been uncovered in doing this work, and it is expected to continue to the floor of No. 4 level, where it shows in places. The management therefore intends to carry the raise up higher and extend the intermediate level toward the north as indicated in dotted lines. By that time several other raises will be necessary both for ventilation and handling of ore from the large block of ground between the levels, which are 225 feet apart.

The property is being developed at the present time by a crew of about 20 men. All the workings of importance are being carried forward, and it will readily be seen from the longitudinal section of the mine that by continuing the lower level a large section of ground will be opened up; also that the proving in depth of such ore bodies as have been exposed ahead of No. 5 (in No. 4) are of great importance to the future of the property. The manager estimates that about 50,000 tons of concentrating ore are now blocked out, besides about 25,000 tons partly blocked out.

The company has now under construction at the lower terminal a concentrator of 75 tons daily capacity. The intention is to do away with the hand-sorting at the mine, which will be specially arranged for at the mill. The mill will be driven by water power, which is now being developed. The company has erected an efficient saw mill and all the lumber required for construction work is being cut from standing timber, of which there is no lack on the premises.

The problem of concentrating the Molly Gibson ore and effecting a high saving of the silver is by no means a small task, and will require a very careful study. However, the company experimented a good deal with the ore before it was decided to erect a concentrator. A small lot of the average grade of concentrating ore was obtained for testing at Denver, Colo.

This property is managed by Capt. T. H. TRETHEWEY.

BISMARK MINE.

This mine is situated on Briggs Creek, which is one of the tributaries emptying into the south Fork of the Kaslo River from the east. It is about twelve miles from Kaslo and seven miles from the Kaslo & Slocan railway station at South Fork. With the exception of the last mile or so of mountain trail there is a good wagon road leading to the property, following the fork.

Three adit tunnels have been driven on the vein, which is well defined, averaging about 2½ feet in width. The two upper levels, which are respectively at 6,700 and 6,588 feet elevations, are of interest only in reference to silver-lead ore. The ore from them is principally lead and zinc carbonates, with considerable iron oxide and quartz, thus furnishing a very desirable smelting product. The lead carbonate averages from 6 to 10 per cent., in lead with 135 ozs. silver per ton, while the zinc carbonate ore has run up as high as 15 per cent. in zinc, but is usually considerably lower. The production from the mine has never been large but the property has paid its way very nicely. During 1904, about 110 tons of ore were extracted with a crew of three or four men.

The general strike of the vein is 55 degrees to 65 degrees east of north, and it dips at an angle of 70 degrees toward the north. Level No. 1 is 290 feet long and No. 2 is 200 feet long, exclusive of 50 feet of cross-cutting into the hanging wall in each level. In the upper one there is an ore shoot of lead carbonate 30 feet long, near the portal, while the second one has two ore shoots, viz., 40 and 20 feet in a direction leading toward the upper tunnel. By keeping up the present development work there are fair prospects for continuing shipments at the same rate. The two above mentioned levels are connected by means of a raise.

The lowest level, which is really the only one of interest from a zinc point of view, is 440 feet long and 238 feet vertically below No. 2. As will be observed from Plate IV the vein was not struck until a distance of 135 feet from the portal had been reached. A cross-cut of about 20 feet was made here into the foot wall. The ledge at this point shows plainly for a width of 36 inches. A narrow but well defined and high grade streak of blende and galena can be seen in the face of the drift. Apparently this ore is just coming in as a new shoot different in character from anything found in the upper levels of the mine. It is a heavy mixture of sulphides, assaying, according to sample No. 85, taken at its full width of 6 inches in two cuts (top and bottom) 38.4% zinc, 26.8% lead and 196.3 ozs. silver to the ton. Of this class of ore there is at the present moment practically no tonnage developed in the mine, but work on the discovery is being kept up, and the company is justified in continuing exploration work by drifting as well as raising on it. No connection between this level and the upper one exists.

By referring to Plate IV it will be observed that beyond the cross-cut the level was continued along a slip in the formation, and that the ledge referred to in the foregoing was not exposed again until it reached a distance of 245 feet from the first place. At this point another short cross-cut was made intersecting the vein and showing it from 2 to 3 feet wide. From here onward the ledge was followed to the present face, for a distance of 60 feet, without any further difficulties. The ledge here shows that the same composition and bunches of galena and blende are met with, although not to the same extent nor as regular in occurrence as in the first place. At the forebreast I measured the ledge for a width of 24 inches, leading off at a course of N. 55° degrees E. It is of interest to notice the proximity of a large dike, which can be seen at the face of a third cross-cut, run into the foot wall, 35 feet back of the face. This dike, I was informed, can be traced plainly on the surface.

BLACK FOX MINE.

This property is owned by the Black Fox Mining Company. It is situated below the Bismark, slightly to the northeast of the latter, and adjoins the Cork mine to the south.

The principal vein of the Cork and Province mines, which has been extensively developed and is the most important in the South Fork district, is generally supposed to traverse the Black Fox claim, and also the Daisy, belonging to the same company. From a brief examination of the workings, which consists of a cross-cut tunnel 150 feet long, I obtained the impression that while there has been a vein cut in the Black Fox tunnel, 100 feet from its portal, it is not the principal vein of the Cork, but a parallel one of secondary importance, which can be seen in the Cork at 700 feet from the portal of the main tunnel. There are several parallel veins in this district, and the identification of a particular one, in the absence of absolute connection, is chiefly a matter of surmise and is always doubtful.

In the vein exposed in the Black Fox tunnel only a few inches of zinc blende can be seen. The vein is about 4 feet wide and crosses the tunnel. A sample (No. 86) consisting of a few selected pieces of blende taken from here gave the following assay: 36.6% zinc, 2.3 ozs. silver. The quantity of ore exposed is so small that the sample cannot be regarded as of much importance. The surface showing on the property, (approximately 100 feet further up the hill), is on the other hand quite strong and can be followed for over 200 feet, disclosing a width of several feet. The vein dips at an angle of about 50 degrees to the southeast. Several pits have been dug, exposing the vein and the same character of ore as in the Cork and Province mines, that is to say, a mixture of blende, galena and spathic iron.

In order to intersect this surface vein the tunnel would, in my opinion, have to be driven for some distance yet, as the vein is dipping away from the cross-cut into the hill.

B. & A. MINE.

This property is situated approximately three miles south of the Bismark, at an elevation of about 5,000 feet. It is reached from the Kaslo & Slocan railway station by the excellent South Fork wagon road for the first 8½ miles; thence by following the mountain trail for an additional mile and a half along the tributary to the South Fork, known as Lake Creek.

When this property was operated, several years ago, it was entirely for silver-lead ore. The lower tunnel being the only one of interest to the Zinc Commission, was alone examined. The development is shown in Plate V. The vein has a strike of N. 45° E., and a dip of 85° to the southwest. It is of considerable width and appears to be a crushed zone in which lenses of both zinc blende and galena are found. The first 35 feet of the tunnel, which has a total length of 158 feet, was driven through "wash" until the true hanging wall was found in place. At this point the tunnel was turned off slightly to the east, and within 18 feet the true foot wall was encountered, and followed to the present face, representing a distance of 110 feet. The foot wall, which is perpendicular, is remarkably well defined as far as it has been developed. The hanging wall, which can only be seen in places, appears to be equally well defined. It has a slight dip to the southwest. The vein matter is soft and considerably crushed. It contains a good deal of slate and calcite. Two short cross-cuts have been run across the vein. The first one which cuts through to the hanging wall at a point 98 feet from portal, shows the vein to be 16 feet wide. From three to four inches of zinc blende can be seen here close to the wall. In the next cross-cut, 55 feet further ahead, the vein is 20 feet wide, and apparently the same streak of blende continues through and can be seen 6 feet from the hanging wall. The pay-streak here has widened out to 15 inches, and a sample (No. 90) which was taken in two cuts (top and bottom) gave the following result: 45% zinc and 16.2 ozs. silver to the ton.

I think there is but little doubt that the zinc showing in the two cross-cuts are of the same shoot of ore. At the present time there is not sufficient ore developed to warrant any estimate of tonnage. So far as can be seen, the exposure of zinc in the second cross-cut is of more importance than in the first. By driving further ahead, and by keeping closer to the hanging wall than has so far been done, the present showing of ore might possibly improve in width and importance. The vein is well defined and wide; the vein matter is soft and readily mined. Timber and water is plentiful in the immediate vicinity.

BLACK PRINCE MINE.

This is a prospect, located near the wagon road on the hill side, a mile or so from the Mollie Gibson landing. The vein, which is from 3 to 5 feet wide, is in the Nelson granite, but is not very clearly defined. Its strike is N. 20 W. Development work consists of an open cut, exposing the vein for a vertical distance of 10 or 12 feet. In the vein a few stringers of zinc-blende can be seen, but an average sample across the face would not have given any satisfactory results. I selected a few pieces of the blende found in the excavated material on the dump in order to obtain an idea of its purity. The result of this sample (No. 64) was 50.4% zinc and ½ oz. silver to the ton.

Apparently on the same vein, but 200 feet below the wagon road, I noticed that some work had been done on the extension of the Black Prince claim. Sample No. 65 taken in a similar way gave 16.8% zinc, 5.5% lead and 2.7 ozs. silver to the ton.

EMILY EDITH MINE.

The Emily Edith mine is situated from 800 to 1,100 feet above Slocan Lake, and is three or four miles north of Silverton. It is reached over a good wagon road. The country rock in the immediate vicinity consists of shales and slates, and the Emily Edith lode impressed me as being a considerably crushed and widely mineralized ore zone in that formation. It shows much irregularity in its width, course and dip. Its general strike is about N. 65 degrees E. It ranges in width from a few feet up to 30 feet. Its dip in the upper levels is about 45 degrees, while in the lower ones it is from 60 degrees to 75 degrees to the southwest. There are altogether six adit tunnels on the property from which many drifts and cross-cuts extend in the direction of either walls. The total lineal feet of work is probably at least 8,000. I visited four of the tunnels and located the principal zinc ore shoot of the mine in each of them, these representing a vertical depth of 173 feet, or about 250 feet measured on the dip of the vein. The ore body is no doubt continuous for this distance, and of importance as a concentrating ore. Recently a mill test was made on a hundred ton lot at the Wakefield concentrator, and a representative sample of the zinc concentrate, produced from the ore, was secured for experimental purposes.

In the upper tunnel No. 02, the ore body referred to above shows for a distance of 60 feet, averaging about 2 feet in width. In the next tunnel No. 01, which is 57 feet below No. 02, the ore can be seen to better advantage and measures in one place 6 feet in width. Sample No. 74 was taken here across the face in a short upraise, and in three other cuts, representing an average width of four feet for 30 feet along the level. It assayed 34.2% zinc, 7.7% lead and 7.2 ozs. silver to the ton. This was apparently the best grade of ore showing in the mine. Beyond the 30 feet sampled the same character of ore can be seen and followed for 100 feet or more, but the pay ore for this distance would hardly average over 2 feet in width. The vein itself is of considerable width, and the pay ore appears to lie on both walls, which are from 10 to 30 feet apart. This feature holds good throughout the mine, and during development work in many cases there has been a drift run on the foot as well as on the hanging wall. Moreover, many intermediate cross-cuts have been made in the ore zone in all of the tunnels.

In tunnel No. 1, which is 53 feet vertically lower than No. 01, the same ore body can be observed immediately below its exposure in No. 01, and a sample (No. 75) in a 30 foot raise along the foot wall in three cuts (top and bottom) was taken for an average width of 3 feet. This assayed 29.2% zinc, 12.4% lead, and 13.2 ozs. silver to the ton. The ore shoot has been exposed intermittently along the supposed foot wall for 250 feet; also in a drift back of the supposed foot wall for about 75 feet. The pay-ore in both drifts is from 2 to 3 feet wide, and fairly continuous. At one place along the level, about 150 feet from where the last mentioned sample was taken, the ore body widens out to 6½ feet. A raise has been put up here, proving its extension for a certain distance. Complete examination could not be made here with safety. The blende is of fine grain, and lies on the foot wall. A sample (No. 76) was taken here for a width of 6½ feet in three cuts, 10 feet apart. It assayed 26.2% zinc, 8.7% lead and 10.4 ozs. silver to the ton.

The next level, No. 2, is 63 feet vertically below No. 1, and exposes the ore shoot for about 75 feet along the level; also in a drift 10 feet above the level, where a sample (No. 77) was taken in three cuts covering a distance of 25 feet, representing a width of 3 feet. It assayed 24.6% zinc, 10% lead, and 10.3 ozs. silver to the ton. In a cross-cut about 40 feet from where this sample was taken the ore body shows a distinct width of 10 feet. This showing has however, not been followed up, evidently because the original company was not developing the property for zinc. The galena associated with the ore could evidently not be mined and sorted at a profit and would require concentration. Conditions at the present time are somewhat different, and the property has been for the last year or so under lease to Mr. M. S. Davys of Nelson, who is planning to work it for its combined zinc and silver-lead values.

While there is no tonnage of ore actually blocked out in this property there is undoubtedly a considerable quantity of concentrating ore to be relied on. I should estimate this roughly at 10,000 tons of ore, carrying

22% zinc, 6% lead, and 6 ozs. silver to the ton. Systematic development work is likely to increase this tonnage materially.

In tunnels No. 3 and 4 there has been considerable drifting and cross-cutting done, but so far it has not proved successful in locating the above mentioned ore body or any others. Tunnel No. 3 is entirely off the course and must have been driven for other purposes than the one of showing up existing ore bodies. All work on the property, including cross-cutting and drifting is distributed in the levels about as follows:

No. of Level.	Elevation above sea-level.	Approx. Amount of Development Work.
02	2879 feet	700 to 800 feet.
01	2822 feet.	1200 to 1300 feet.
1	2769 feet.	1500 to 1600 feet.
2	2706 feet.	1800 to 1900 feet.
3	Not observed.	400 to 500 feet.
4	2561 feet.	1500 to 1600 feet.

The elevation of Slocan Lake above sea-level is 1,761 feet.

The Emily Edith property is equipped with an aerial tramway for handling shipping ore; also a number of good buildings, such as office, bunk-houses, stables, ore sheds, etc., all of which are located below the main workings and go to make up a complete and well arranged camp. So far the shipments from the property have not been extensive. The group consists of six Crown-granted claims and comprises 150 acres. It is owned by the Emily Edith Mines, Limited, of London, England.

GALENA FARM MINE.

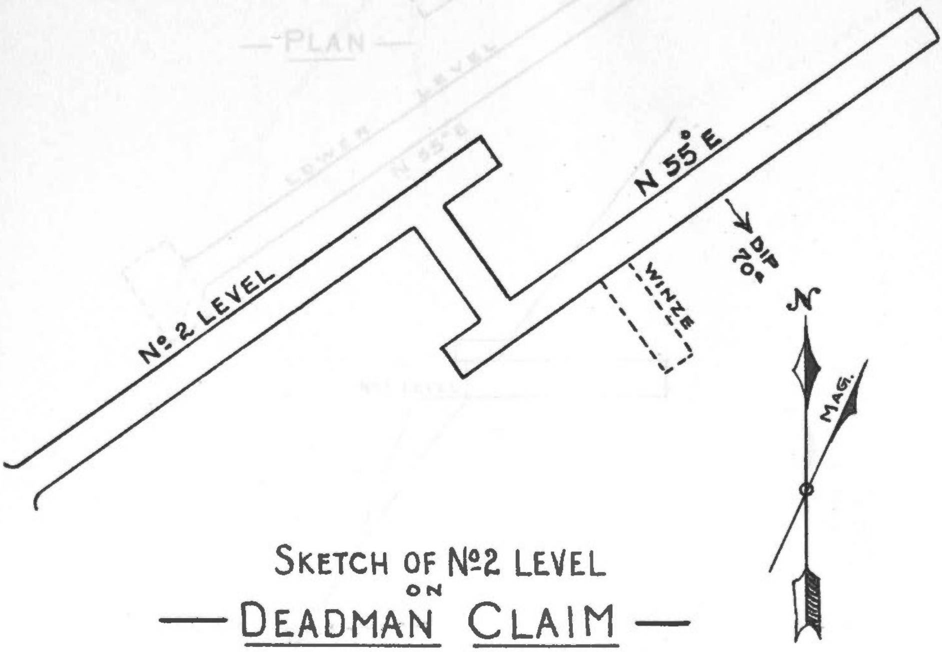
This property, formerly known as the Currie group, comprises several Crown-granted claims in the Silverton district, which are owned by an English Company. It contains 193 acres of mineral land, situated about 1,000 feet above Slocan Lake, 1¾ miles southeast of Silverton, with which it is connected by a good wagon road. The vein has a strike of about S. 60 degrees E. and a dip of 50 degrees N. It is in granite formation and out-crops strongly on the surface for several hundred feet. It can be seen in places with a width of 9 to 10 feet. It shows quartz, spathic iron, galena and blende mixed with fragments of granite and slate.

All workings except two or three surface pits are at the present time under water, which prevented examination at depth. A sample (No. 78) was taken 15 feet under ground in the original prospect shaft. This assayed 38.4% zinc, 13.4% lead, and 32 ozs. silver. Another sample (No. 79) taken in two or three places on the surface croppings assayed 42% zinc, 6.8% lead, and 14.6 ozs. silver to the ton. These results should not be considered an average of the ore, but will convey an idea of its composition.

About 175 feet west of the shaft mentioned above, a perpendicular two-compartment working shaft has been here sunk to a depth of 220 feet, and levels 100 feet apart have been run off from here for the purpose of intersecting the vein. Considerable work has been done under ground. I had no means of ascertaining the result of this work.

The machinery plant is well arranged and has, during the long shut down, been taken good care of. It consists of a first-class geared hoist driven by a 4-foot Pelton water wheel, one standard cage and hoisting equipment, and one 4 drill air compressor driven by a 6-foot Pelton water wheel. Besides the above machinery there is a steam hoist and boiler plant, which was used before water power was installed, still in position; also several pumps apparently in good order. The water power is obtained from Gold Creek, where a dam has been constructed. A 3,500-foot pipe line conveys 100 miners' inches of water to the water wheels under a head sufficient to furnish a pressure of 110 pounds.

PLATE I



SKETCH OF No. 2 LEVEL
ON
— DEADMAN CLAIM —
— NOBLE FIVE GROUP —
SANDON, — SLOCAN MG. DIV. — B.C. —
SEPT 18TH 1905

SCALE
10 20 30 40 50 60 70 80 90 100 FT.

ALF. C. GARDÉ M.E.

levels 100 feet apart have been run off from here for the purpose of intersecting [illegible]. Considerable work has been done under ground. I had no means of ascertaining the result of this work.

The machinery plant is well arranged and has, during the long shut down, been taken good care of. It consists of a first-class geared hoist driven by a 4-foot Pelton water wheel, one standard cage and hoisting equipment, and one [illegible] air compressor driven by a [illegible] Pelton water wheel. Besides the above machinery there is a steam hoist and boiler plant, which was used before water power was installed, still in position; also several pumps apparently in good order. The water power is obtained from Gold Creek, where a dam has been constructed. A 3,500-foot pipe line conveys the [illegible] inches of water to the water wheels under a head sufficient to furnish a pressure of 130 pounds.

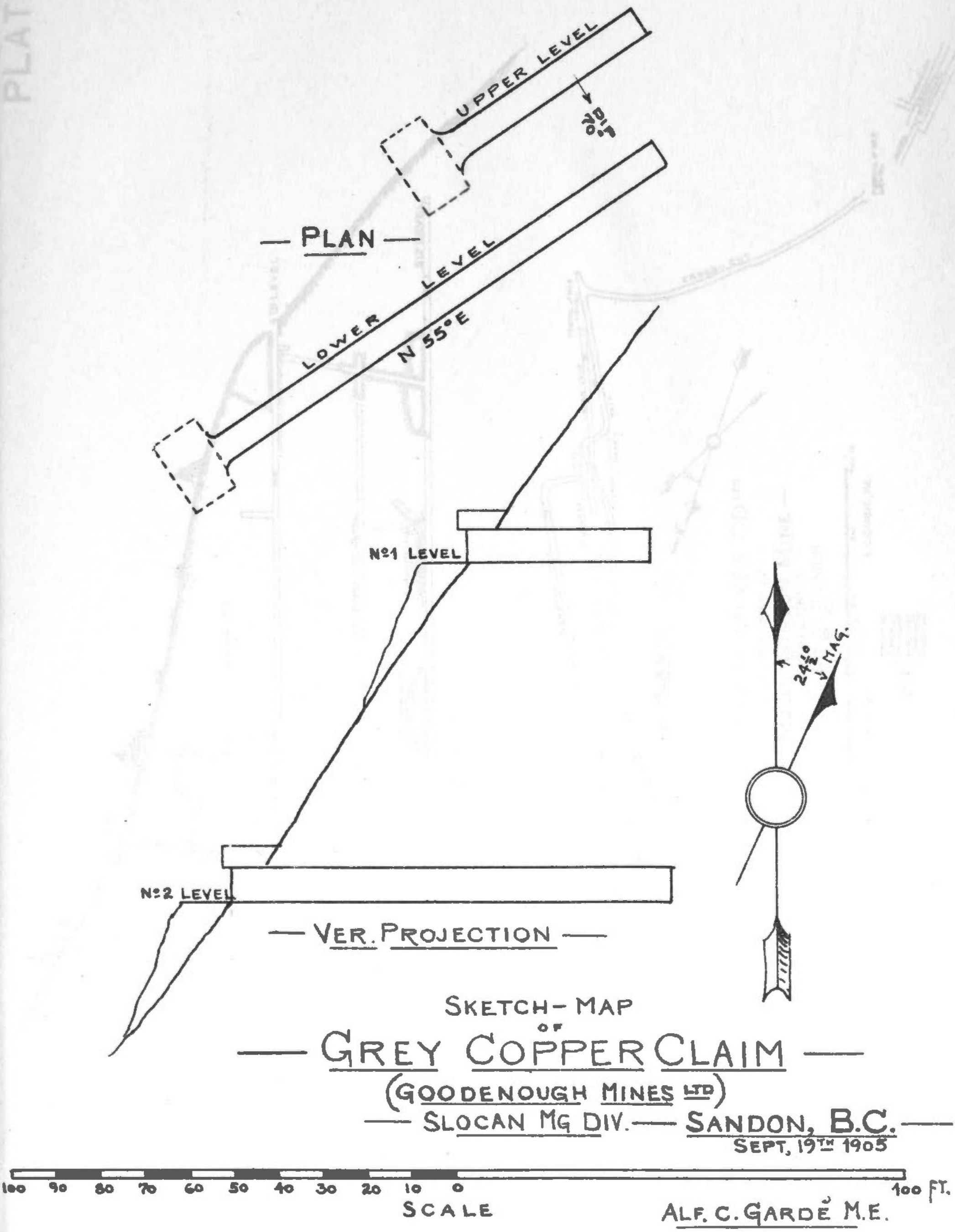

UPPER LEVEL
DIP 70°
— PLAN —
LOWER LEVEL
N 55° E
No. 1 LEVEL
No. 2 LEVEL
— VER. PROJECTION —
24½° MAG.
SKETCH-MAP
OF
— GREY COPPER CLAIM —
(GOODENOUGH MINES LTD)
— SLOCAN MG DIV. — SANDON, B.C. —
SEPT. 19TH 1905
100 90 80 70 60 50 40 30 20 10 0 100 FT.
SCALE
ALF. C. GARDE M.E.

PLATE II

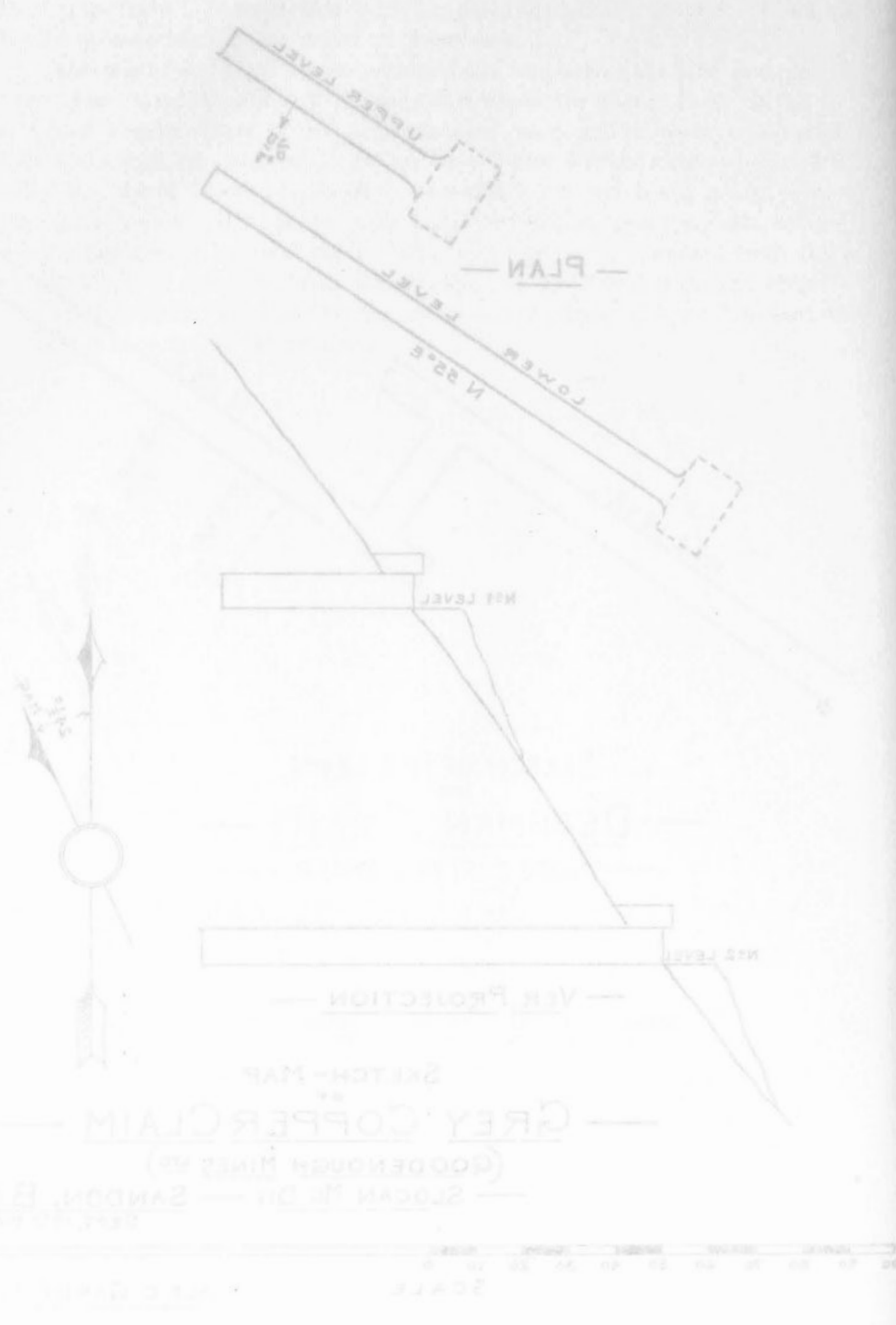
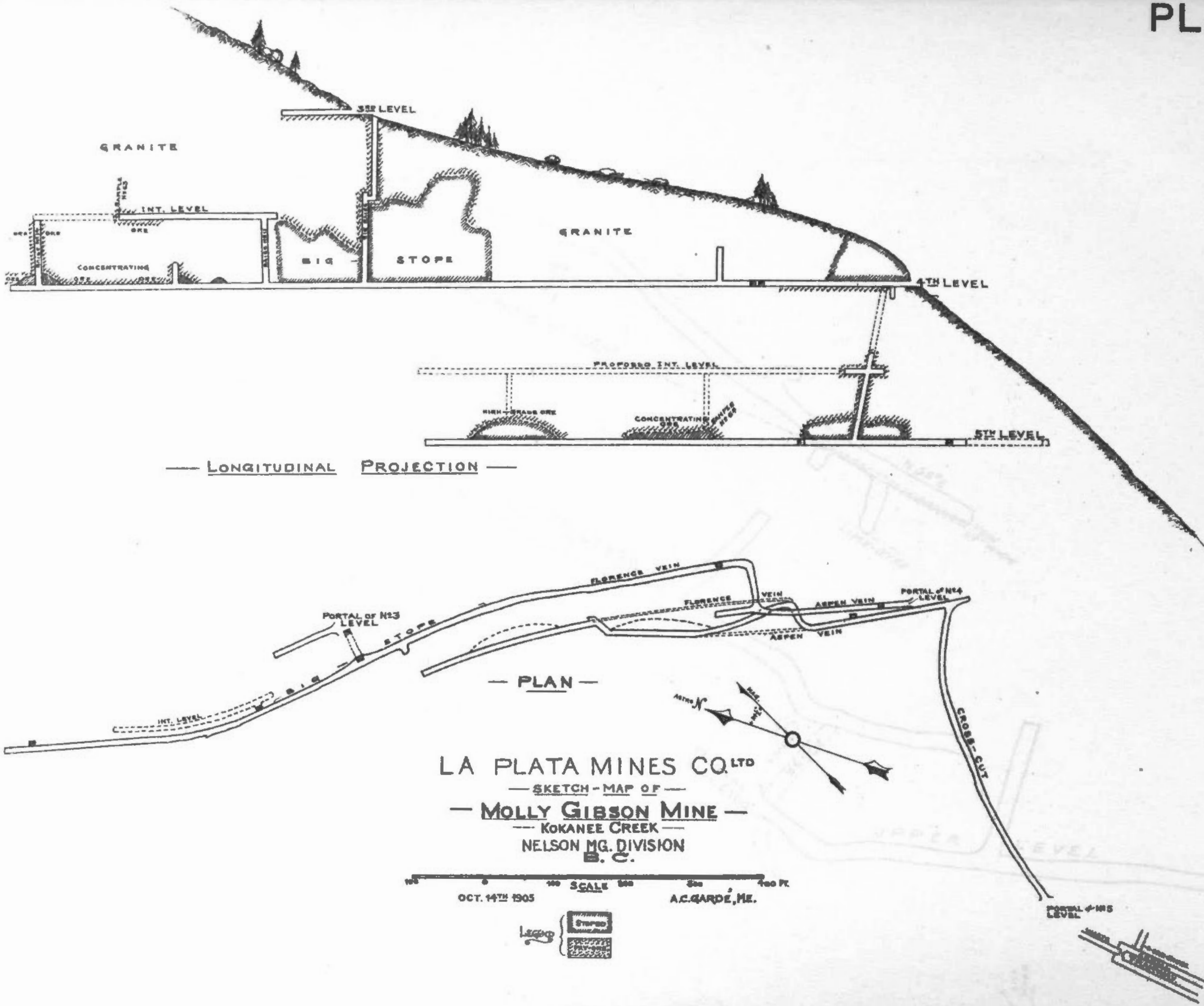
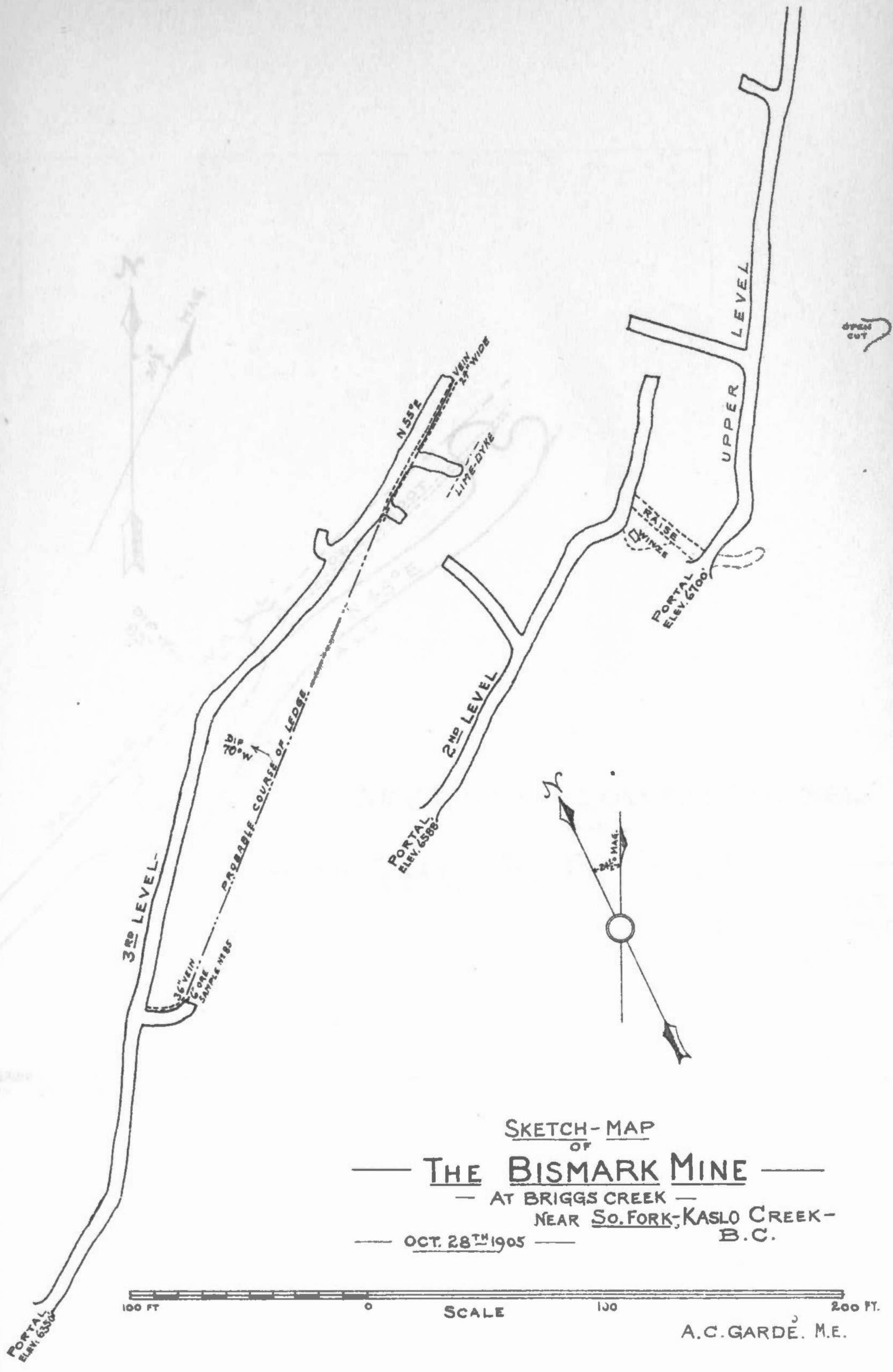

UPPER LEVEL
DIP
LOWER LEVEL
N 55° E
— PLAN —
No 1 LEVEL
No 2 LEVEL
— VER. PROJECTION —
SKETCH-MAP
OF
— GREY COPPER CLAIM —
(GOODENOUGH MINES LTD)
— SLOCAN M. DIV. — SANDON, B.C.
SCALE

PLATE III



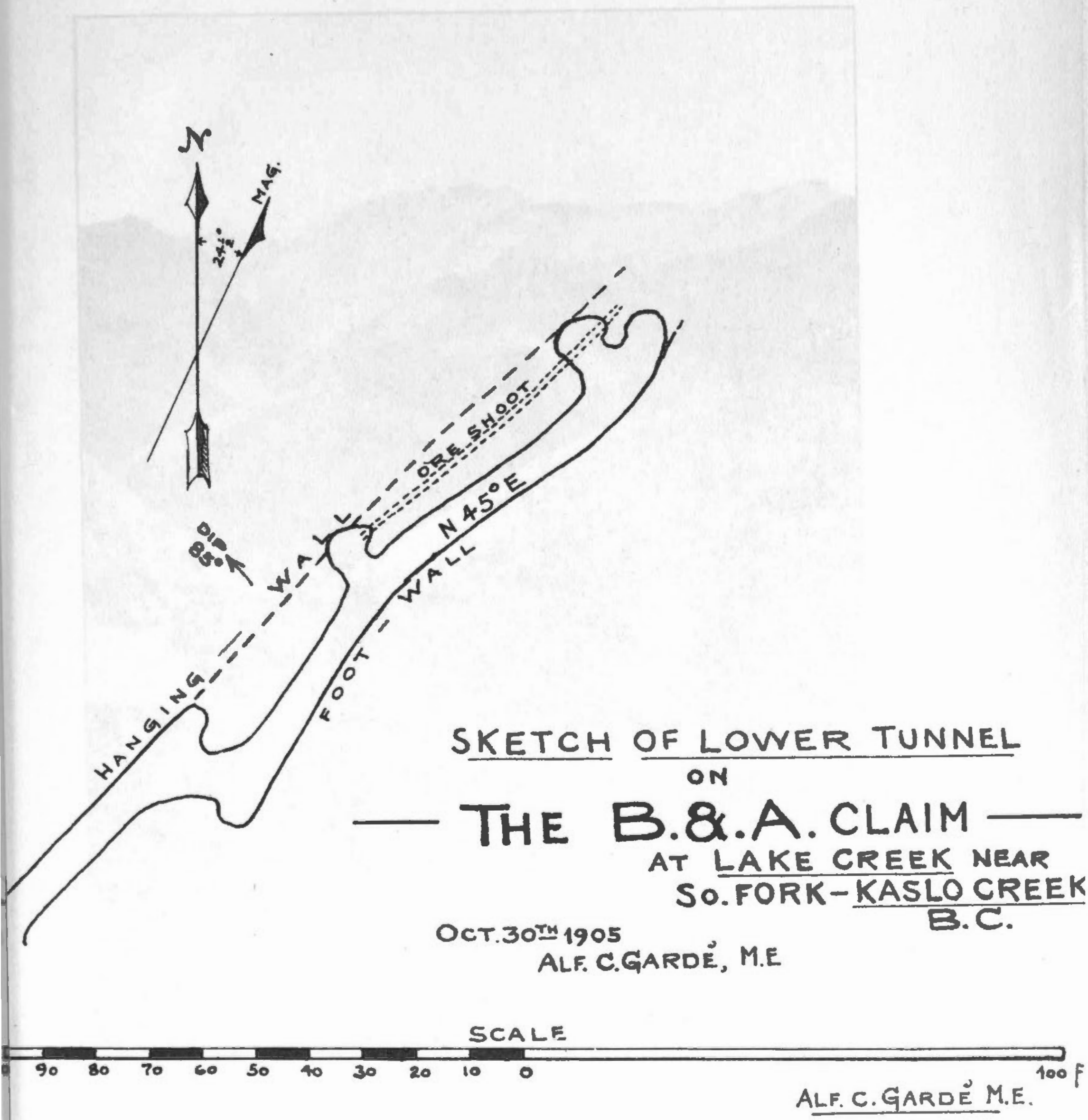
3RD LEVEL
GRANITE
INT. LEVEL
CONCENTRATING ORE
BIG STOPE
GRANITE
4TH LEVEL
PROPOSED INT. LEVEL
HIGH GRADE ORE
CONCENTRATING ORE
5TH LEVEL
LONGITUDINAL PROJECTION
FLORENCE VEIN
PORTAL OF No 3 LEVEL
BIG STOPE
FLORENCE VEIN
ASPEN VEIN
PORTAL OF No 4 LEVEL
ASPEN VEIN
INT. LEVEL
PLAN
CROSS-CUT
PORTAL OF No 5 LEVEL
LA PLATA MINES CO. LTD
SKETCH-MAP OF
MOLLY GIBSON MINE
KOKANEE CREEK
NELSON MG. DIVISION
B.C.
SCALE
OCT. 14TH 1905
A.C. GARDÉ, ME.
LEGEND
STOPED

PLATE IV

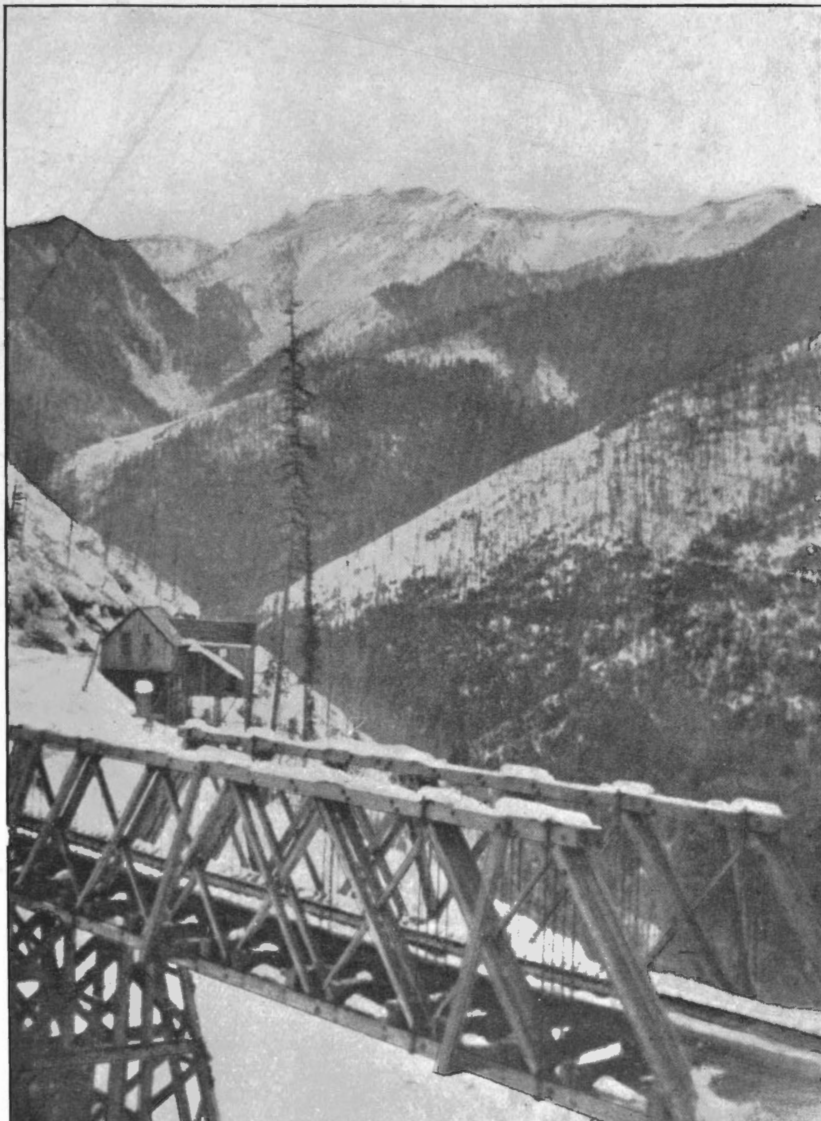


UPPER LEVEL
OPEN CUT
RAISE
WINZE
PORTAL ELEV. 6700'
2ND LEVEL
PORTAL ELEV. 6588'
N 55° E
VEIN 2½' WIDE
LIME-DYKE
DIP 70° W.
PROBABLE COURSE OF LEDGE
3RD LEVEL
36" VEIN 6" ORE SAMPLE No 85
PORTAL ELEV. 6350'
N
MAG.
SKETCH-MAP OF
THE BISMARK MINE
AT BRIGGS CREEK
NEAR SO. FORK, KASLO CREEK
B.C.
OCT. 28TH 1905
100 FT
0
SCALE
100
200 FT.
A.C. GARDÉ. M.E.

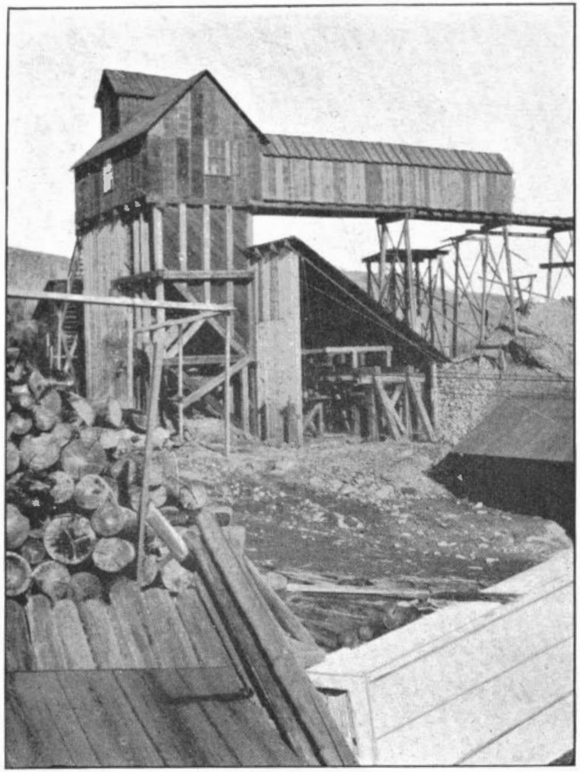
PLATE V



N
MAG.
24½°
DIP 85°
HANGING WALL
ORE SHOOT
N 45° E
FOOT WALL
SKETCH OF LOWER TUNNEL
ON
THE B.&A. CLAIM
AT LAKE CREEK NEAR
So. FORK-KASLO CREEK
B.C.
OCT. 30TH 1905
ALF. C. GARDÉ, M.E
SCALE
90 80 70 60 50 40 30 20 10 0
100 F
ALF. C. GARDÉ M.E.



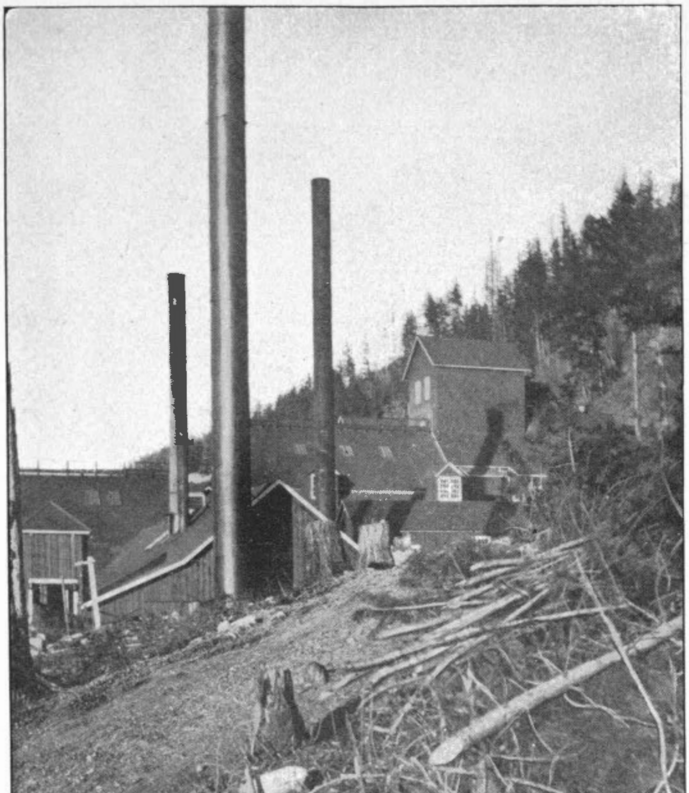
View from Payne Residence Looking South.



Sullivan Mine, No. 5 Shaft House.

Head of Aerial Tramway at Sullivan Mine, Kimberley, B.C.

Slocum Star Mine, Sandon.
Byron N. White Co.



SULLIVAN MINE.—Boiler House.

WHITEWATER MILL.—Whitewater, Ainsworth, B.C.

View of the Payne Lower Tramway Terminal.

ENTERPRISE MINE– SLOCAN LAKE.

REPORT

ON

SOME MINES OF AINSWORTH AND THE SLOCAN.

BY

ALFRED C. GARDÉ, M.E.,
Member of the Canadian Mining Institute.


SOUTH FORK OF KASLO CREEK
Embracing District from Whitewater to Kaslo
SCALE, 3,000 FEET TO ONE INCH.
Compiled by A. R. HEYLAND
PROVINCIAL LAND SURVEYOR
KASLO, B.C.
NORTH FORK OF KASLO CREEK
KASLO AND SLOCAN RAILROAD
WAGON ROAD
KASLO CREEK
KASLO WAGON ROAD TO KASLO
KASLO AND SLOCAN RY
KOOTENAY LAKE
CITY OF KASLO
FORK OF KASLO CREEK
NASHVILLE
CEDAR CREEK
MONTEZUMA CREEK
SOUTH FORK OF KASLO CREEK
CROWN GRANTED MINERAL CLAIMS
SIRTEEN TO ONE
STARS & STRIPES
LIBERTY
MASTODON
MANHATTAN
PROVINCE
CORK
DUBLIN
ROSE
BOSTON
CALIFORNIA
DAISY
BLACK FOX
PATRICKS
GOLD CURE
" " FRACTION
FULL RIG
CROWN POINT
B.N.A
SILVER BELL
HUB
GRANITE KING
GREY EAGLE
KIMBERLEY
MONTEZUMA
MEXICO
SCALE 1500 FEET TO ONE INCH
COMPILED BY
A.R. HEYLAND
PROVINCIAL LAND SURVEYOR
KASLO, B.C.

Index of Mineral Claims shown on the Map of the Slocan Mining Camp prepared by Messrs. Drewry & Twigg, New Denver, B.C., April, 1906.

The district embraced in the map has been divided into Localities. In defining these the natural boundaries and means of access have been considered as far as possible.

In the index under the heading of Locality the following abbreviations have been used.

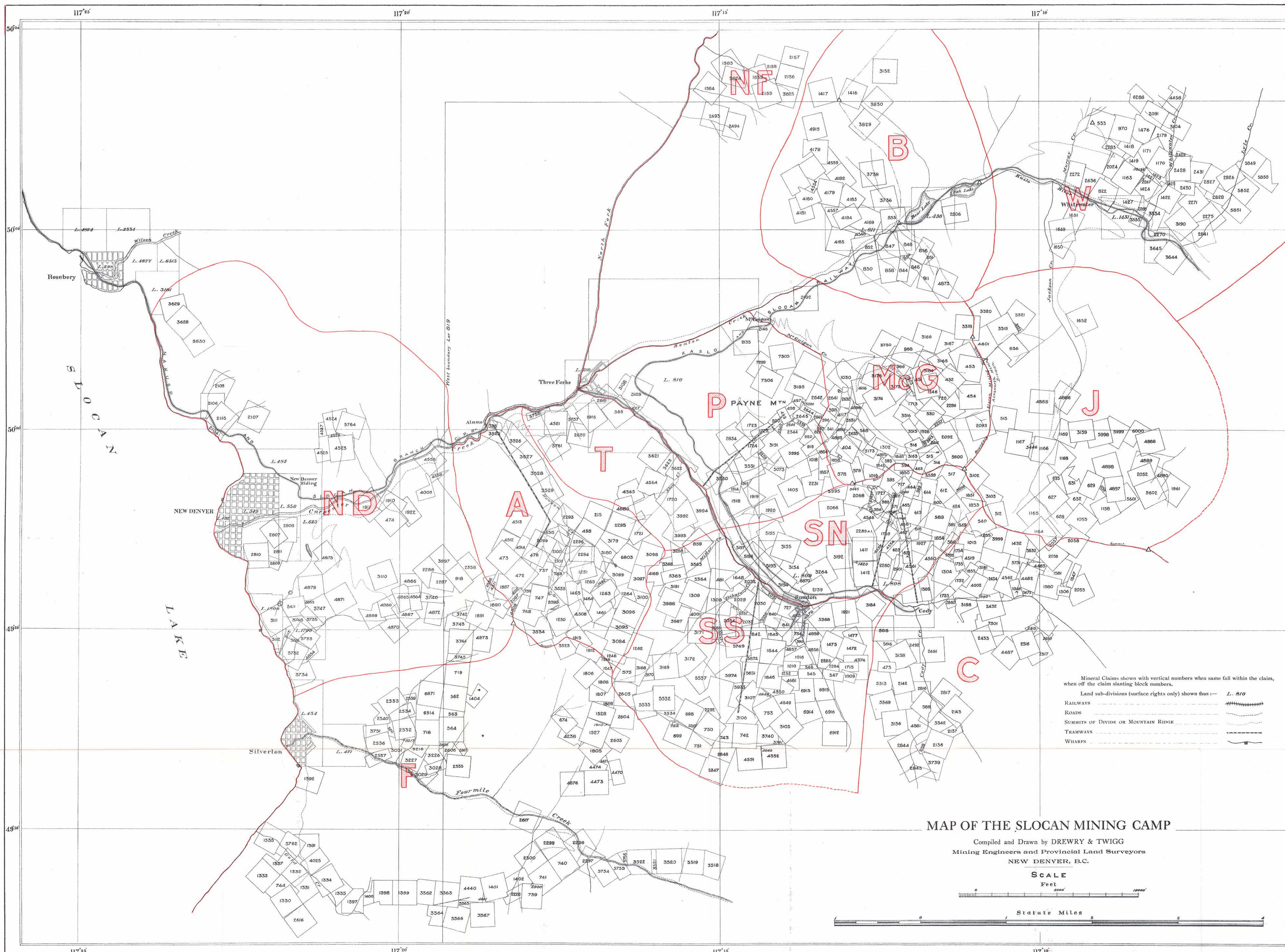
Alamo and basin of Howson Creek	A.
Bear Lake	B.
Cody	C.
Four-Mile Creek and Silverton	F.
Jackson Cr. basin and East	J.
McGuigan Cr. basin	McG.
New Denver	N.D.
North Fork Carpenter Cr.	N.F.
Payne Mountain	P.
Sandon, North of Carpenter Cr.	S.N.
Sandon, South of Carpenter Cr.	S.S.
Three Forks, mountain South of	T.
Whitewater	W.

INDEX

Name of Claim.	Locality.	Official Lot and Reference No.
Acme	W.	5852
Adams	S.S.	750
Adela	F.	1535
Adirondack	A.	4511
Admiral Nelson	S.S.	3106
Ajax	S.N.	583
Ajax Fr.	S.N.	1727
Alamo	A.	737
Alma No. 2	C.	2433
Alice No. 2	J.	4897
Alice Fr.	McG.	1928
Alligash	C.	2517
Almeda	J.	628
Alpha	F.	562
Alpha No. 2	W.	5819
Alpha No. 3	N.D.	3733
Alta	B.	853
Altoona	P.	1918
Amazon	A.	4513
American	W.	5851
American Boy	S.N.	571
American Girl	A.	2295
Ancestor Fr.	A.	6514
Anchor	B.	4169
Antelope	McG.	452
Antoine	McG.	516
Apex	N.D.	1911
Arena Fr.	F.	2539
Argenta	S.N.	1412
Aspen	J.	1158
August Flower	N.D	4865
Aurora	S.S.	2030
Aurora Fr	S.S.	2036
A.Y.	W.	2272
Baby Fr.	C.	5542
Badger State	S.S.	2033
Baldwin	A.	4512
Baltimore	F.	5755
Baltimore Fr.	F.	3522
Beaver	F.	1807
Belladonna	S.S.	6913
Bell	C.	2844
Bell	J.	1165
Belle Smith	N.F.	3824
Belt	S.N.	2139
Benbow	S.S.	4551
Bendigo	S.S.	3740
Bendigo Fr.	S.S.	3741
Bessie	B.	4183
Best	McG.	451
Best Fr.	McG.	3164
Betsy Ross	N.D.	3743
Big Kanawha	S.S.	4550
Big Timber	P.	3191
Black Bird	B.	4180
Black Coal	A.	1721
Black Hawk Fr.	S.N.	4546
Black Knat Fr.	S.N.	2490
Blade	C.	4519
Blizzard	S.N.	1849
Blue Bird	C.	540
Blue Grouse	S.S.	1846
Blue Jay	S.S.	482
Blue Jay	S.N.	594
Bondholder	J.	1061
Boatswain Fr	N.D.	3112
Bolander	C.	2143
Bonanza King	S.N.	465
Bon Ton	C.	636
Bosphorous	McG.	3167
Bow Knot	P.	1919
Brandon	S.S.	751
Breslau	W.	2430
Britomarte	S.S.	698
Broken Lock	N.D.	3735
Bryan No. 4	C.	1581
Bryan Fr.	W.	3751
Buck Fr.	F.	674
Buffalo	F.	[illegible]
Burnside	F.	2617
California	N.D.	918
Cape Fr.	J.	2101
Capella	N.D.	4527
Carbonate King	P.	919
Cariboo	J.	631
Cariboo	McG.	730
Carrie	A.	2840
Carrie	S.S.	575
Carnation	C.	5751
Cashier	F.	1528
Cazabazua	F.	1809
Cazabazua Fr.	F.	3528
Cedar	A.	3226
Centaur	F.	2415
Central	McG.	4557
Century Fr.	C.	1752
Chambers	C.	699
Chamblet	W.	2091
Charleston	S.S.	2631
Charlotte	McG.	4809
Chatham	N.F.	2493
Chetopa	A.	3534
Chicago	S.N.	622
Chicago No. 2	C.	2142
Chief	N.D.	2808
Christine	C.	5369
Cinderella	J.	3621
C. & K.	S.S.	5974
Clara Moor	F.	5762
Clarence	A.	3998
Cliff	F.	2606
Cliff Fr.	F.	2608
Cliff Extension	F.	2607
Clifton	S.N.	517
Climax	C.	1846 A.
Clipper	N.D.	3997
Cody Fr.	J.	4601
Colonial	C.	5313
Colorado	W.	1476
Columbus	S.S.	1309
Comet	A.	1462
Comiskey	B.	1351
Concord	A.	2293
Condor	F.	3518
Condore	C.	5615
Conductor	J.	1251
Congress	J.	513
Consolidated Virginia	T.	3992
Constant	C.	2817
Continental	A.	2097
Convention Fr.	N.D.	2288
Corean	W.	6788
Corinth	A.	1264
Cornerracker Fr.	N.D.	4854
Corless Fr.	C.	5616
Cow Boy	S.S.	4888
C. P. R.	N.D.	4875
Cracker Jack	N.D.	3747
Crawford Fr.	S.N.	5197
Crescent	F.	2540
Cross Roads	McG.	4116
Crow Fr.	F.	3365
Crown Point	S.N.	586
Cuba	S.S.	4168
Cumberland	A.	478
Curley	A.	1335
Currant	A.	3529
Currie	F.	744
Daisy	A.	2099
Dalkeith	F.	2603
Daniel	S.N.	3193
Dardanelles	McG.	453
Daybreak	A.	1464
Day Dawn	McG.	596
Day Dawn Fr.	McG.	598
Daylight	C.	3181
Dayton No. 2	J.	1914
Deadman	S.N.	613
Deardayer	C.	2491
Defender	A.	1453
Defender	W.	2827
Delorane	C.	3156
Democrat	A.	1250

INDEX—*Cont'd.*

Name of Claim.	Locality.	Official Lot and Reference No.
Denver	N.D.	2807
Derby	C.	1855
Despair	S.S.	840
Detroit	A.	3525
Dewey	N.D.	4525
Diamond Fr.	S.S.	5832
Dixie Hummer	F.	1912
Dominion	McG.	2641
Donnelly	S.N.	5195
Dorothy	S.S.	2034
Dragon	B.	848
Dublin Queen	J.	1167
Duluth	McG.	1019
Dum Dum	F.	5073
Dump Fr.	McG.	2257
Dunedin	S.N.	1853
Eagle	F.	3031
Eagle Fr.	F.	3011
Eastern	W.	1427
Eastern	A.	2100
Early Bird	S.S.	1648
Echo	J.	2057
Echo	F.	719
Echo Fr.	S.S.	4858
Edith	N.D.	4867
Edith Fr.	N.D.	4870
Edinburgh	F.	4473
Egalite	J.	3103
Egypt	N.D.	1922
Elazahar Fr.	B.	4548
Eldorado	S.N.	2901
Electric	B.	3738
Elgin	S.S.	742
Elk	T.	3993
Elkhorn	S.S.	859
Elkhorn	W.	2428
Elvira	S.N.	3155
Emily Edith	F.	2532
Emily Edith Fr.	F.	3217
Emma	S.S.	1009
Empire	S.S.	1477
Empire No. 3	C.	1580
Ephriam Fr.	S.N.	600
Erie	A.	3995
Essex Fr.	F.	4474
Estella	N.D.	3742
Eureka	S.N.	1753
Eureka No. 2	S.S.	2284
Eva Fr.	C.	6920
Evening	S.S.	3169
Evening Star	N.F.	1584
Exeter	T.	2645
Fair View	J.	5998
Fairy	A.	4508
Faustina	J.	3102
Felix No. 3	P.	2643
Fidelity	S.N.	2411
Fidelity Fr.	N.D.	3748
First Extension	C.	2294
Flood Fr.	McG.	5730
Florence	N.D.	5764
Flower	P.	7305
Fourth of July	C.	2138
Fourth of July	P.	2052
Frank Fr.	S.N.	4547
Franklin	C.	588
Franklin	W.	3645
Freddy	F.	4025
Freddy Lee	C.	475
Fremo	W.	1423
Friday Fr.	T.	5757
Frisco	N.D.	4879
Galena	McG.	593
Galena Fr.	McG.	4895
Galt	W.	1650
Galt	S.N.	5194
Gem	McG.	1858
General Sheridan	S.N.	2066
Gentle Annie	B.	4184
Gibralter	McG.	3166
Gladstone	McG.	2286
Glasgow	J.	3321
Gleaner	P.	2344
Good Day Fr.	P.	2345
Goodenough	S.N.	581
Gopher	S.N.	1851
Gordon	S.N.	5196
Grace	S.S.	5532
Grand View	S.S.	3998
Great Eastern	S.S.	753
Great Eastern No. 2	S.N.	2289 A. 1
Great Western	McG.	548
Greenhorn	C.	1306
Grey Copper	S.N.	580
Grey Eagle	F.	2137
Grover	F.	1330
Halifax Fr.	S.S.	4001
Halton Chief	N.F.	2158
Handy	P.	2146
Happy Delivery	N.D.	4973
Hardscrabble	P.	2135
Harriet	J.	3319
Hartney	N.D.	4864
Hazard	F.	1397
Hazard Fr.	F.	1400
Hazel	W.	2636
Hazel C.	W.	1424
Heber Fr.	S.S.	4681
Helena	S.S.	4238
Herbert	McG.	2632
Hewitt	F.	4440
Hidden Treasure No. 2	S.S.	1715
Highland	S.S.	1337
Hill Side	F.	4514
Hill Side	W.	1651
Hill Top Fr.	S.S.	2849
Hinckley	P.	1720
Home Rule	C.	2053
Hoodo	C.	3188
Hope	S.S.	842
Howard	W.	2828
Hub & Hub	N.D.	4869
Humboldt	F.	2225
Humbolt	W.	3644
Humphrey	McG.	3165
Hunter	N.D.	4868
Hustler	W.	2431
Hustler	A.	1886
Hustler Fr.	S.S.	3619
I. C.	McG.	4893
Idaho	A.	472
Idaho No. 2	C.	1013
International Fr.	P.	2834
Ireland	S.S.	3986
Irene	W.	1171
Irene Fr.	S.S.	4530
Iron Chest	W.	3016
Iron Duke	W.	3196
Iron Hill	S.S.	3520
Iron Mask	N.D.	3521
Iron Mask Fr.	W.	2841
Iron Mountain	B.	4182
Ironside	B.	4873
Isis	S.S.	743
Ivanhoe	S.S.	738
Ivy Leaf	A.	[illegible]
I.X.L. Fr.	S.S.	5366
Jay Gould	C.	1754
J. C.	J.	2263
Jeanie	N.D.	3629
Jennie	S.S.	546
Jennie	S.S.	3172
Jennie No. 3	McG.	1713
Jenny Jones	F.	2534
Jenny Lind	F.	1866
Jessie	S.N.	620
J.I.C.	F.	2533
Jim Fair	McG.	969
John Plummer Fr.	B.	851
John W. Mackey	McG.	968
Jo-Jo	N.F.	1839
Kaas Fr.	S.N.	3100
Kaslo	W.	822
Kaslo	C.	2432
Kaslo Fr.	C.	7301
Kate	F.	1333
Katie D.	S.S.	2848
Keewatin	S.S.	5761
Kelpie	N.D.	2809
Kelso	F.	2604
Keno	McG.	530
Kenef	C.	2492
Keyser Fr.	T.	3623
Keystone	N.F.	1585
Keystone Fr.	W.	2179
King Fr.	S.N.	1069
Kingston	N.D.	4871
Kingston	W.	3104
Kinkora	A.	2104
Kitchener	J.	4487
Kitchener Fr.	S.S.	3107
Knoxville	S.N.	466
Kootenay Star	J.	1168
Kopje Fr.	N.D.	3749



INDEX—*Cont'd.*

Name of Claim.	Locality.	Official Lot and Reference No.
Labour Day Fr	J.	3446
Lady Jane	C.	1305
Lake Shore	N.D.	2810
Lake View	N.D.	3630
Lake View	N.D.	3732
Lake View Fr.	N.D.	3734
Lalla Rookh	P.	1856
Last Chance	S.N.	717
Last Chance	W.	1649
Last Chance No. 4	McG	3516
Last Link	W.	1425
L. D. Fr.	S.N.	4535
Legal Fr.	S.N.	4534
Legal Tender	S.N.	1749
Le Roi	F.	5754
Liberal	W.	2271
Lillian No. 4	P.	1724
Lillie Fr.	J.	1159
Lily Fr.	N.F.	2494
Link Fr.	N.D.	3745
Lipton	McG.	1030
Little Estella Fr	S.S.	5534
Little Ralph	W.	2429
Little Widow Fr.	S.N.	1850
London	B.	1416
Lone Batchelor	T.	4564
Lone Jack	McG.	2633
Lone Star	S.S.	1844
Lone Star No. 3	N.D.	1891
Look Out No. 2	S.S.	1308
Lorna Doone	F.	1401
Lorna Doone Fr.	F.	4441
Lost Bear	F.	6871
Lost Tiger	N.D.	3744
Loudoun	S.S.	727
Lucetta	S.N.	599
Lucky Boy	J.	632
Lucky Edd	J.	5999
Lucky Jim	B.	844
Madison	S.N.	1411
Madison Extension	S.N.	5192
Maid of Erin	P.	500
Majestic	P.	1405
Major Fr	McG.	4896
Mammoth	N.D.	1910
March	F.	1392
Mardon Fr.	S.N.	1420
Marie Fr	S.N.	6870
Marion	N.D.	2287
Mary S. Fr.	A.	3533
Maud E.	S.N.	463
Maud S	W.	1422
May	P.	7299
Mayflower	W.	4458
Mazeppa	A.	4510
Medford	T.	3622
Mentor Fr.	A.	3180
Mercury	P.	3531
Merrimac	N.D.	3746
Metlakhatla	W.	3334
Miantonomah	A.	1461
Midnight	F.	3227
Midnight Fr.	S.S.	2292
Miller Cr. Fr	S.S.	5191
Millie	S.S.	6914
Milton	N.F.	2159
Milton Fr.	N.F.	3825
Mine	A.	3099
Mineral Hill	S.S.	2285
Miner Boy	B.	4915
Minneapolis	McG.	578
Minnesota	S.S.	6915
Minnie	S.S.	5535
Minnie	J.	4890
Minniehaha	S.S.	3171
Mocking Bird	A.	1265
Mohawk	F.	2537
Mona Fr	C.	7300
Monday	A.	3094
Monday Fr	A.	3095
Monitor	T.	1916
Monte Carlo	J.	635
Mole	F.	3364
Mollie	S.N.	621 A.
Mollie Hughes	N.D.	2106
Mollie O.	A.	1913
Morn Fr.	A.	748
Morning No. 3.	S.S.	3105
Morning Star No. 2	J.	4885
Morning Sun	S.S.	4856
Moses	B.	856
Mountain Boomer	F.	740
Mountain Chief	P.	498
Mountain Chief No. 2	N.D.	474
Mountain Chief No. 3	F.	4470
Mountain Goat No. 2	J.	629
Mowitch	N.D.	4558
Myrtle R.	W.	1418
Millsites		
B. N. White Co.	S.S.	590
Idaho-Alamo Mines	A.	736
Nabob	C.	3999
Nancy Hanks	W.	1421
Naoma	C.	2516
Naoma Fr.	C.	2519
Napier	F.	2299
Nellie Fr	T.	3108
Nettie Fr.	F.	1249
New Denver	S.N.	612
New England	S.S.	3987
New Park	F.	1398
Newport	T.	4521
New Springfield	S.S.	5363
Night Hawk Fr.	S.S.	2032
Nil Desperandum	B.	2806
Noble 5	S.N.	467
Nonpareil	B.	4179
Nonpareil Fr.	B.	4554
Noonday	F.	1334
Noonday	C.	2136
Northern Bell	J.	1166
Northern Belle	McG.	3173
Northern Pacific	McG.	3174
Number One	S.N.	4560
Number Two	S.N.	4561
Number Three Fr.	C.	3532
Oakland	F.	4876
O. B. H	P.	2644
Ocean	P.	1723
Ogema	McG.	3163
O. K.	N.F.	2156
Okanagan	McG.	454
Old Man	N.D.	2811
Oma	F.	2616
Omega	S.N.	618
Oom Paul	S.S.	5973
Opatunka	A.	3523
Opher	J.	1169
Ophir No. 3	S.S.	5368
Oregon	A.	3098
Oregon	W.	2275
Orient Fr.	S.S.	5831
Oro	C.	2434
Oscar Fr	J.	3322
Ottawa No. 2	F.	1805
Ouray	T.	3109
Ouray Fr.	T.	1017
Pacific	McG.	597
Palmita	T.	4880
Pamlico	C.	2843
Panama	B.	3152
Paupers Dream	W.	1163
Pay Rock	N.F.	2157
Paymaster	T.	5732
Payne	P.	499
Payne Fr.	P.	2646
Peerless (Revised)	F.	1332
Pelly	F.	2298
Pembroke	F.	1399
Pembroke	S.S.	6912
Penobscot	F.	3367
Peoria	J.	3318
Phoenix	N.D.	2358
Phroso	B.	852
Pinto	N.D.	2107
Pirate	C.	1304
Polo	S.N.	2260
Portland No. 5	T.	2620
Prescott	F.	3028
Prescott Fr. No. 1	F.	3029
Prior	F.	1402
Purcell	S.N.	849
Queen Bess	A.	215
Queen Fr.	A.	3179
Rabbit Paw	S.S.	1252
Rambler	McG.	1246
Random Shot	S.N.	1726
Rawdon	C.	855
Read	F.	1247
R. E. Adams	J.	1652
Real Idea No. 2	N.D.	2105
Reciprocity	P.	1722
Red Cross	McG.	3176
Red Fox	McG.	2413
Red Fox Fr.	McG.	2414

INDEX—*Cont'd.*

Name of Claim.	Locality.	Official Lot and Reference No.
Redress No. 2 Fr	P.	3209
R. E. Lee	McG.	579
Reliance	B.	4185
Return	P.	1018
Revenue	W.	2826
Ricardo	F.	2300
Richmond	S.S.	1472
Rienzi	S.S.	1262
Rincon	F.	3363
Rincon Fr.	F.	3362
Rio	McG.	2093
Roadley	B.	858
Roanoke Fr.	B.	854
Robertson	F.	1808
Robin	F.	1404
Rosedale	P.	7306
Rose Marie	N.D.	4003
Royal Five	N.D.	3628
Ruben	P.	1915
Ruby Silver	McG.	515
Ruecau	S.N.	624
Rugby Fr	A.	3527
Rushford	S.N.	2068
Ruth	S.S.	841
Ruth Fr.	S.S.	1845
Rutland Fr.	A.	3526
Saddie	McG.	1302
St. Charles	S.N.	3264
St. Clair	N.D.	4559
St. George	B.	846
St. John	A.	473
St. Keverne	P.	2642
Salem	J.	5601
Sampson	S.S.	5365
San Antonio	J.	5602
Sandon Chief	S.S.	3258
Sandow	N.D.	1890
Sapphire	P	1857
Sarah B	F.	2847
Sarnia	F.	4471
Satisfaction	N.D.	3110
Scotland	S.S.	3988
Seaton Fr.	S.S.	4848
Seattle	B.	4178
Seattle Fr.	B.	4555
Second Extension	A.	6803
Seelig Fr	N.D.	4855
Shareholder	B.	3736
Shields	B.	847
Shiloh	B.	850
Shoeswap	C.	3158
Shogo	S.S.	4857
Shoshone	C.	4002
Shunia	S.N	2067
Shunieaw	F.	718
Silver Bell	A.	1887
Silver Bell No. 2	McG.	2092
Silver Bill	P.	3530
Silver Cord	McG	1848
Silver Cord	C.	3739
Silver Glance	B.	3829
Silver Reef	P.	3996
Silver Smith	S.S.	1010
Silver Star	F.	2297
Silver Star Fr	S.S.	4878
Silver Tip Fr.	C.	4881
Silverton Boy	F.	2536
Silver Wedge Fr	F.	5756
Sincher	McG.	1303
Sir Charles	J.	627
Slater	S.S.	752
Sligo Fr.	N.D.	4526
Slocan Belle	S.S.	1921
Slocan Boy	P.	626
Slocan King	S.S.	547
Slocan Star	S.S.	545
Slocan Sovereign	S.N.	1927
Snap	B.	911
Snowflake	C.	1847
Snowflake	J.	3320
Snowstorm	P.	920
Snowstorm Fr	S.N.	3445
Soho	McG.	3175
Speculator	T.	3994
Spokane	McG.	3515
Spokane-Kaslo	W.	5849
Spokane-Kaslo No. 2	W.	5850
Standard	F.	564
Stanley	McG.	2102
Stanley No. 2	N.D.	4872
Starlight No. 3	S.N.	595
Starlight Fr.	McG.	2343
Starview	S.S.	1473
Stevenson	F.	1331
Stormount	McG	4117
Stranger	J.	512
Suburban Fr.	S.S.	2031
Sultana	F.	3519
Summit	S.S.	4374
Summit Fr	S.N.	5599
Summit Queen	B.	3830
Sunlight	J.	2058
Sunrise	S.S.	5749
Sunrise	J.	4886
Sunrise No. 4	C.	4562
Sunset	W	970
Sunset	J.	1164
Sunshine	A.	3096
Surprise	F.	563
Surprise No. 2	McG.	514
Surprise Extension	McG.	5600
Sylvanite	N.D.	4866
Tawanda	P.	1920
Telephone	P.	3185
Tenderfoot	F.	1248
Tennie C.	W.	1419
Texas	S.N.	589
Texas	J.	4889
The Ben	F.	2605
Third of July	B.	1417
Thistle	A.	4515
Thursday Fr	P.	3524
Tom Bowling	S.S	4549
Tonka Fr.	A.	4509
Toothpick Fr.	C.	2816
Tornado	S.S.	5537
Toronto	J.	6000
Trade Dollar	C.	1432
Tramp Planet Fr	P.	2359
Tranquility	F.	3366
Treasure Vault	S.N.	587
Triumph	S.S.	4552
Triumph Fr.	S.S.	4849
Tryon	N.D.	2108
Twilight	S.N.	1854
Twin Lakes	A.	747
Two Jacks	P	497
Tyro	N.D.	3111
Tyro Fr.	N.D.	3113
Turris	N.D.	4523
United	C.	2059
Unexpected	P.	2231
U. S.	J.	1055
Vancouver	W.	2024
Vancouver No. 2	F.	739
Vancouver Fr	F.	2900
V. Fr.	A.	749
Vernon	B.	4181
Victor	S.S.	3184
Victor	T.	4565
Victoria No. 6	S.N.	3154
Violet	S.S.	3168
Violet Fr.	S.S	3170
Vult	C.	4483
Vulture	C.	4482
Vulture Fr.	C.	3672
Wanacott	McG.	404
Wakefield	F.	1527
Wakefield Fr.	F.	1810
Wallace	N.D.	4524
Washington	McG.	541
Washington Fr.	McG.	4894
Wedge Fr	W.	2267
Wellington	W.	553
Wellington	C.	1755
Western	T.	565
Whistler Fr.	S.S.	6916
White Elephant	J.	3159
Whitewater	W.	1170
Whitewater Deep Fr	W.	2268
W. H. R.	F.	2535
Wild Goose	S.N.	614
Wild Goose No. 2	A.	1263
Windsor	S.S.	1016
Wonderful	S.S.	481
Wonderful Fr.	S.S.	5364
Worlds Fair	S.N.	464
Wyoming	S.S.	754
Yakima	A.	3097
Yosemite Fr.	W.	2270
Young Dominion	A.	458
Young Dominion Fr.	A.	2296
Yuma	S.S.	2029
Yuma Fr	S.S.	2037
Zilor	F.	741
Zuni	J.	4898

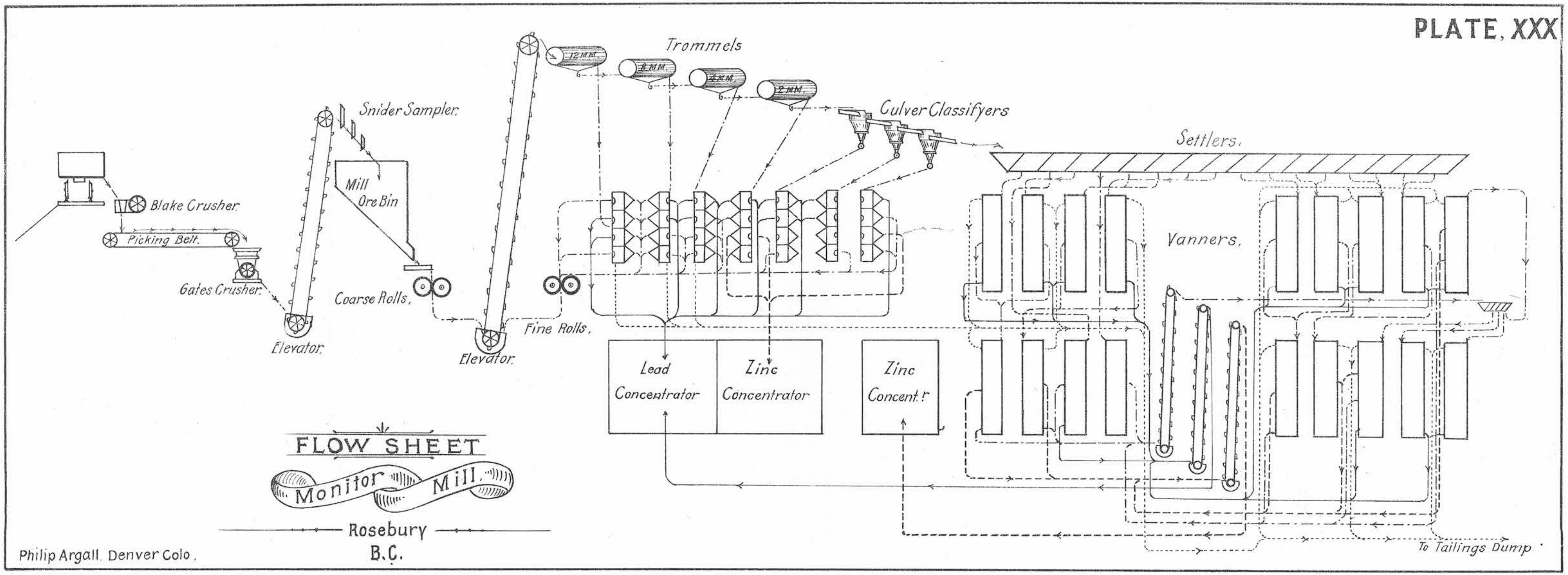
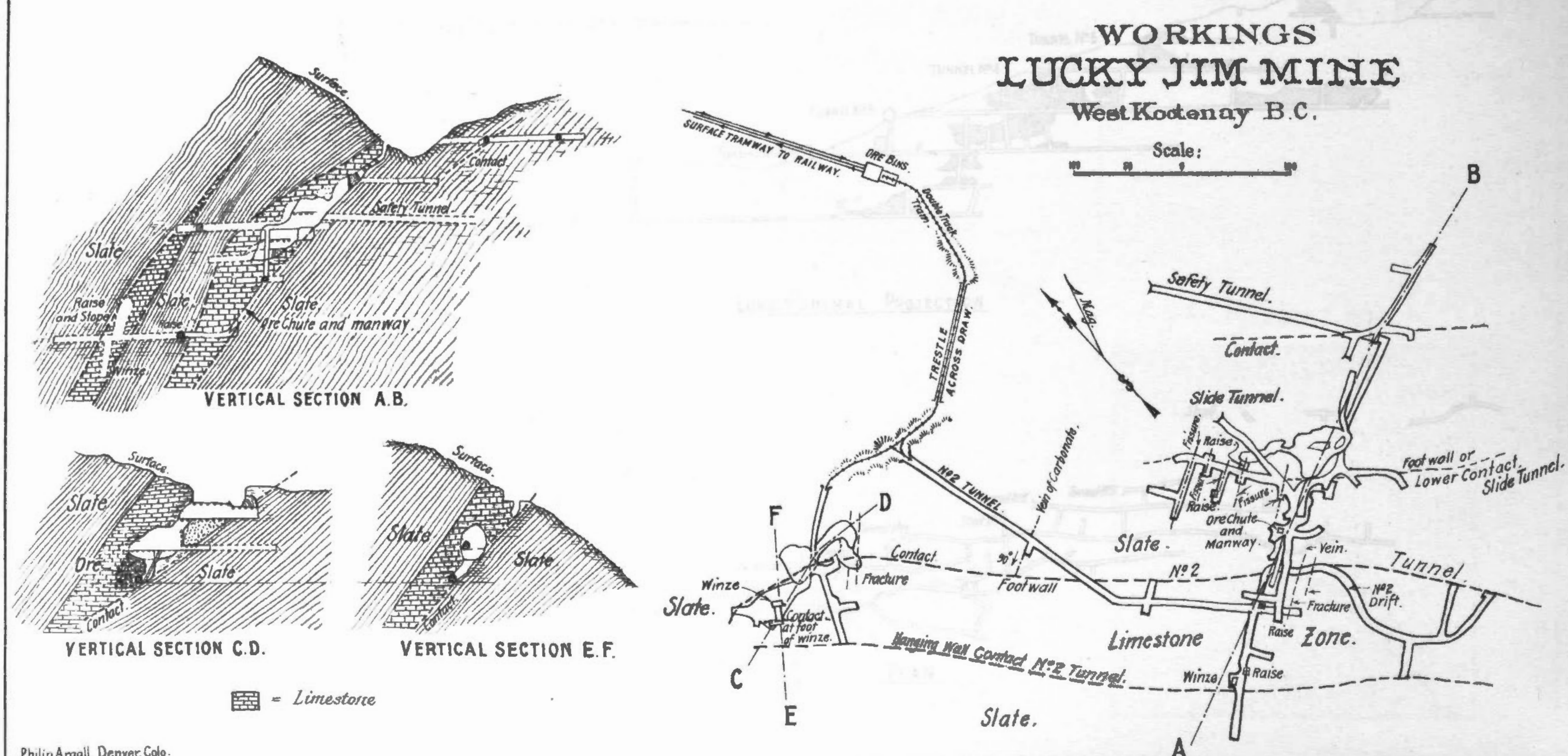


PLATE. XXX
Trommels
12MM.
8MM.
4MM.
2MM.
Snider Sampler.
Culver Classifyers
Settlers.
Mill Ore Bin
Blake Crusher.
Picking Belt.
Gates Crusher.
Coarse Rolls.
Elevator.
Elevator.
Fine Rolls.
Vanners.
Lead Concentrator
Zinc Concentrator
Zinc Concent.r
FLOW SHEET
Monitor Mill.
Rosebury
B.C.
Philip Argall. Denver Colo.
To Tailings Dump

PLATE XXXI.



WORKINGS
LUCKY JIM MINE
West Kootenay B.C.
Scale:
Surface.
Contact
Safety Tunnel
Slate
Raise and Stope
Slate
Raise
Slate
Ore Chute and manway.
Winze
VERTICAL SECTION A.B.
Surface.
Slate
Ore
Slate
Contact.
VERTICAL SECTION C.D.
Surface.
Slate
Slate
Contact.
VERTICAL SECTION E.F.
= Limestone
SURFACE TRAMWAY TO RAILWAY.
ORE BINS.
Double Track Tram.
TRESTLE ACROSS DRAW.
Mag.
Safety Tunnel.
Contact.
Slide Tunnel.
Fissure
Raise.
Fissure
Raise.
Fissure.
Ore Chute and Manway.
Foot wall or Lower Contact.
Slide Tunnel.
Vein.
Fracture
Nº 2 Drift.
Nº 2
Tunnel.
Nº 2 TUNNEL.
Vein of Carbonate.
30°
Footwall
Contact
Fracture
Winze
Slate.
Contact at foot of winze.
Slate.
Limestone
Zone.
Raise
Winze
Raise
Hanging Wall Contact Nº 2 Tunnel.
Slate.
A
B
C
D
E
F
Philip Argall. Denver. Colo.

PLATE XXXII

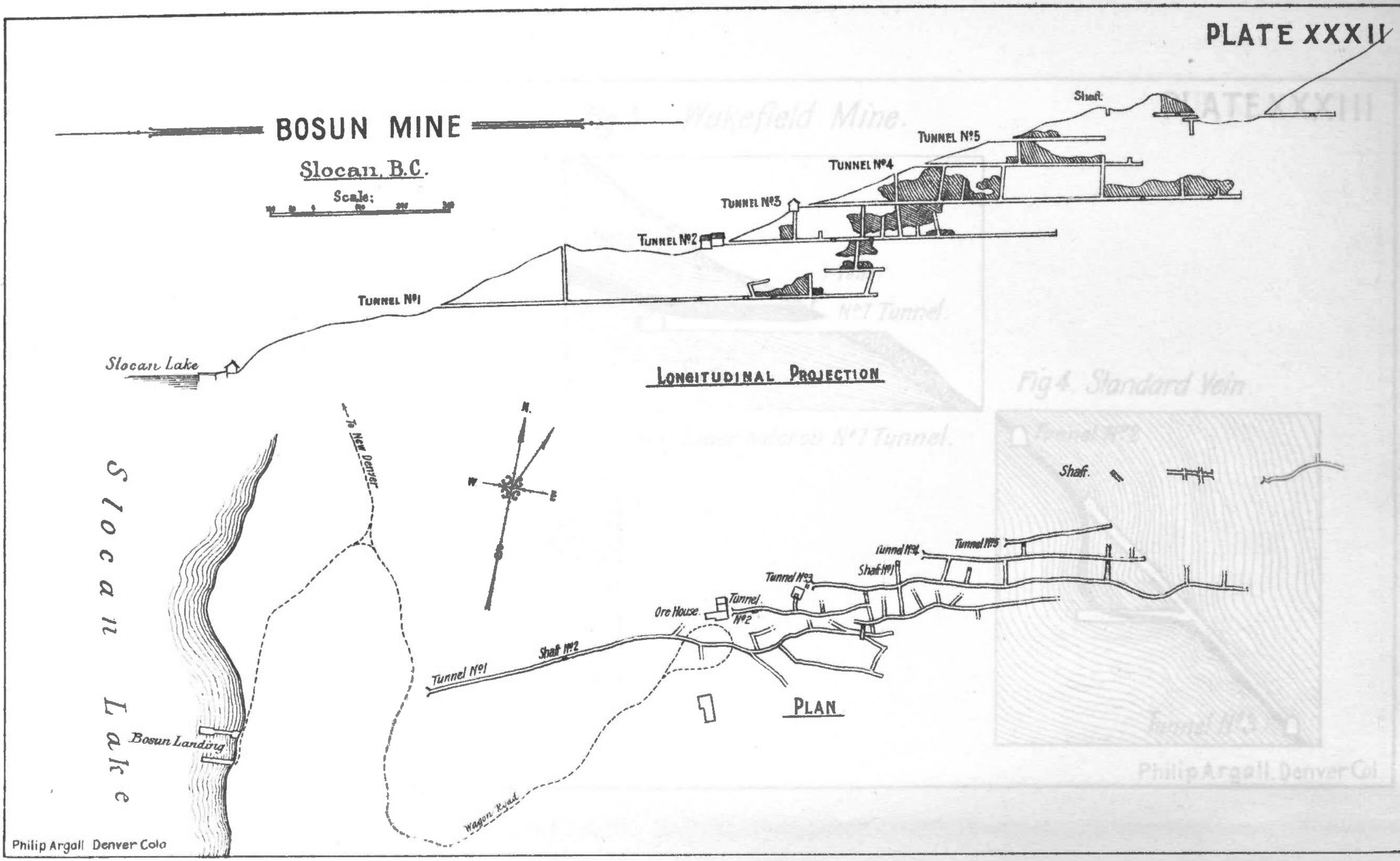

BOSUN MINE
Slocan, B.C.
Scale:
Shaft
TUNNEL No 5
TUNNEL No 4
TUNNEL No 3
TUNNEL No 2
TUNNEL No 1
Slocan Lake
LONGITUDINAL PROJECTION
To New Denver
N.
W
E
Slocan Lake
Tunnel No 5
Tunnel No 4
Shaft No 1
Tunnel No 3
Tunnel No 2
Ore House
Shaft No 2
Tunnel No 1
PLAN
Bosun Landing
Wagon Road
Philip Argall Denver Colo

PLATE XXXIII

Fig 1. Wakefield Vein,
Nº 2 Tunnel.

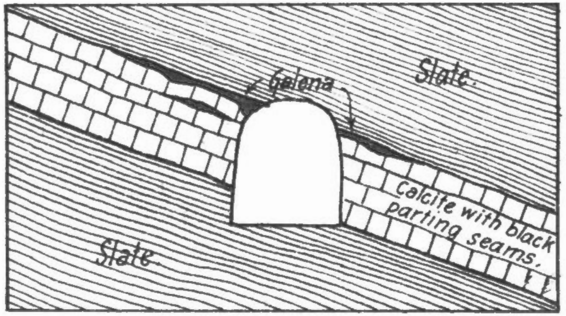
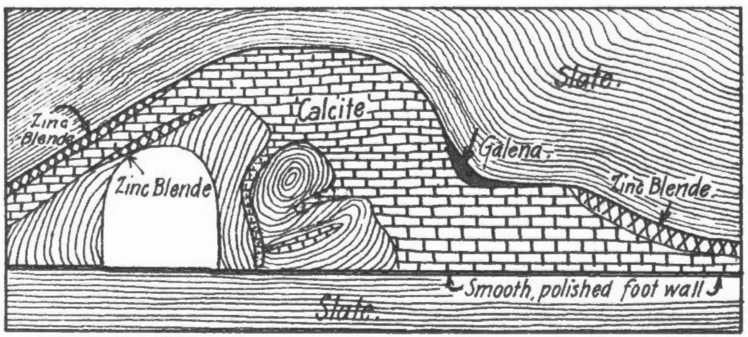
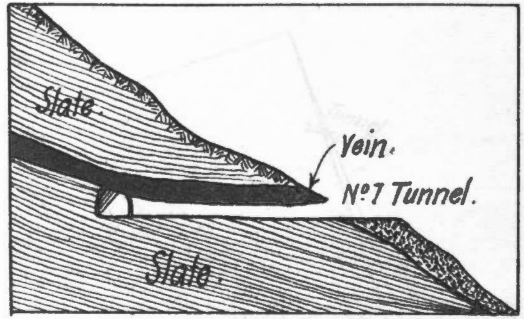


Fig. 2. Wakefield Mine.



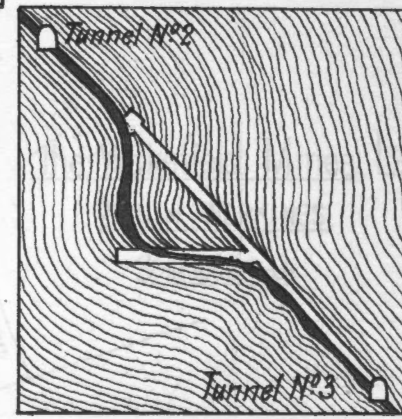
Showing a roll in the vein over Nº 3 Tunnel.

Fig 3. Wakefield Mine.

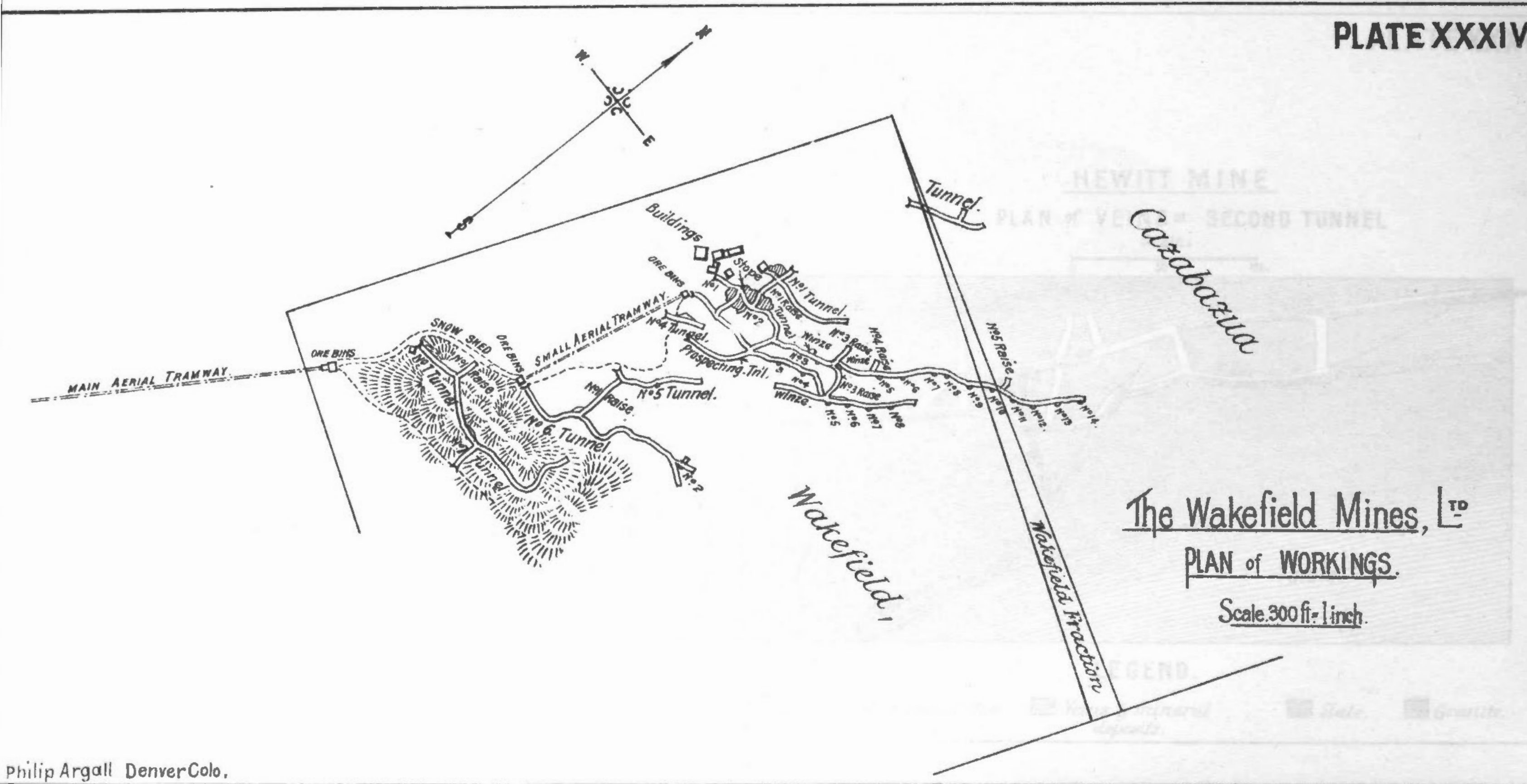


Showing Lower outcrop Nº 1 Tunnel.

Fig 4. Standard Vein.



Philip Argall, Denver Col.



Philip Argall Denver Colo.

PLATE XXXV.

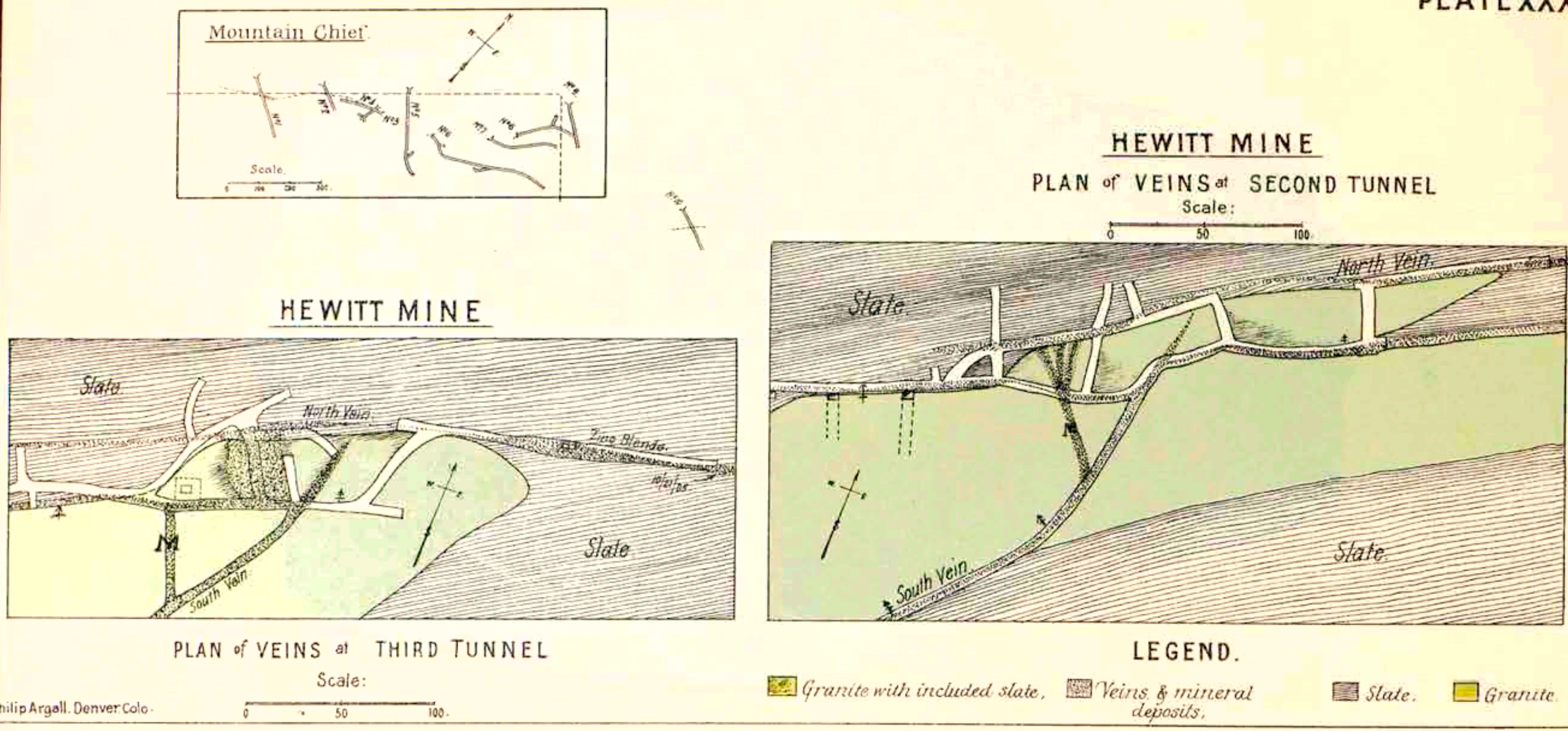
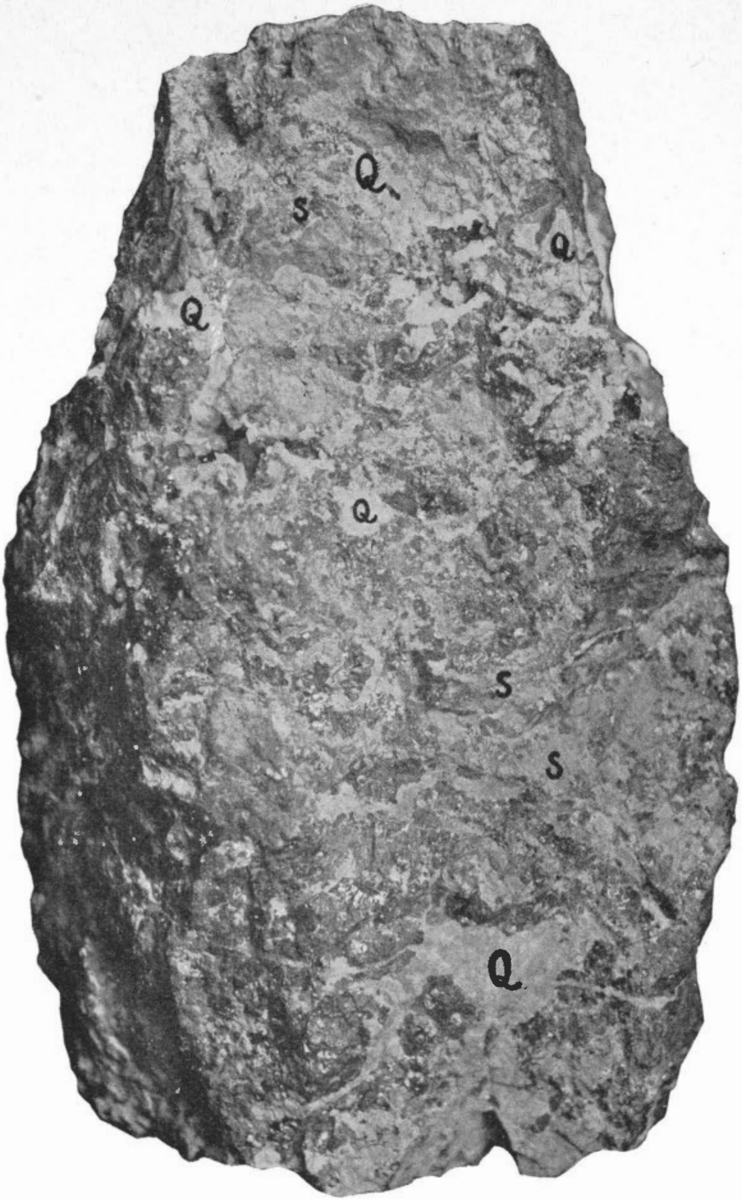

Mountain Chief.
Scale.
HEWITT MINE
PLAN of VEINS at SECOND TUNNEL
Scale:
0
50
100
North Vein.
Slate.
South Vein.
Slate.
HEWITT MINE
Slate.
North Vein.
Zinc Blende.
Slate.
South Vein.
PLAN of VEINS at THIRD TUNNEL
Scale:
0
50
100.
LEGEND.
Granite with included slate.
Veins & mineral deposits.
Slate.
Granite.
Philip Argall. Denver Colo.

PLATE XXXVI.



HEWITT ORE.
Q—Quartz. S—Siderite. The dark bands are zinc (Z).

PLATE XXXVII.

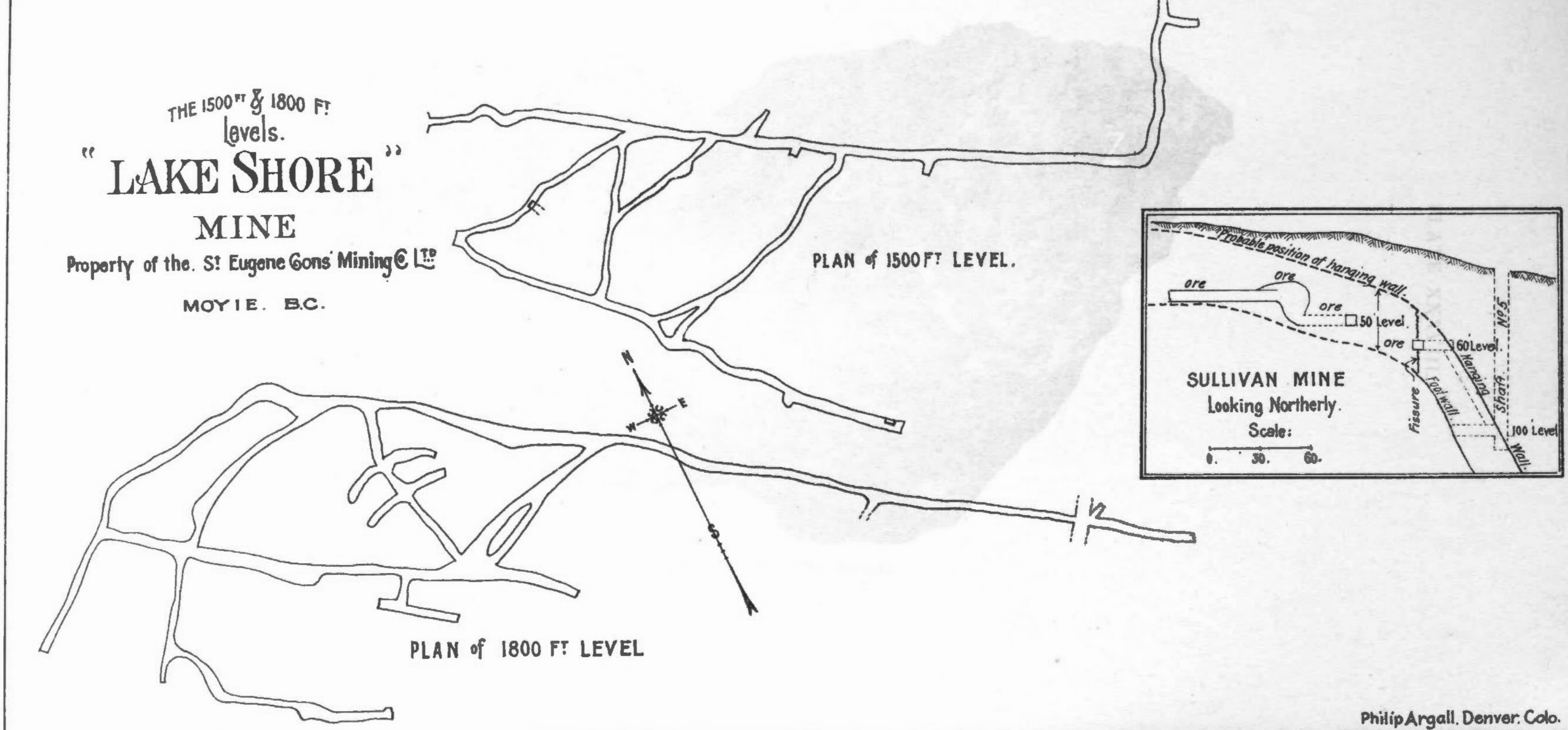
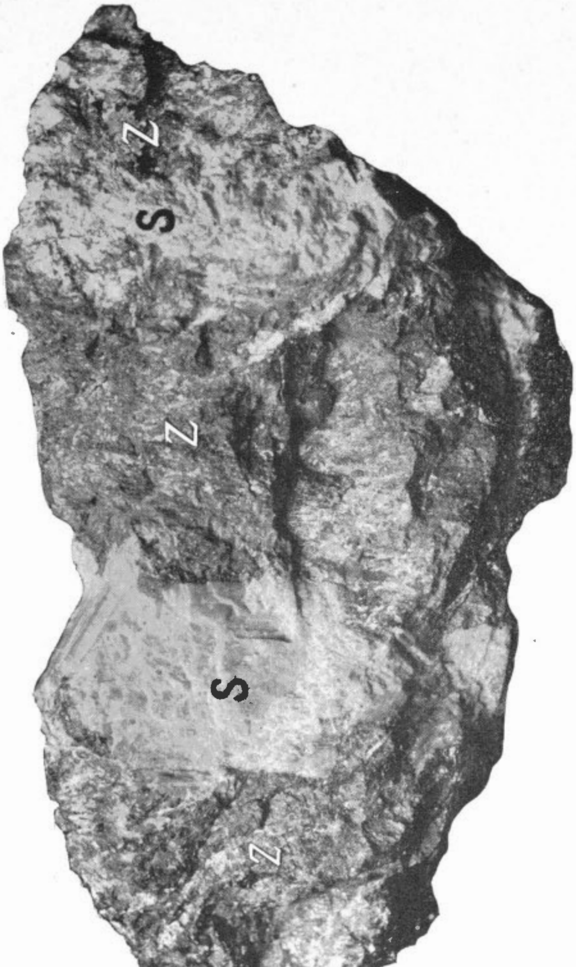


PLATE XXXVIII.

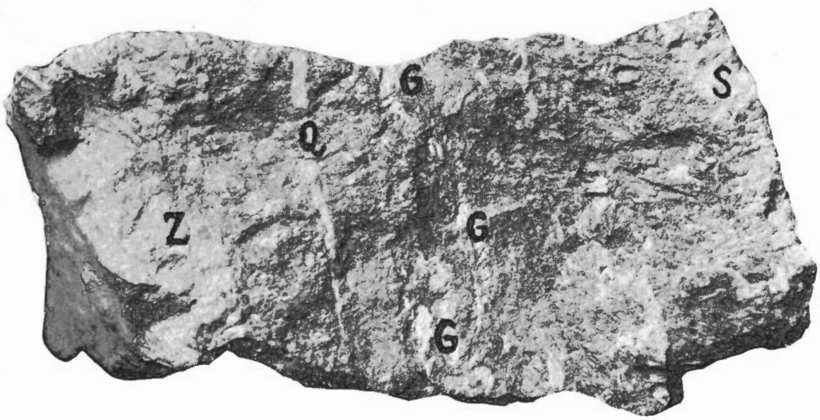


S—Siderite. Z—Zinc blende.

PLATE XXXIX.

S—Siderite. Z—Zinc blende.

PLATE XL.



Z—Zinc blende. Q—Quartz. S—Siderite. G—Galena.

PLATE XLI.

Calcite from Wakefield Vein.

PLATE XIX

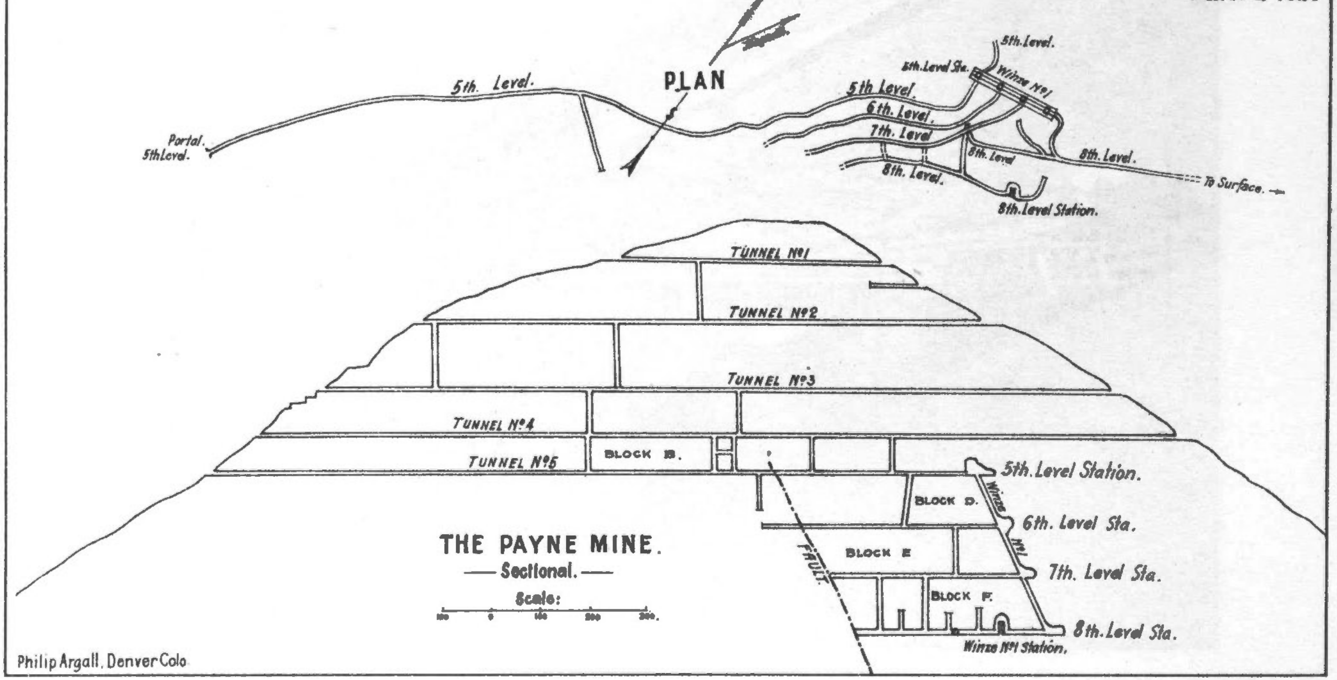
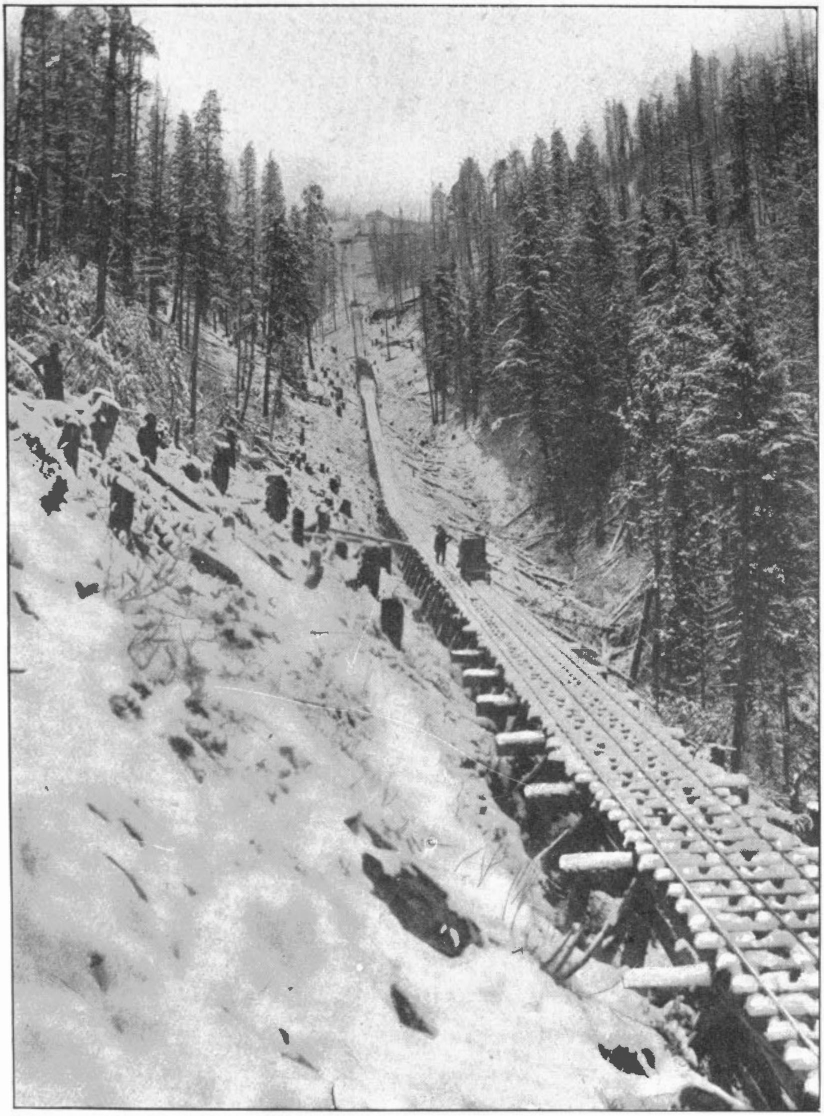

PLAN
5th. Level.
Portal.
5th Level.
5th. Level.
5th Level Sta.
Winze No1
5th Level.
6th. Level.
7th. Level
8th. Level
8th. Level.
8th. Level.
To Surface.
8th. Level Station.
TUNNEL No1
TUNNEL No2
TUNNEL No3
TUNNEL No4
TUNNEL No5
BLOCK B.
BLOCK D.
BLOCK E
BLOCK F.
FAULT
Winze No1
5th. Level Station.
6th. Level Sta.
7th. Level Sta.
8th. Level Sta.
Winze No1 Station.
THE PAYNE MINE.
— Sectional. —
Scale:
Philip Argall, Denver Colo.

PLATE XX.

PAYNE LOWER TERMINAL.

PLATE XXI.

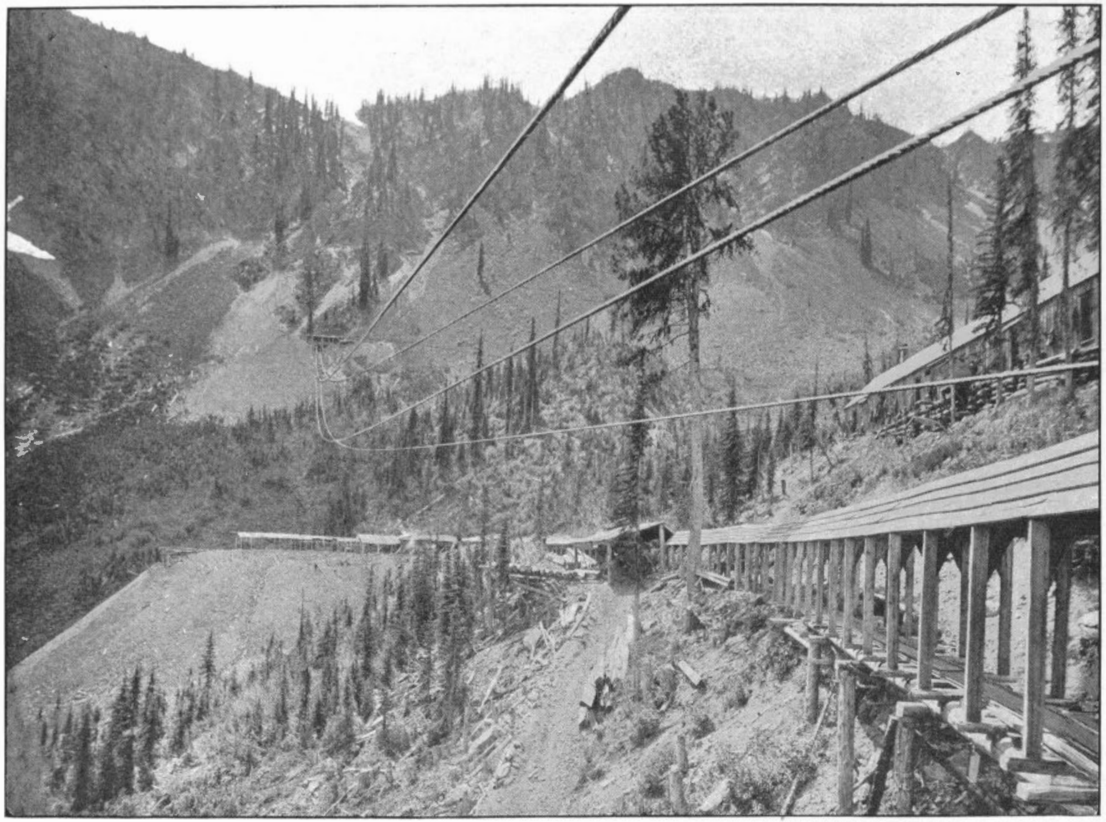


Payne Tramway, Sandon, B.C.

PLATE XXII.

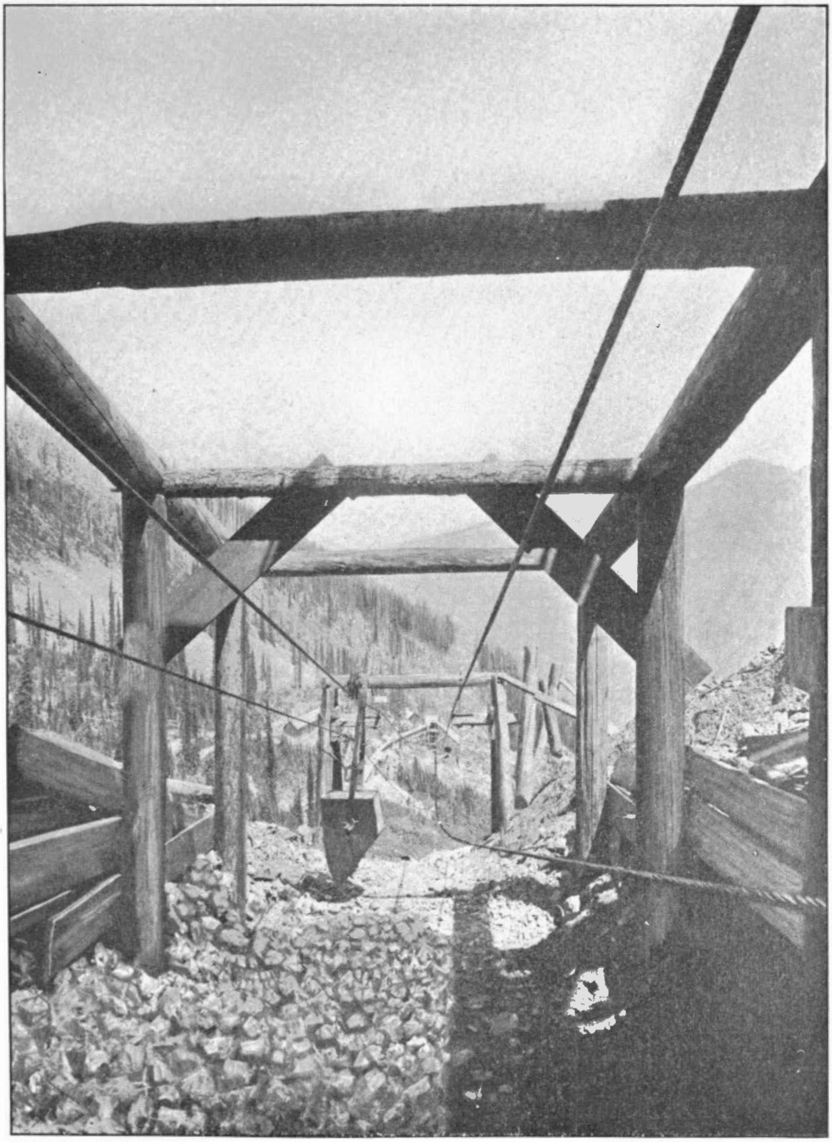
IVANHOE MILL.

PLATE XXIII.



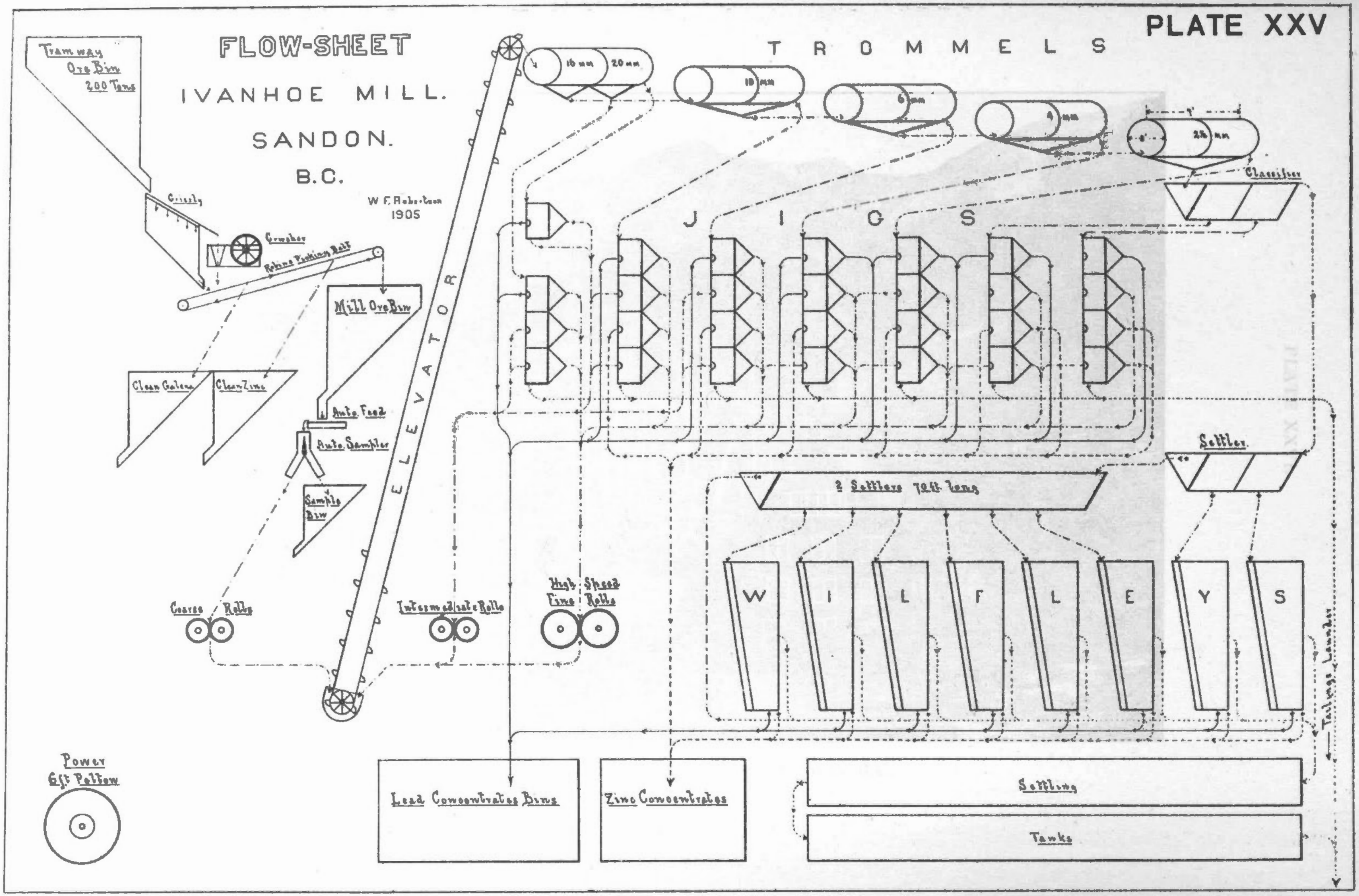
IVANHOE, NO. 2.

PLATE XXIV.



IVANHOE WIRE TRAM.

PLATE XXV


FLOW-SHEET
IVANHOE MILL.
SANDON.
B.C.
W. F. Robertson
1905
Tramway Ore Bin 200 Tons
Grizzly
Crusher
Robins Picking Belt
Mill Ore Bin
Clean Galena
Clean Zinc
Auto. Feed
Auto. Sampler
Sample Bin
ELEVATOR
TROMMELS
16 mm
20 mm
10 mm
6 mm
4 mm
2½ mm
Classifier
JIGS
Settler
2 Settlers 70 ft. long
WILFLEYS
Tailings Launder
Coarse Rolls
Intermediate Rolls
High Speed Fine Rolls
Power 6 ft. Pelton
Lead Concentrates Bins
Zinc Concentrates
Settling
Tanks

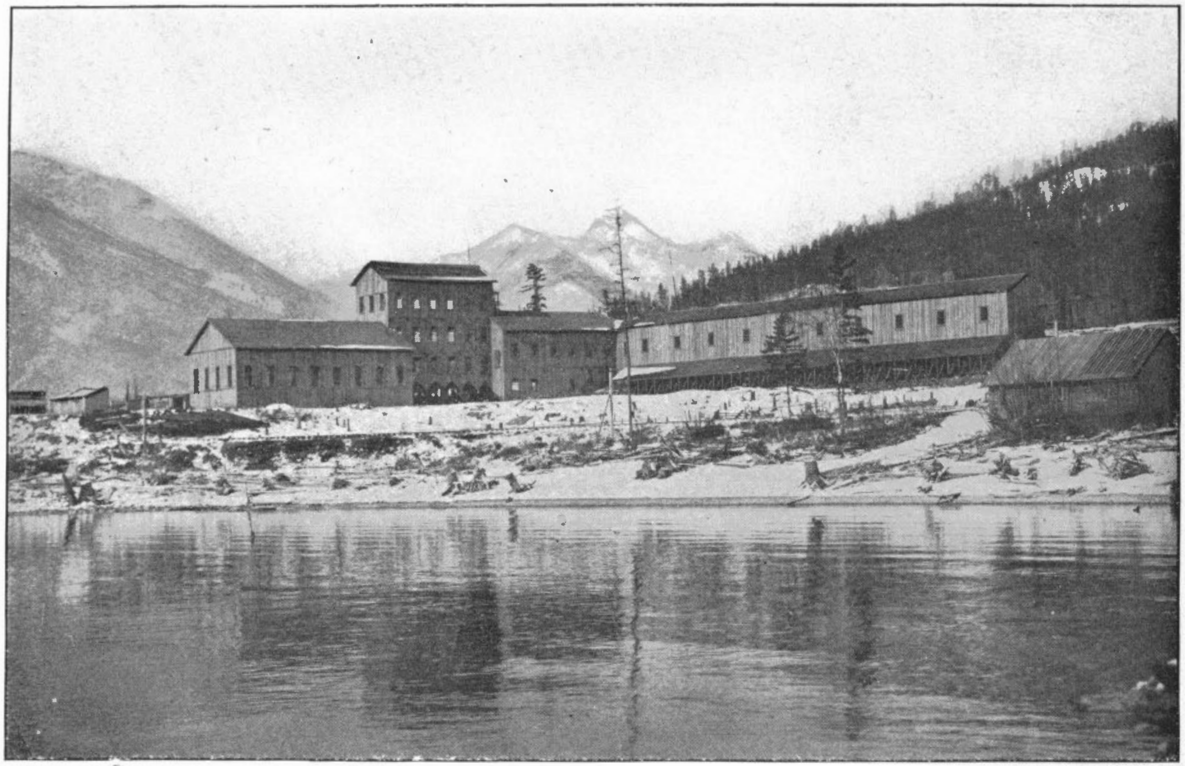
Noble Five Concentrator. Sandon, B.C.

PLATE XXVI.

PLATE XXVII.

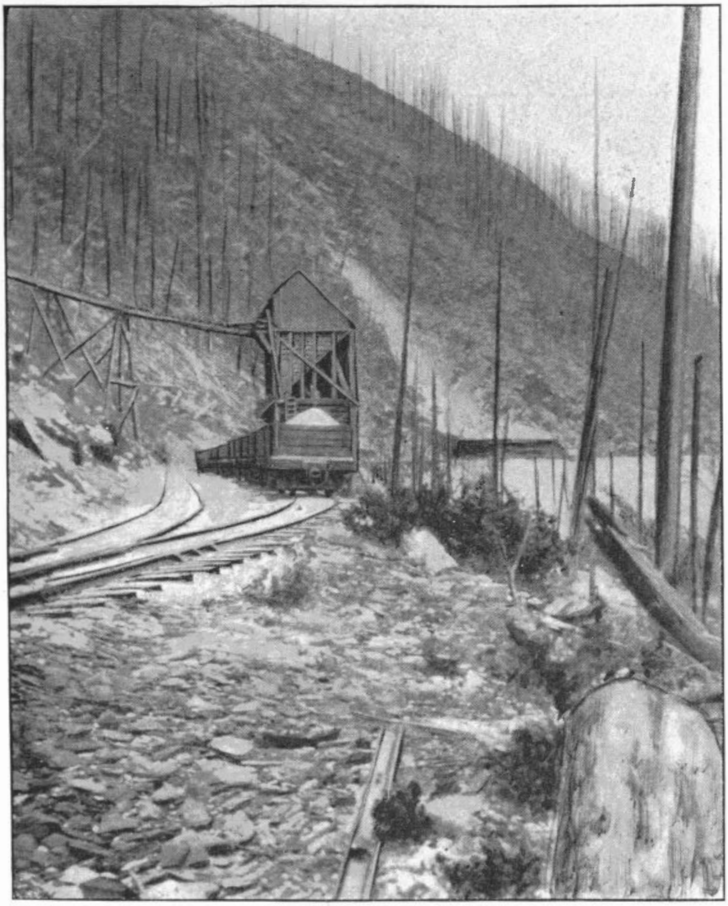
NOBLE FIVE TRAMWAY, CODY, B.C.

PLATE XXVIII.



MONITOR MILL, BROCKMAN, B.C.

PLATE XXIX



LUCKY JIM.

Lower Terminal of Tram on Kaslo and Slocan R.R.

PLATE VII.

A southerly view of the main open cut on the Blue Bell Mine, showing a face of mineralized limestone 60 feet high from the place where the man is standing. The Glory Hole is the connection to the big galleries or pillared stopes below; these stopes vary from 80 feet to 100 feet in width.

PLATE VIII.

A NORTHERLY VIEW OF THE MAIN OPEN CUT ON THE BLUE BELL MINE.

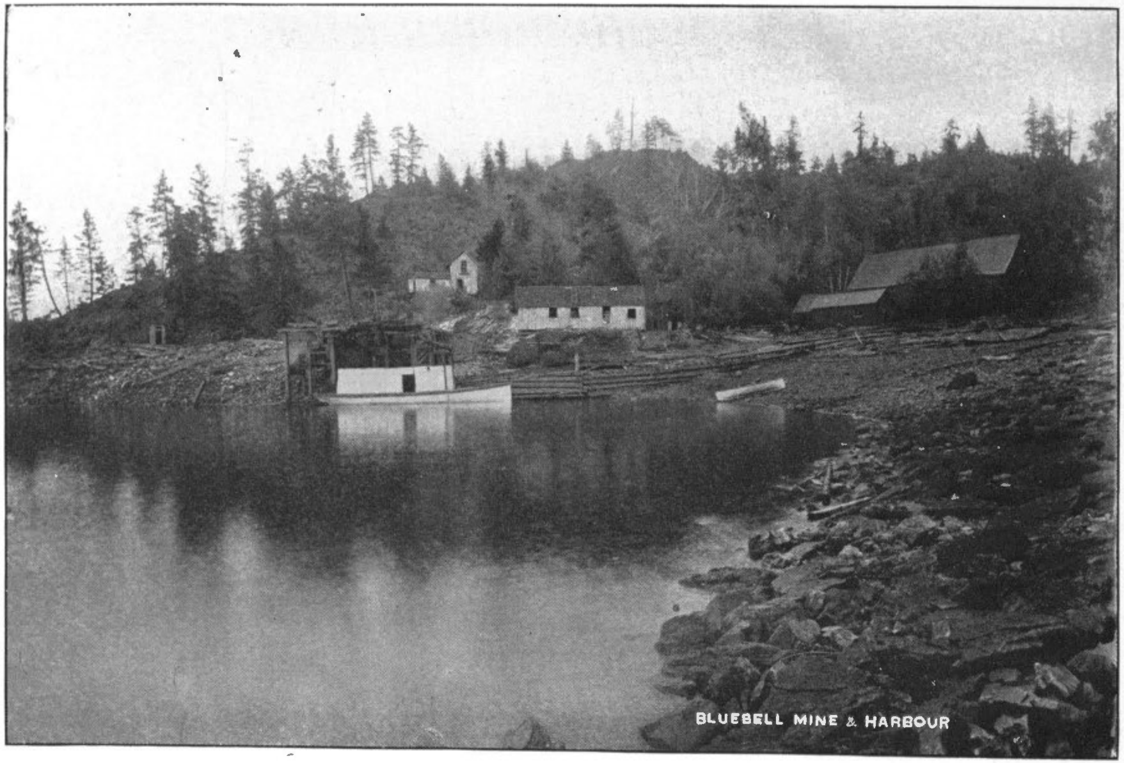
(A) Shows the tramway aud floor of the old Hudson Bay tunnel.
(B) Oxidized vein material of one of the principal fissures.
(C) Shows the foot wall of this fissure vein in the limestone.
(D) The opening into the smaller and paralell surface stope.
(E) The sheeted structure is well seen at this place.
(F) Capping of Muscovite schist. All the rock below this point is crystalline limestone containing irregular deposits of the mixed sulphides of lead, zinc and iron, the surface oxidation of which has stained the rock brownish red.

PLATE IX.

WESTERLY VIEW, LOOKING INTO THE FACE OF SMALLER OPEN CUT.

(A) Shows a bed of pyrites, galena and zinc blende, practically solid ore.

(B) One of the numerous small fissures, clearly seen, because filled with oxidized ore. The sheeting structure is well seen in the face of the stope.

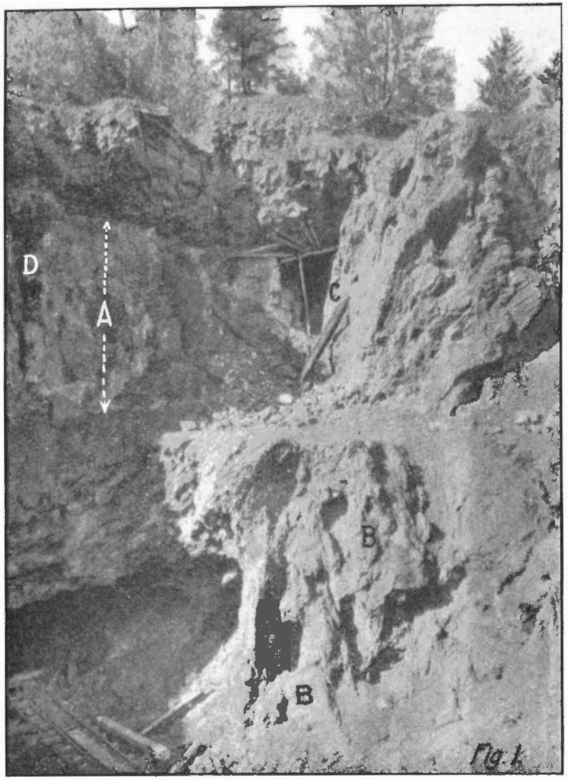


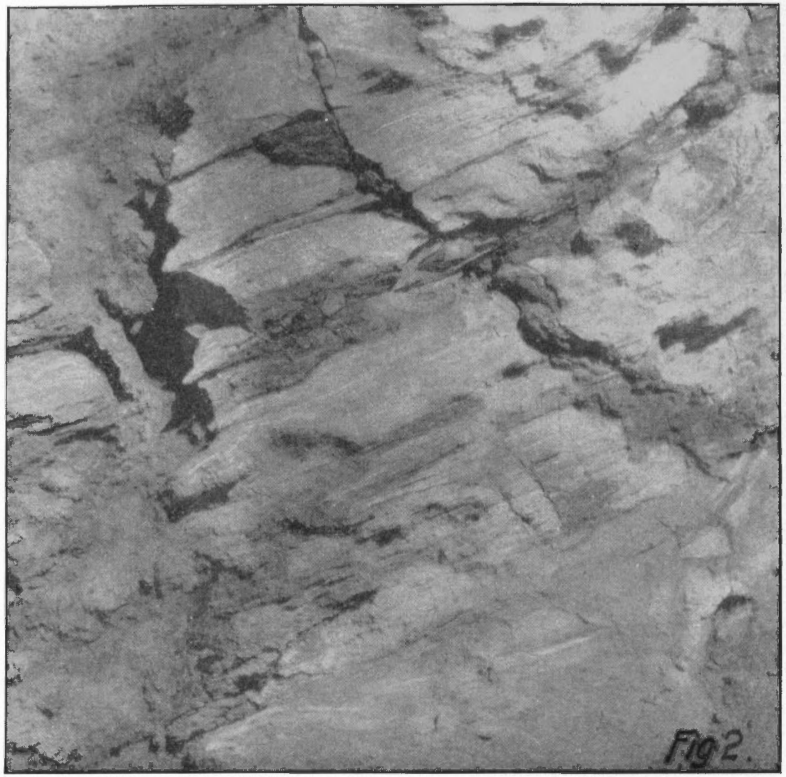
BLUE BELL HARBOUR AND PIER.

The building to the right is the compressor and power house. The portal of the main tunnel can be seen immediately to the left of the small house in front.

PLATE X.

PLATE XI.



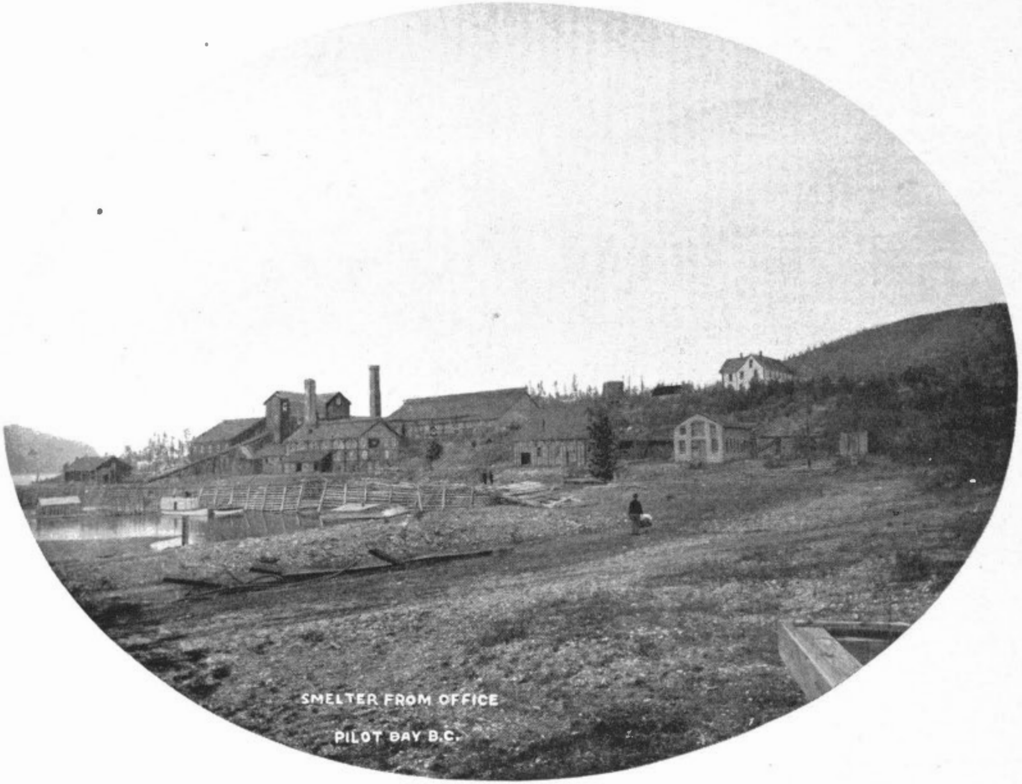


MAIN OPEN CUT BLUE BELL MINE, SHOWING WORKINGS ON FISSURE VEIN AFTER OXIDIZED ORE.

FIG. 1. (A) Bed of pyrites and pyrrhotite with irregular bunches ot zinc blende between fissures "C" and "D."
(B) Oxidized vein matter in a main fissure.
(C) Foot wall of above vein and old drift in carbonate and oxide working.

FIG 2. Showing shattering of the crystalline limestone near a main fissure and filling of the fractures with ore, also replacement of the limestone along liues of bedding.

PLATE XII.



PILOT BAY CONCENTRATOR AND SMELTER.

The crude ore from the Blue Bell was treated at this plant. A large tonnage from the Lucky Jim Mine was also concentrated here.

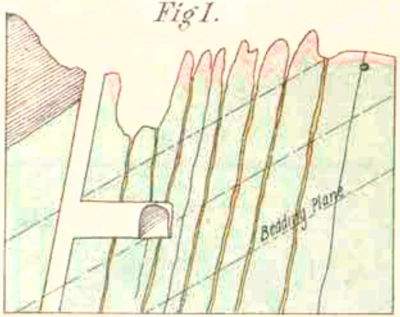
PLATE XIII.

SHEETED ZONE,

Krao Mine,

AINSWORTH. B.C.

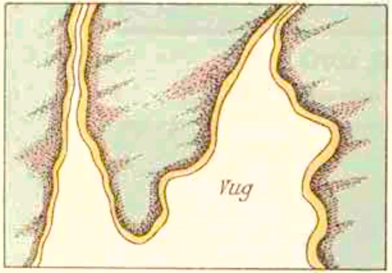
Fig 1.



LEGEND

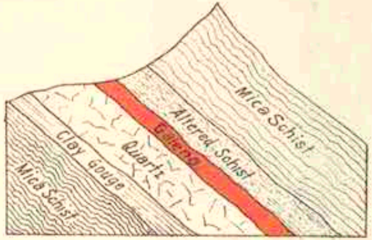
Limestone Schist Galena Blende & Siderite

Fig 2.



ORE PIPE FORMED ALONG FISSURE

Fig 3.



THE SPOKANE VEIN

Philip Argall. Denver Colo.

PLATE XIV.

Fig 1. Cork Mine.

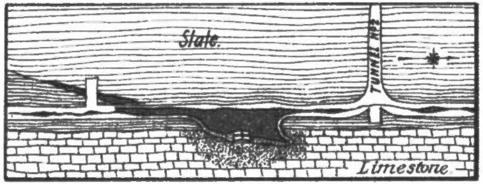


Fig. 2.

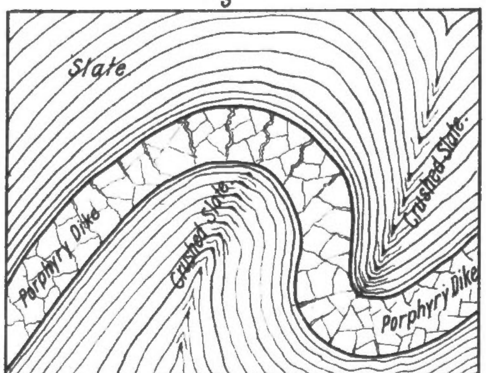
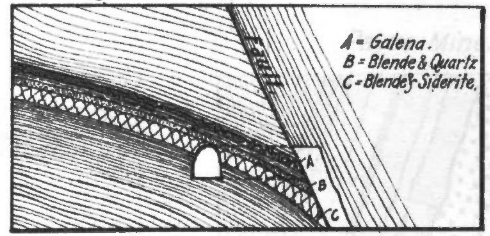
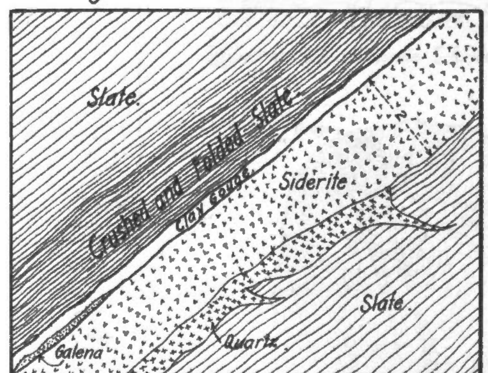


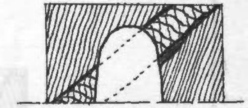
Fig. 4 Jackson Mine.



Cross Section.

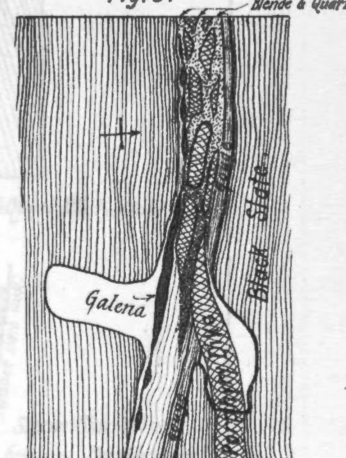
Fig. 3. Whitewater Deep Vein.





Cross Section.

Fig. 5.

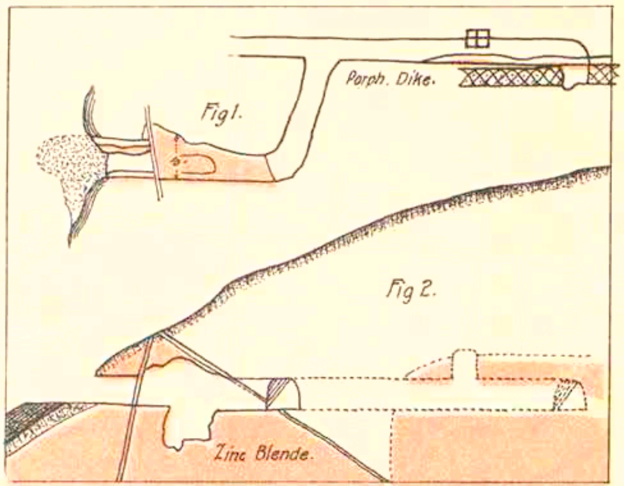


Diagrammatic sketch of Vein fissure forming along Dike.

Jackson Mine.

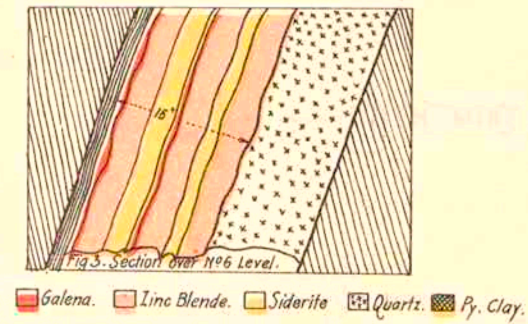
Philip Argall. Denver Colo.

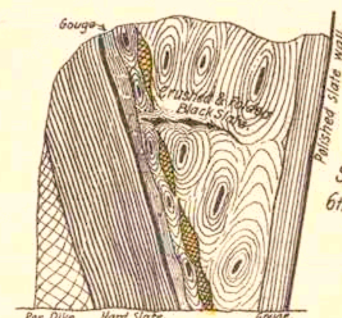
PLATE XV.



Sketch of No 1 Tunnel, Bell Mine.

Payne Mine.



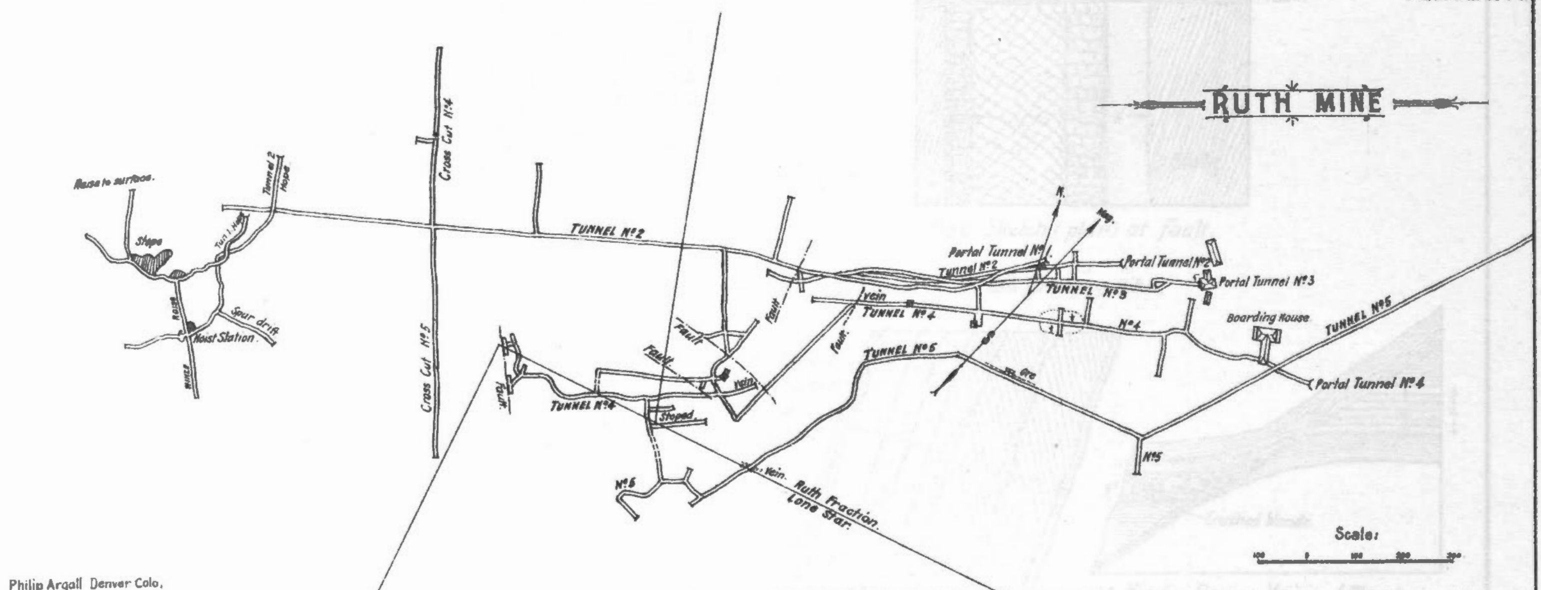


Slocan Star Vein.
6th Level. North branch.

Illustrating a barren vein.

Philip Argall, Denver Colo.

PLATE XVI



Philip Argall Denver Colo.

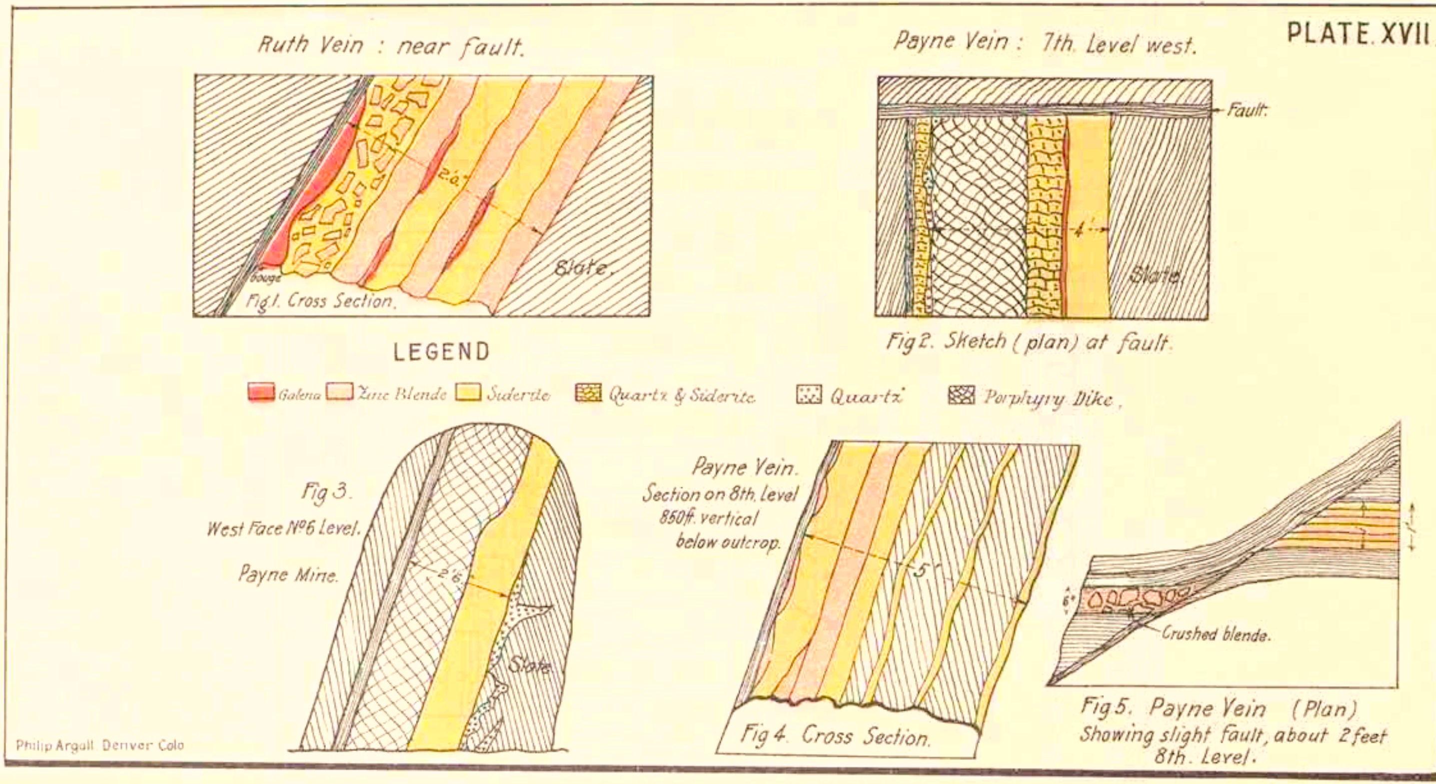


PLATE. XVII.
Ruth Vein : near fault.
2'6"
Slate.
Gouge
Fig. 1. Cross Section.
Payne Vein : 7th. Level west.
Fault.
4'
Slate.
Fig 2. Sketch (plan) at fault.
LEGEND
Galena
Zinc Blende
Siderite
Quartz & Siderite
Quartz
Porphyry Dike.
Fig 3.
West Face No. 6 Level.
Payne Mine.
2'6"
Slate.
Payne Vein.
Section on 8th. Level
850ft. vertical
below outcrop.
5'
Fig 4. Cross Section.
1'
6"
Crushed blende.
Fig 5. Payne Vein (Plan)
Showing slight fault, about 2 feet
8th. Level.
Philip Argall. Denver Colo.

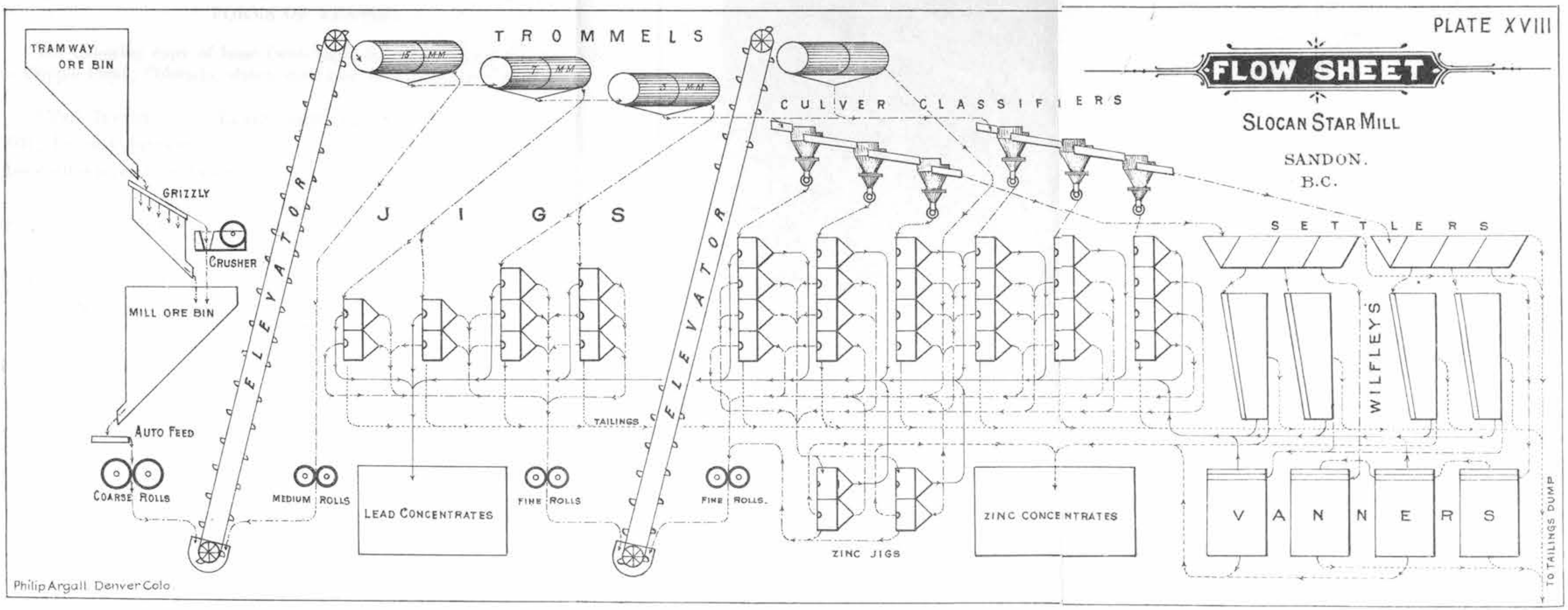


PLATE XVIII
FLOW SHEET
SLOCAN STAR MILL
SANDON.
B.C.
TRAMWAY ORE BIN
GRIZZLY
CRUSHER
MILL ORE BIN
AUTO FEED
COARSE ROLLS
ELEVATOR
TROMMELS
15 MM
7 MM
3 MM
JIGS
MEDIUM ROLLS
LEAD CONCENTRATES
FINE ROLLS
TAILINGS
ELEVATOR
FINE ROLLS.
CULVER CLASSIFIERS
SETTLERS
WILFLEYS
ZINC JIGS
ZINC CONCENTRATES
VANNERS
TO TAILINGS DUMP
Philip Argall Denver Colo.

The lessees, however, cannot be left entirely to their own devices. They should be bound to do a certain amount of drifting, raising, etc., each month; to employ steadily a certain number of men and to keep the workings in their leased ground securely timbered and free from loose rock, etc. Lastly, the owner or his agent should see to it that the lessees live fully up to the covenants of their leases, for it must be admitted that many mines are ruined, by incompetent miners and irresponsible lessees, over whom proper supervision has not been exercised.

FORMS OF LEASES.

The following copy of lease (with right to purchase) recently in force in Cripple Creek, Colorado, shows the usual covenants and terms:—

"THIS INDENTURE OF LEASE, made this 25th day of November, A.D. 1901, by and between..............................of Denver, Colorado, (hereinafter termed the lessor), party of the first part, and.............................. ....of..............................(hereinafter termed the lessee) party of the second part;

WHEREAS, the lessor is the owner of the Ada Bell lode mining claim, United States Patent Survey No. 8700, situate in the Cripple Creek Mining District, Teller County, Colorado, saving and excepting those portions of said claim which were in conflict with the Stella Girl and Mountain Goat lode mining claims, Patent Survey No. 10308, which said lessor has heretofore conveyed to the Comanche Plume Gold Mining Company under a decree of the District Court of Teller County, Colorado, in a suit wherein the Comanche Plume Gold Mining Company was plaintiff, and James McClurg, *et al*, were defendants, which the lessee desires to lease for mining purposes;

Now, this agreement, WITNESSETH, that the said lessor for and in consideration of the covenants, agreements, payments and royalties herein reserved and by the said lessee to be kept, paid and performed, has granted, demised and let, and by these presents does grant, demise and let unto the said lessee, all of the right, title and interest of the said lessor in and to the said Ada Bell lode mining claim, United States Patent Survey No. 8700, with the machinery and shaft houses, ore houses and all buildings appertaining to the operation and working of the mine, if any;

TO HAVE AND TO HOLD the same from the date hereof until the first day of December, A.D. 1903, at 12 o'clock noon, unless sooner forfeited or terminated by election of the lessee herein provided, or through violation of any covenant or condition herein required to be kept and performed, or through default in any payment herein agreed to be made by the said lessee.

In consideration thereof, the said lessee covenants and agrees with the said lessor as follows:

1. To enter upon said premises on or before December, 15th, 1901, and work the same in mine-fashion and in manner necessary and appropriate to good and economical mining so as to develope the property and to extract ore with due regard to the development of the same as a workable and developing mine.

2. To perform labor upon said premises in the working and development thereof not less than 200 shifts of eight hours each during each period of ninety days during the term hereof after commencement of work. It is understood and agreed that the term "shift" as here used so far as relates to sinking, drifting, cross-cutting and stoping and all similar work refers to and is intended to mean hand work and that each machine drill at work shall be deemed equivalent to six shifts of hand work.

3. To well and sufficiently timber with strong, well fitted and durable timber all the workings on the premises hereby leased where said lessee shall be or shall have been working at all points where proper in accordance with good mining and to promptly repair and replace all timbering rendered insufficient by shock of blasting, caving in, pressure of water, wear and tear or other cause. To keep the timbering and shafts at all times in good repair and serviceable condition.

4. To allow the lessor, its officers, agents and representatives at any and all times to enter upon and visit all parts of said premises and workings therein for the purpose of inspecting the workings and machinery and taking assay samples and to allow said lessor, his manager or agent to take average samples at any and all times from any ores awaiting shipment as he or they may see fit and to allow the lessor's surveyors and assistants to make survey of the workings whether under or above the ground or both at least once in each calendar month, if the lessor or manager or agent shall so order.

5. To cause or permit no building or structure of any kind to be erected or remain on said premises, except such as are necessary for the actual workings of the mine or for the storing, hoisting, sorting or shipping of ores extracted therefrom, except and in so far as authorized in writing by said lessor.

6. To ship all ores extracted from said premises to such smelter, mill, sampler or reduction works as shall be mutually agreed upon by the lessor and lessee. The lessor shall have a lien upon all ore which has been extracted at the plant, mine or in transit to or in possession of smelter, mill, sampler or reduction works, for royalties at the rate herein specified.

7. To pay or to deliver to the lessor as rental of the demised premises, royalties at the rate of 20 per cent. upon all ores extracted, shipped and sold from said premises during the preceding calendar month, to be computed on the net sampler, mill, smelter or reduction works returns of said ores remaining after deducting the cost of hauling, freight and treatment thereof only. All such royalties to be paid by the sampler, mill, smelter or reduction works to the lessor at the time of each settlement on duplicate returns.

8. The said lessee further covenants and agrees to settle, pay and discharge on or before the 15th of each month, all indebtedness and liabilities incurred by him during the preceding month for work performed, services rendered or materials furnished in respect to said leased premises and to furnish evidence of such payment to said lessor whenever requested by him within thirty days after such request.

9. To make all working shafts at least 4 x 8 feet in the clear and all cross-cuts at least $3\frac{1}{2}$ x $6\frac{1}{2}$ feet in the clear.

10. To keep at all times all shafts, tunnels and other workings thoroughly drained and clear of loose rock and rubbish, unless prevented by extraordinary mining casualty and not to obstruct the main openings in any manner whatever. To store no waste in drifts, except with the consent of the said lessor, or his agents, privilege being given to fill all stopes with waste.

11. To do no underhand stoping and to refrain from stoping in said premises within ten feet of any working shaft, without stulling and filling so as to insure the full and complete safety of the shaft.

12. To mine, sort and select all ore taken from said premises in such a manner as will secure the best results therefrom. That all ores mined that are too low grade for profitable shipment during the life of this leaseare to remain the property of and subject to the control and disposition of said lessor.

13. The lessee hereby assumes all responsibility in case of accident to himself or his assigns or any employee in or upon the demised premises.

14. It is mutually agreed and understood that all dumps, shaft houses, buildings and other improvements, except machinery erected or placed upon said premises by the lessee shall be and remain the property of said lessor at the expiration or other termination of this lease, the said lessee, however, having sixty days after the expiration or termination of this lease, to remove said machinery.

15. The lessee agrees not to assign this lease and not to sub-let said premises or any part thereof without the written consent of said lessor.

16. It is also agreed that any failure of said lessee to work said premises caused by water, gas or extraordinary mining casualty or by strike or labour troubles or by any litigation or injunction which arises out of or which restrains the exercise of any of the rights or the enjoyment of any of the privileges expressly demised herein, shall not cause a forfeiture of this lease; provided, however, reasonable means shall be exercised by the lessee to expel such gas and to properly ventilate said mines, and to keep the workings drained and free from accumulations of gas or water; provided further, that litigation involving other property or other rights than those expressly demised hereby, shall not be included in the foregoing exception.

17. It is further agreed that said lessee may operate leases on other properties through the shafts and levels of the said Ada Bell claim, including the right to dump on the said Ada Bell claim, after first obtaining the written consent of lessor, and shall have the right to work the said Ada Bell claim through the shafts and levels of the Moose mining claim.

18. If the said lessee shall desire to surrender this lease before the expiration thereof, he shall have the right to do so without liability for a failure to carry on the same for the full term, provided that he shall give twenty days written notice to the lessor declaring his intention to surrender, and during this twenty days he shall continue work, as above stated, unless prevented by some of the causes heretofore mentioned.

19. That upon the violation of any covenant or covenants hereinabove reserved, and notice in writing thereof delivered to the agent, manager or foreman, or any person in charge of the property for said lessee, such violation shall cease forthwith; and unless the covenant or covenants so violated shall be made good, and the effects of such violation repaired, and the covenants of the lessee be fully complied with, within thirty (30) days after a written notice is sent by registered mail to the aforesaid lessee, at his address in.............................., it shall, at the option of the lessor, work a forfeiture of this lease and bond, and all rights that may accrue thereunder.

20. The said lessor further agrees to sell and convey the demised premises to the said lessee, his heirs, assigns or nominees by good and sufficient mining deed at any time prior to 12 o'clock noon on the first day of December, A.D. 1903, for the sum of One Hundred Thousand Dollars ($100,000) cash in hand paid to said lessor, or deposited to the credit of said lessor in the First National Bank of Denver, Colorado; and said lessor further agrees that within thirty days after the said lessee, his heirs or assigns shall leave with the cashier of said Bank, a request in writing addressed to the said lessor that the said deed be executed and deposited with said Bank, the said lessor shall execute said deed and deposit the same together with abstract of title to said premises in said Bank subject to inspection and with directions to deliver said deed to the said lessee, his heirs, assigns or nominees upon the deposit of said sum of money to the credit of the said lessor within the time herein limited; provided, however, that if at any time after the date hereof the said lease shall be forfeited under the provisions of paragraph 19 hereof, or by virtue of a failure to comply with any of the provisions aforesaid, or if it shall be otherwise terminated, then and in that event the said option whereby said lessee, his nominees or assigns are given the right to purchase said property within said time for said sum of money, shall absolutely cease and determine at the same time with said lease.

And time is expressly made of the essence of this agreement.

This agreement and every covenant therein contained, shall be binding upon and enure to the benefit of the heirs, assigns and legal representatives or the respective parties hereto.

IN WITNESS WHEREOF, said parties have hereunto set their hands and seals on the day and year first hereinbefore written.

................................(SEAL).

................................(SEAL).

The following form may be said to represent the average Leadville lease:—

"THIS AGREEMENT OF LEASE, made this third day of July, in the year of our Lord one thousand nine hundred and one, between ....; ...., of the city of Denver, State of Colorado, lessor, and..of Leadville, Colorado, Lessee or tenant,

WITNESSETH, That the said lessor, for and in consideration of the royalties, covenants and agreements hereinafter reserved and by the said lessee to be paid, kept and performed, has granted, demised and let, and by these presents does grant,demise and let unto the said lessee, all his right, title and interest in and to the Montgomery lode mining claim, Mineral Entry 222, California Mining District, County of Lake and State of Colorado.

To have and to hold unto the said lessee for the term of five (5) years from and after the first of April, 1901, expiring at noon on the first day of April, A. D. 1906, unless sooner forfeited or determined through the violation of any covenant hereinafter against the said tenant reserved.

And in consideration of such demise, the said lessee does covenant and agree to and with the said lessor as follows, to-wit:—

To enter upon said mine and premises and work the same in mine fashion, in manner necessary to good and economical mining, so as to take out the greatest amount of ore possible, with due regard to the development and preservation of the same as a workable mine, and to the special covenants hereinafter reserved.

To work and mine said premises as aforesaid, steadily and continuously from April 1st, 1901; and any failure to work said premises to the extent of opening by drifts or shafts three hundred (300) feet during any one year, may be considered a violation of this covenant, and at the option of the lessor work a forfeiture of this lease.

To well and sufficiently timber said mine at all points where proper in accordance with good mining, and to repair all old timbering in the accessible parts of the mine whenever it may become necessary.

To allow said lessor and his agents from time to time to enter upon and into all parts of said mine for the purpose of inspection.

To keep the drifts, shafts, levels and other workings which are now open and clear of water, rock and rubbish, in the same order and condition at all times during the term hereby demised.

To deliver the said lessor as royalty for the use of the said demised premises, the following percentages of the net returns from all ores mined and sold from said premises: On all ores netting Ten Dollars ($10) and under per ton, ten (10) per cent. of the net smelter returns therefrom; and on all ores netting over Ten Dollars ($10) and under Fifteen Dollars ($15) per ton, fifteen (15) per cent. of the net smelter returns therefrom; and on all ores netting over Fifteen Dollars ($15) and under Twenty Dollars ($20) per ton, twenty per cent. (20 %) of the net smelter returns therefrom; and on all ores netting over Twenty Dollars ($20) per ton, twenty-five (25) per cent. of the net smelter returns therefrom. It is agreed, however, by and between the parties hereto, that in case silver should reach a market price of eighty cents (80c.) per ounce, the royalty to be paid by the said lessee to the lessor upon all ores netting Twenty-five Dollars ($25) and over per ton, shall be thirty (30) per cent. of the net smelter returns therefrom.

To sink the Montgomery shaft through the white limestone, to develop the lower horizon, with all reasonable dispatch.

To deliver to said lessor said premises, with the appurtenances and all improvements, with all drifts, shafts, tunnels and other passages drained and clear of loose rock and rubbish, and the mine ready for immediate continued working, and in as good order and condition as when received by him, (accidents not arising from negligence alone excusing) without demand or further notice, on the said first day of April, A.D. 1906, at noon, or upon the sooner determination of this lease for the violation of any of the covenants hereinafter set forth.

AND FINALLY, That upon the violation of the covenant or covenants hereinbefore reserved, the term of this lease shall at the option of the said lessor, expire, and the same and said premises shall be and become forfeited to said lessor; and the said lessor and his agents may thereupon, after demand of possession in writing, enter upon said premises and dispossess all persons occupying the same, with or without process of law; or, at the option of the said lessor, the said tenant and all persons found in possession may be proceeded against as guilty of unlawful detainer.

The said lessor expressly reserves to himself the property and right of property in all minerals to be extracted from said premises during the term of this lease, and all shipments and settlements shall be made in his name.

Each and every clause and covenant of this agreement of lease shall extend to and bind the executors and assigns of the said lessor, and the executors, administrators and assigns of said lessee.

IN WITNESS WHEREOF, The parties hereto have hereunto set their hands and seals, this third day of July, 1901.

..............................(SEAL).

..............................(SEAL).

ORE RESERVES.

Coming now to the ore reserves and output of zinc ore from the Slocan, the largest amount, partially blocked out, "being developed", in any one mine is 60,000 tons of zinc blende, concentrating ore, probably equivalent to 25,000 tons of shipping ore of 48 to 50% in zinc. Owing to the lack of development in the majority of the mines examined any general estimate of "ore developed" or "ore being developed" would be very misleading, while the "ore expectant" under these conditions it is impossible even to guess at.

In the mines I examined, a careful comparison shows that the zinc blende will average 2½ times the weight of the galena present in the veins; figuring on the higher saving of galena in the concentrating mills, one might reasonably expect, when arrangements are completed to save the zinc blende in all the ores milled, that the output of zinc blende will be double that of the galena; this I believe will be a safe figure and as the mills are improved and better saving effected, the zinc blende output should approach 2¼ times that of the silver-lead in any given year.

The lead output depends on several matters: the permanency of the veins; whether the mines are worked throughout the year or only in the summer months; or if the present half-hearted mining methods are to give way to a vigourous leasing system.

As regards the veins, we have seen that they carry pay ore to great depths and have been but partially worked, for silver-lead ore only; that with lead and zinc as profitable minerals, the value of the veins is greatly enhanced.

Lastly, in such a densely wooded country, it is very doubtful if one-half of the pay veins have yet been discovered; consequently it is not too much to expect that the Slocan veins can be worked at a profit for at least twenty years to come, more especially if cheaper methods of mining are adopted and mills erected that will effect a better saving of values.

The second reason I have given for the present mining torpidity in the Slocan; lack of proper facilities for making a close saving of values, I shall enlarge upon in the following chapter on milling.

ORE MILLING IN THE SLOCAN.

The flow sheets accompanying this report clearly show the general concentration methods in use in the Slocan mills, and also that with perhaps one exception, the mills are of the same general type, viz.: simple lead ore concentrators, wherein efficiency in the saving of mineral gives way to the efforts to attain large capacity, cheapness of operation and small labour bill. These efforts would all be admirable in principle and also in practice, providing a higher percentage of saving would not warrant the increased cost necessary to secure the higher recovery of mineral.

I am satisfied that considerable improvement can be made in the existing mills and a much higher mineral recovery obtained in the process, leading to greater profit in milling. Of twelve mills visited, but three were in operation. Two of these were running on mine dumps; the other on very low grade mine rock. Consequently I had no opportunity of studying the mills operating on the average silver-lead ore of the Slocan. My conclusions are therefore reached by deductive reasoning, coupled with previous experience in the handling and treatment of high grade silver-lead ores. I may point out, however, that none of the mills is equipped with either sampling or weighing machines, to determine the amount and assay value of the ores entering the plant, hence any close discussion of the actual saving made is entirely out of the question; because reliable data, on which one can alone base such a discussion, cannot be obtained.

I discussed in a general way the matter of saving the valuable mineral with several persons interested, and found the general belief to be that 75% to 85% of the lead is recovered in the present practice; from 55% to 65% of the zinc and 65% to 75% of the silver in the mill feed. The lead saving includes the assay value of the lead in the zinc concentrates. The zinc saving includes the zinc retained in the lead ore concentrate and sold as lead ore. The silver recovery includes necessarily the silver contained in both the lead and the zinc concentrates. These figures are for the lead and zinc, I believe, approximately correct, but owing to the impossibility of procuring complete shipping returns, even for statistical purposes, I am unable to determine the ratio of zinc and lead production in order to compute the net zinc and the net lead saving *per se*. I have, however, grave doubts of the silver saving reaching 70% on the average.

To my mind the almost entire absence of picking belts, or tables, is one of the worst features in the Slocan mills. I strongly advise that the ore from the breakers should be washed in a drum, or for that matter in a screen, and divided into say three products, two sizes for the picking belts and the finer for direct milling in the ordinary manner. For effective ore sorting or picking, sizing is necessary; hence two picking belts should be provided, one for the coarser product, the other for the finer, but washing is of still greater importance. The ore should be freed from clay and spread on the belt in a perfectly clean condition; then the minerals are easily recognized and the proper selections can be made.

Ore sorting is usually assumed to be expensive, and where ore is turned over by hand, it is undoubtedly an expensive operation; but, as part of a milling process in which the ore is automatically conveyed slowly on wide belts from one machine to another, washed free of mud and dust, and brought directly under the ore sorter's eye, it is by no means expensive. In fact there are many places where it pays to pick out the rock and gangue from the ore, and thus increase the capacity of the concentrator, as well as raise the quality of its output. To some it may appear foolish to attempt to sort out lead ore when there is a mill at hand where it can be concentrated for a few cents per ton (seldom less than 50c. per ton, however.) This alleged foolishness is perhaps worth illustrating, to the end that a proper comparison can be made in any given case.

Taking the silver-lead ore only, with an assumed saving by concentration of 75% of the lead and silver in the ore, it is manifest that for each ton of silver-lead ore that goes through the mill, at best but three-quarters of a ton is recovered as concentrate, whereas every ton of silver-lead ore picked out before milling represents 100% of its value, less of course the increased cost of the picking, or sorting, if any.

For the purpose of an illustration, let us take what I assume to be a very low grade mill ore for the Slocan.

(A) It is assumed that the net value of one ton of silver-lead ore is $80.

(B) That only 2½% of this silver-lead ore can be picked out by hand from 100 tons of mill feed.

(C) That the capacity of the mill is 10 tons per hour, of which only six tons passes over the picking belts; these latter would require the services of two ore sorters.

Neglecting the silver-lead mineral in the fine ore, we need confine our attention only to the 2½ tons that can be picked out by hand, *i.e.*, 2½ tons at $80=$200. If crushed and jigged, 75% of this can be recovered in concentrates, or $150; while by employing two men, together with a washing trommel and picking belts, we can recover 2½ tons, worth $80 per ton or a total of $200, at a cost of $6 for labour and $1.50 for power, repairs, etc., (a total expense of $7.50), showing $42.50 in favour of hand picking, which figured back on the crude ore amounts to $0.425 per ton.

The same process and remarks would apply to zinc blende ores, though of course to a less degree on account of the difference in value between silver-lead ores and zinc ores. I will go further and state, *that in my opinion waste rock and worthless* gangue, when over 1 in. in diameter, can in Slocan ores be sorted out of the mill feed *cheaper than it can be milled*, that is, crushed to ¼ inch or less, jigged or passed over the tables and slime plant in the mill. Then, by picking out waste rock the milling ore is partly concentrated, which results in a richer feed to the jigs, etc., and a higher all-around mineral recovery is obtained. Furthermore waste rock can be picked out for 20 to 30 cents per ton, according to conditions, whereas if left in the mill feed it will cost 40 to 70c per ton to mill it.

The colour contrast between the black slate and the vein rock, and between siderite and blende or galena, renders sorting, after the rock and ores are washed clean from mud and clay, a very easy, simple operation.

For hand sorting, the mill feed should not be broken too fine, e.g., if it be passed through a breaker, reducing it to about 2 inch cube, the crushed ore should next pass to a washing drum or a screen, in which it could be washed clean with a spray of water; next divided by screening into three parts, (A) that between 2 inches and 1¼ inch, which should pass at once to a picking belt; (B) the portion between 1¼ inch and ¾ inch, which should be delivered to a second picking belt; (C) the portion finer than ¾inch, which together with the wash water, should pass on to the rolls and jigs for the complete milling process.

On the picking belts, galena, zinc blende and waste rock would be severally picked out by two or more sorters at each belt, as the nature of the feed or necessities of the case indicated. The coarse mixed ores (A) discharged from the end of the conveyor belt might be returned to a roughing roll, and when crushed, again, join the ores in the washing drums, and in part pass over the second belt with the portion (B) and in part pass on to the mill with (C).

In the chapter on "Rocks and Ores" it is shown that the high silver content of the galena is due to the soft and brittle freibergite disseminated through the lead sulphide. So long as the freibergite remains associated with the galena in the dressing process, it can be saved, but once separated its low specific gravity (4.5 to 5.0) almost certainly points to its loss, or at best to its only partial recovery with the zinc concentrate, providing it remains granulated, but if it be slimed (as it must be to a large extent, when crushed) it necessarily means an almost total loss of this very high grade silver mineral, containing over 4,000 oz. of silver per ton, and no doubt in some cases 8,000 oz. of silver per ton.

For these reasons I am led to believe that the loss of silver in milling in the Slocan is much greater than is generally supposed. The remedy, however, is simple, and will be found in close hand sorting after coarse breaking in the manner indicated above.

The mills in the Slocan are, as previously stated, simple lead-ore concentrators, designed to run through a large tonnage with a small force of men. In one instance the superintendent of a mine evidently considered his practice superb, inasmuch as he pointed out with great satisfaction that two men ran the whole mill. This caused me to reflect and endeavour to discover the efficiency of his work.

It will be noticed from the flow sheets that two compartment jigs are very common, four compartment jigs scarce, while there are no five compartment jigs in the district. Fine crushing machinery is, with the exception of the Slocan Star mill, invariably lacking. Even the new mill at Brockman has only two sets of rolls for an assumed capacity of five tons per hour.

The problem to solve is the separation of galena from zinc blende, siderite, etc. This with a five compartment jig, might work out as follows:

First compartment: Best galena concentrate.

Second compartment: Mixed galena-blende, for retreatment, regrinding in part.

Third compartment: Zinc-siderite product (finished).

Fourth compartment: Zinc-siderite middling, for regrinding and retreatment, in part.

Fifth compartment: A lighter zinc-siderite middling for regrinding.

With the present three-compartment jigs, the best that can be done is to save along with the galena the purer (heavier) blende, together with the heavier siderite, allowing most of the blende-siderite middling to escape in the mill tailings.

It is manifest that if close saving of middling products is attempted, fine grinding machinery must be added and also more tables and vanners, or other slime machinery, to take care of the increased proportion of fine ore. This matter must be worked up gradually to the point where the cost of dressing and fine grinding approaches the value of the product derived from the combined operation. Fine grinding machinery is, however, the first requisite to increase mineral extraction along this line of procedure. With the introduction of modern tables of the Wilfley type, a water classified product is no longer necessary for good concentration. In fact a screened (sized) product usually gives better results on such tables. Pointed boxes and other forms of hydraulic classifiers have one great drawback: they dilute the ore pulp enormously, necessitating immense settling tanks to thicken the fine pulp before sending it to the tables or vanners; and even then the effluent water invariably carries off considerable valuable mineral, more particularly in the treatment of silver ores.

Screens can now be obtained that will work efficiently to at least 0.005 inches aperture. Consequently screen-sized products can be had for both jigs and tables, and the wasteful hydraulic classifier eliminated.

Going almost directly from Old and New Mexico to British Columbia, from places where water is scarce and difficult to obtain for dressing operations, to a moist and well watered land, I was positively astounded at the amount of water used in ore dressing in British Columbia and at the great dilution of the ore pulp. The indiscriminate crushing of slate, porphyry, galena and 4,000 to 8,000 ozs. silver ore (freibergite), literally in a veritable torrent of water, gave me a distinct shock. On reflection I decided that while the Mexicans were undoubtedly struggling with the irreducible minimum, the British Columbia mill men had exceeded a reasonable maximum of water supply, or ore dilution, and that probably the better practice lay near the minimum.

To summarize: I believe the milling practice in the Slocan can be materially improved, the saving of silver and lead very much increased, and the zinc blende saving vastly increased by putting into successful operation the foregoing suggestions, which I tabulate as follows:

First: The proper sorting (picking) of the ores before milling.

Second: Increase the number of jig compartments to save one, or more products of zinc-siderite middlings.

Third: Install fine crushing machinery to treat the middling product.

Fourth: Abolish hydraulic classifiers and use screens for all products, down to 100-mesh.

Fifth: By abolishing the classifiers and economizing on water, the pulp dilution should be less than one half the present practice.

Lastly: Should the leasing of small mines, or portions of the principal mines, become general, as I hope it may, one or more custom mills to purchase small lots of ore from the lessees, or to treat such ore at a fixed price per ton, would be desirable. Such a mill centrally located on the railway could even be operated by the mine owners, or on a co-operative basis, with or without a magnetic separation plant, to put the finishing touches on the zinc concentrates.

A custom mill should have a complete sampling plant so that the crude ore could be paid for on its metal contents, when weighed and sampled, less, of course, the proper treatment charge to cover the loss in dressing and the milling cost, plus a reasonable profit on the transaction. Or the custom mill could be hired by the day, the lessee in that case, attending to the concentration of his ore and receiving concentrated products for shipment.

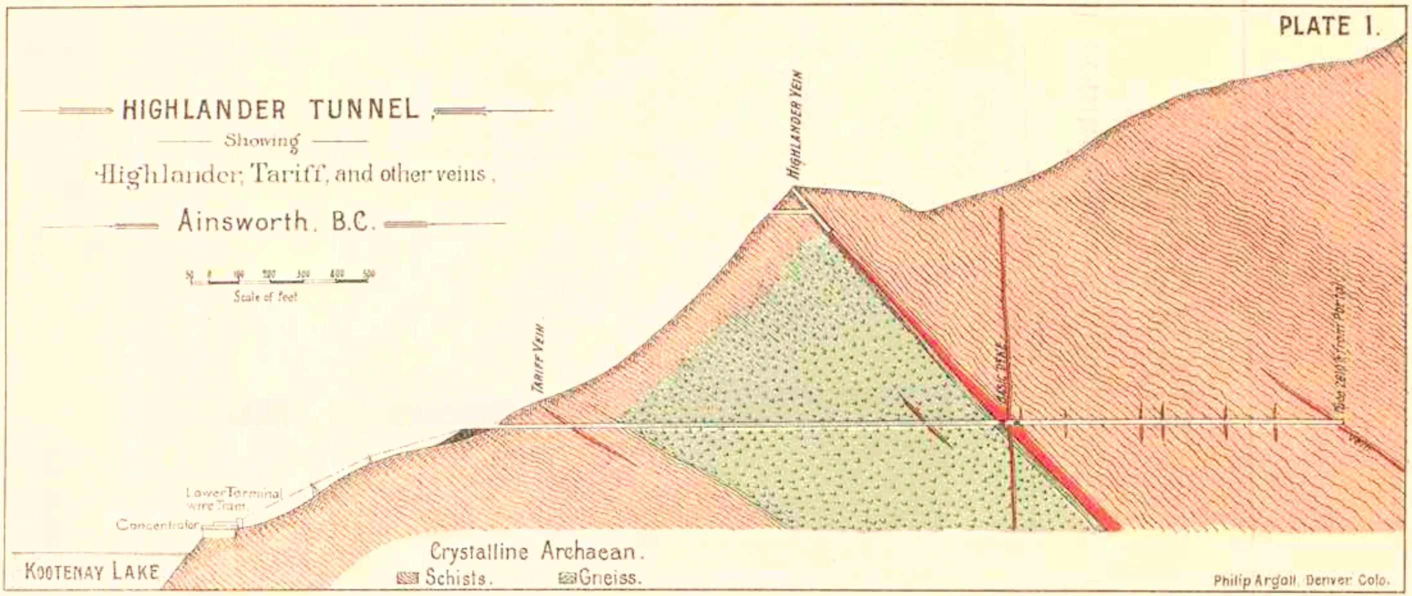


PLATE I.
HIGHLANDER TUNNEL,
Showing
Highlander, Tariff, and other veins,
Ainsworth, B.C.
50 0 100 200 300 400 500
Scale of feet
HIGHLANDER VEIN
TARIFF VEIN
Lower Terminal wire Tram.
Concentrator
KOOTENAY LAKE
Crystalline Archaean.
Schists.
Gneiss.
Philip Argall, Denver, Colo.

To summarize: I believe the milling practice in the Slocan can be materially i[illegible] much increased, and the zinc blende [illegible] successful operation the foregoing [illegible]

First: The [illegible] before milling.

Second: [illegible] to save one, or more product[illegible]

Third: Install [illegible] the middling product.

Fourth: [illegible] screens for all products, down to 100-m[illegible]

Fifth: [illegible] economising on water, the pulp dilution [illegible] present practice.

Lastly: [illegible] at portions of the principal mines, become [illegible] one or more custom mills to purchase small lo[illegible] treat such ore at a fixed price per ton, [illegible] centrally located on the railway could even be [illegible], or on a co-operative basis, with or [illegible] to put the finishing touches on the zinc [illegible]

A custom mill [illegible] sampling plant so that the crude ore could be paid [illegible] when weighed and sampled, less of course, [illegible] charge to cover the loss in dressing and the milling [illegible] profit on the transaction. Or the [illegible] the losses in that case, attending to the [illegible] and shipping concentrated products for shipment.

PLATE 2

DIAGRAM

Showing composition of Highlander Vein, on line of Tunnel.

Philip Argall. Denver, Col.

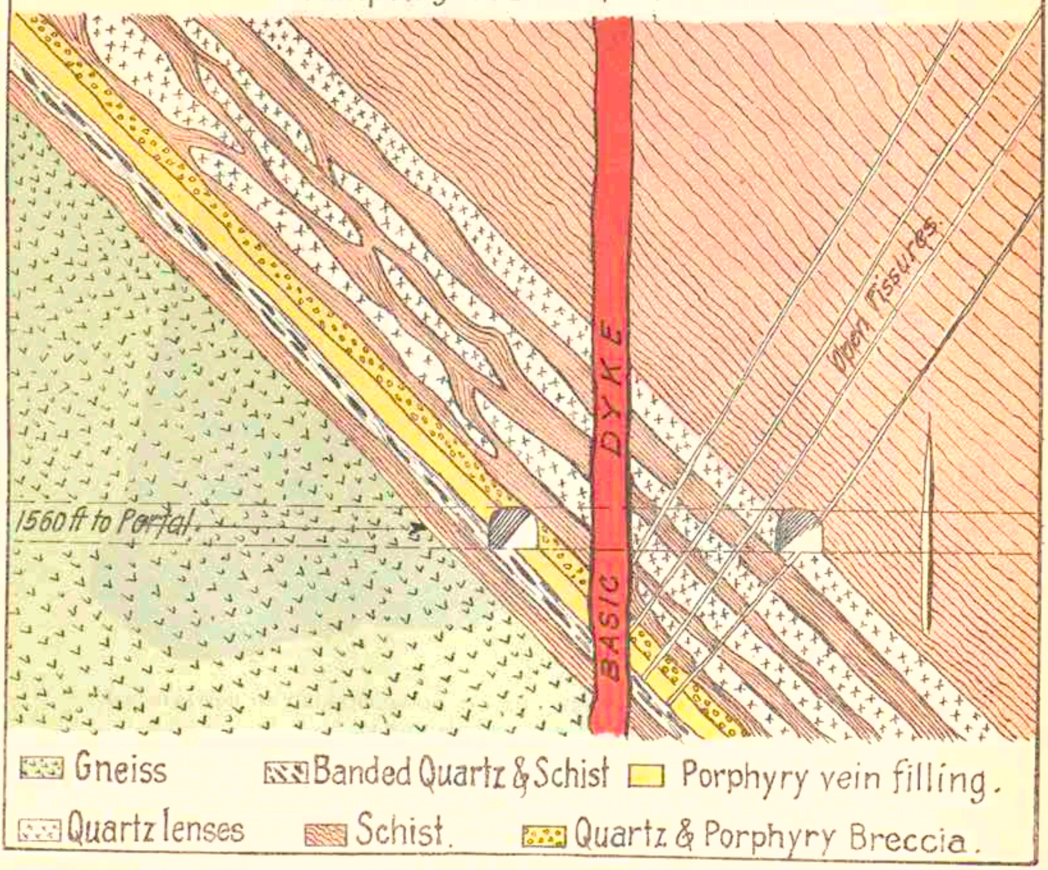


PLATE III.

ILLUSTRATION OF CRUSTIFICATION AND THE ORDER OF MINERAL DEPOSITION IN THE HIGHLANDER VEIN.

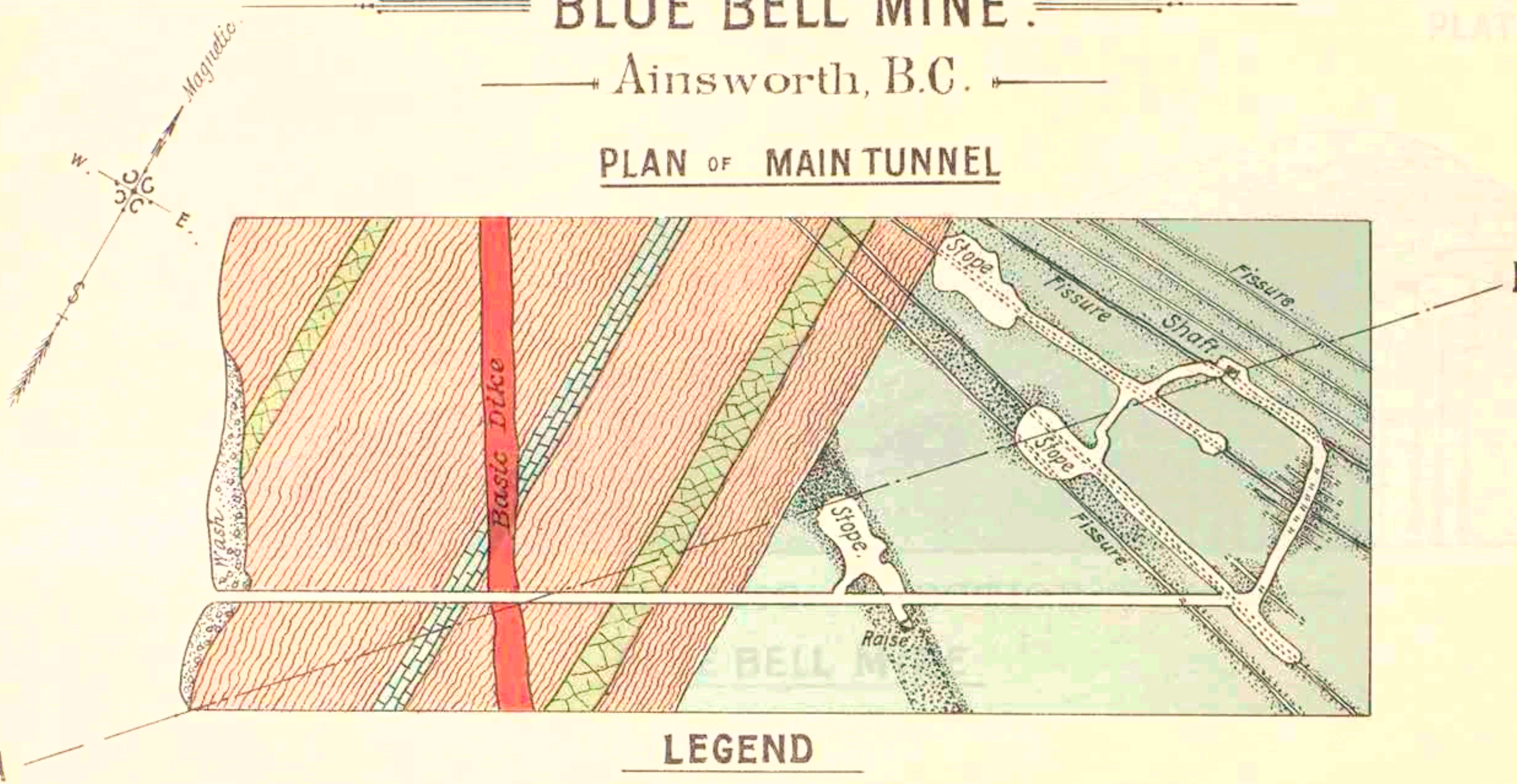
(P) Porphyry fragment cubical in shape, covered with a film of siderite ; next an envelope of zinc blende ¼ inch in thickness surrounds the cube like piece of porphyry, following which is a coating of siderite ⅛ inch thick, and which is in turn surrounded by a thin film of zinc blende forming the outer layer, coincident with the fragment above it.

PLATE IV

BLUE BELL MINE.

Ainsworth, B.C.

PLAN OF MAIN TUNNEL



LEGEND

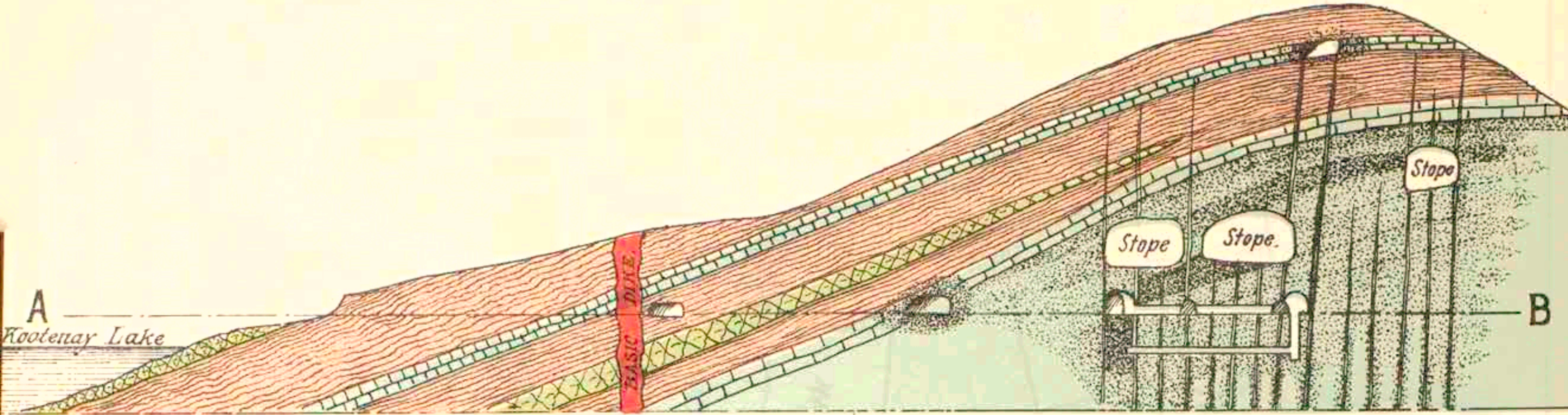
Quartzite, Muscovite Schist, Crystalline limestone Ore.

50 0 100 200

Scale in Feet.

Philip Argall. Denver, Colo.

PLATE V



DIAGRAMMATIC SECTION

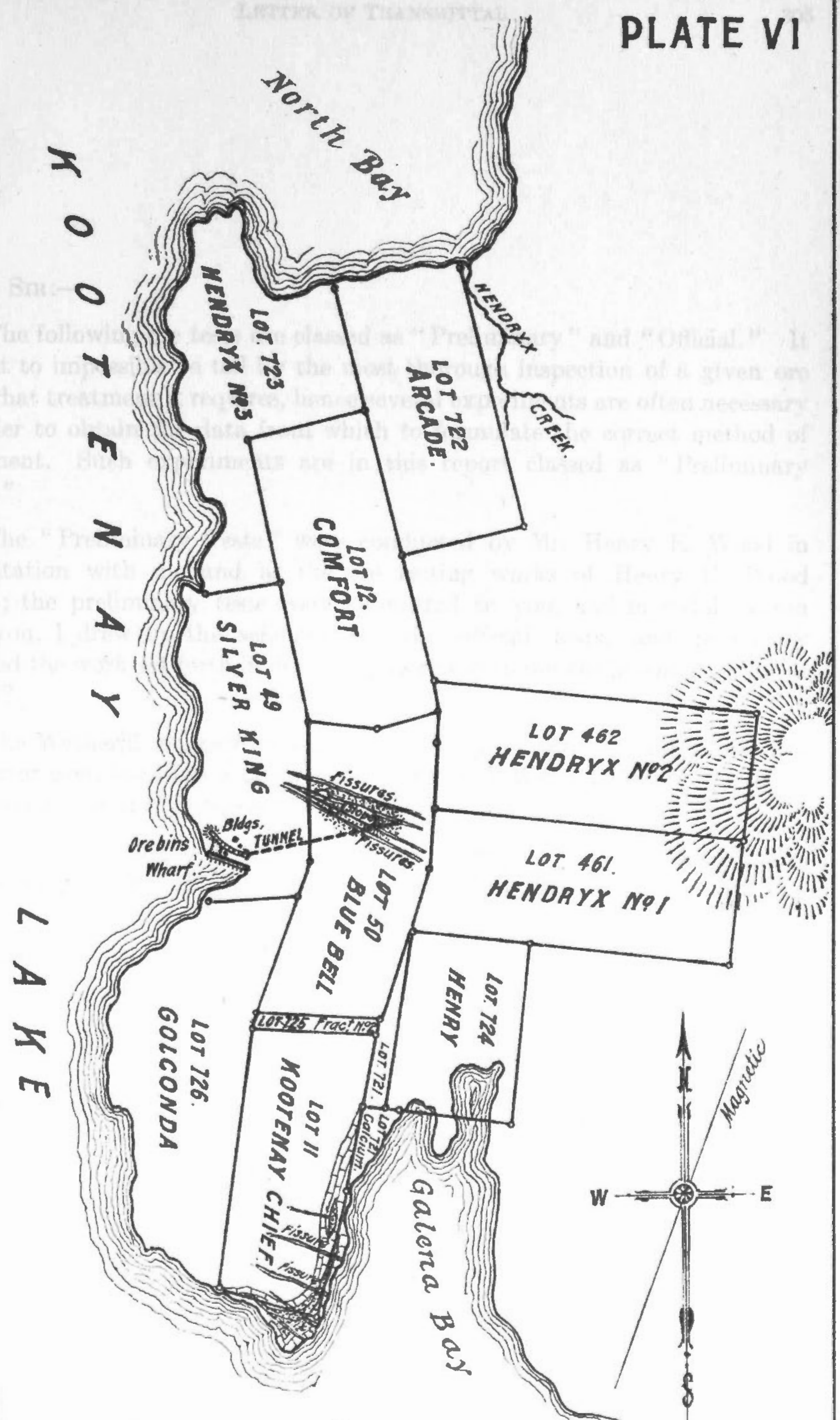
BLUE BELL MINE

Philip Argall, Denver Colo.

LEGEND

Quartzite, Muscovite Schist, Crystalline limestone Ore.

PLATE VI



BLUE BELL PENINSULA.

Ainsworth. B.C.

Scale:

100 0 400 800

Philip Argall. Denver Colo.

LEGEND

DEAR SIR:—

The following ore tests are classed as "Preliminary" and "Official." It is next to impossible to tell by the most thorough inspection of a given ore just what treatment it requires, hence several experiments are often necessary in order to obtain the data from which to formulate the correct method of treatment. Such experiments are in this report classed as "Preliminary Tests."

The "Preliminary Tests" were conducted by Mr. Henry E. Wood in consultation with me and at the ore testing works of Henry E. Wood & Co.; the preliminary tests were submitted to you, and in collaboration with you, I drew up the schemes for the official tests, and personally directed the work set forth in detail in this report under the heading "Official Tests."

The Wetherill magnetic separator, the Dings separator and the Blake Separator were installed in the works of Henry E. Wood & Co.; the International separator at the Denver Ore Testing Works.

All the assaying and chemical determinations embodied in this report, are the work of Mr. Henry E. Wood, who had direct charge of all the ore testing.

For convenience in comparison the results have been uniformly reduced to tons, 100 tons corresponding to 100 units.

Yours respectfully,

WALTER RENTON INGALLS ESQ.,
Chief Commissioner

PHILIP ARGALL.

REPORT ON ORE TESTS.

LOT NO. 1—JACKSON MINE.

Concentrates from the ores of the Jackson mine, Jackson Basin. Containing zinc blende, siderite, pyrites and some porphyry gangue. The concentrates all passed a 6-mesh screen, 0.10 in. aperture, and gave the following screen analysis:

+	8-mesh	26% by weight.	0.077 inch aperture.
+	12-mesh	32% by weight.	0.050 inch aperture.
+	20-mesh	6% by weight.	0.030 inch aperture.
—	20-mesh	36% by weight.	0.030 inch aperture.

It assayed gold 0.02 ozs., silver 5.00 ozs., lead 1.4% and zinc 35%.

Test A in the preliminary examination of this ore was conducted on the unsized concentrate, as received from the mill, by passing it through the Wetherill magnetic separator, to remove the siderite and thus increase the percentage of zinc contained in the ore; the result with a current of 120 volts and 2.10 amperes to excite the magnets being

69.1 tons blende	@ 47.1% zinc	=65,092.2 lbs. zinc, saving	93.19%
30.9 tons siderite	@ 7.7% zinc	= 4,758.6 lbs. zinc, loss	6.81%
100.0 tons	@ 34.92% zinc	=69,850.8 lbs. zinc.	

The efficiency of the process, or the percentage of the total zinc recovered as marketable ore, is thus seen to be 93.19%, and the zinc lost in the siderite 6.81%. The recovery is therefore high, but the percentage of zinc in the blende is not quite satisfactory. The best results are perhaps obtained from an ore carrying not less than 50% of metallic zinc. An examination of the blende showed numerous pieces of attached siderite, indicating that finer crushing was desirable.

Test B.—The ore was next crushed to pass a 20-mesh sieve, 0.03 in. aperture, and passed through a Wetherill magnetic separator, using a current of 120 volts and 2.30 amperes, giving the following separation:—

68.7 tons blende	@ 49.10% zinc	=67,463.4 lbs. zinc, saving	93.50%
31.3 tons siderite	@ 7.50% zinc	= 4,695.0 lbs. zinc, loss	6.50%
100.0 tons	@ 36.08%	=72,158.4 lbs. zinc.	

which appears to be quite satisfactory, both as to the saving effected and the tenor of the zinc product.

Siderite, or iron carbonate, becomes magnetic when exposed for a short time to a dull red heat (from 15 to 25 minutes in a hot muffle). Advantage can be taken of this fact to effect separation of the blende and the iron by a low intensity machine, cheaper both in first cost and in operation, than the high intensity machines. The siderite in the Slocan zinc concentrates averages about 60% $FeCO_3$, 27% $MnCO_3$, and 13% $MgCO_3$ and $CaCO_3$ combined.

The specific gravity of the siderite varies from 3.7 to 3.9, and that of the zinc blende from 3.9 to 4.1; these minerals are therefore not separable by any process of gravity concentration. Consequently the mixed concentrates must be treated in magnetic separating machines of either high or low power. For the low intensity machines the ore must first be subjected to a light roast, which changes the siderite wholly or in part to magnetic oxide, which, for experimental purposes can be picked out from the zinc by an ordinary horse shoe magnet. We have previously seen that it required a current of 120 volts and 2.30 amperes to lift the raw siderite out of the blende in the Wetherill tests on this lot, but with the roasted ore 0.40 amperes sufficed. The roasting also concentrates the ore somewhat by driving off part of the carbon dioxide and sulphur in the siderite and pyrites. The latter mineral is also in part changed into magnetic sulphide, removable by the magnets, hence a higher grade of zinc can be obtained by roasting, than by treating it raw on the high intensity machines.

Test C.—Ore crushed to pass a screen aperture of 0.03 in. and roasted. The loss in weight during roasting was 16.66%. The roasted ore was then passed through a Wetherill machine at 120 volts, 0.40 amperes, giving:—

63.75 tons blende	@ 52.7% zinc	= 67,192.5 lbs. zinc, saving	86.87%
36.25 tons siderite	@ 14.0% zinc	= 10,150.0 lbs. zinc, loss	13.13%
100.00 tons	@ 38.67%	= 77,342.5 lbs. zinc.	

The above figuring is of course on the zinc in 100 parts of roasted ore.

There is some loss in all these operations; hence I prefer taking the computed assay value in each case. The dust loss also tends to concentrate the ore and thus the computed values increase slightly with each test; for example the assay value of Lot No. 1 was 35% zinc:

The first computed value was 34.92% zinc.
The second computed value was 36.08% zinc.

The computed value, however, being based on the weights and assay values of the magnetic and non-magnetic ores secured from the machines, must necessarily be more accurate, inasmuch as two assays are used instead of one; while the dust loss is usually a fraction of one per cent. The assay value of the crude ore or concentrates is, however, given in each instance, but the percentage of recovery and loss are, for the reasons given, and for clearness and simplicity, figured on the computed assay.

Test D.—A portion of this ore was next reduced to pass a screen with 0.042 in. aperture, for the purpose of competitive test on the Wetherill and on the International machines. The results on the former were satisfactory, but the representative of the International machine objected to the dust produced by crushing the concentrates, claiming that he could do better work on the sized concentrates without crushing. His objection was allowed, but the test was conducted as the previous ones on this lot as an experiment or preliminary test.

International machine. Armature 90 r.p.m. Current 110 volts, 4 amperes. Ore passed twice through the machine:

66.34 tons of blende	@ 47.1% zinc	= 62,492.3 lbs. zinc, saving	86.65%
33.66 tons of siderite	@ 14.3% zinc	= 9,627.7 lbs. zinc, loss	13.35%
100.00 tons	@ 36.06%	= 72,120.0 lbs. zinc.	

Test E.—Official test on Wetherill magnetic separator. Lot 1, Jackson mine concentrates, screened to pass 0.042 in. screen aperture and the oversize crushed to pass the same opening (12-mesh). Current 120 volts, 2.20 amperes. Two pole machine, each pole taking off a magnetic product. The feed was unsized, all the dust remaining in the ore. Result:

66.62 tons of blende	@ 49.10% zinc	= 65,420.8 lbs. zinc, saving	91.59%
33.38 tons of siderite	@ 9.00% zinc	= 6,008.4 lbs. zinc, loss	8.41%
100.00 tons	@ 35.71%	= 71,429.2 lbs. zinc.	

Test F.—Official. The same ore was sized, giving the following screen analysis and passed over the International machine.

+0.03 in. screen aperture	45%
—0.03 in. to dust	40%
Dust	15%
	100%

The dust assayed 33.9% zinc and was not treated on the International machine. The result on the material treated separately in the above two sizes, and mixing the two magnetic portions together to form one product, and the non-magnetic portions for the other product, thus combining the separate tests on different sized products, to form one competitive test resulted as follows:

Revolutions of armature	90 to 95 per minute.
Current	110 volts and 2.4 amperes.

The non-magnetic zinc from the first pass through the machine was given two further passes to free it from the siderite, making three passes in all. Result:

69.4 tons of blende	@ 49.7%	= 68,983.6 lbs. zinc, saving	93.06%
30.6 tons of siderite	@ 8.4%	= 5,140.8 lbs. zinc, loss	6.92%
100.0 tons	@ 37.06%	= 74,124.4 lbs. zinc.	

Test G.—Official. The zinc value as computed in the feed is raised, on account of the removal of the dust, to 37% zinc; hence another test on the sized but uncrushed concentrates was decided upon. A screen analysis of the crude concentrates gave:

+	0.042 in. screen aperture............	54.8%
+	0.013 in. screen aperture............	34.1%
—	0.013 in. screen aperture............	1.1%

Current 110 volts, 2.5 amperes, 95 revolutions of armature per minute for the two first sizes and 90 revolutions of the armature at 4.5 amperes for the –40 product.

Each size of product was passed three times through the machine, and the products combined for one general average. Result:

71.4 tons blende	@ 47.90%=68.401.2 lbs. zinc,	saving	91.56%
28.6 tons siderite	@ 11.00%= 6,292.0 lbs. zinc,	loss	8.44%
100.00 tons	@ 37.35%=74,693.2 lbs. zinc.		

Test H.—Official test on roasted ore. The ore was first charged into a muffle and roasted at a dark red heat until there was a strong smell of sulphur. The heat was then continued five minutes and the charge drawn; the whole operation occupying 20 minutes; for 10 minutes the ore was kept at a dark red. The loss in weight amounted to 13.8%, or 116 tons of raw ore gave 100 tons of roasted ore. The roasted ore was then passed through a Dings magnetic separating machine with a current of 120 volts and 0.35 amperes exciting the magnets. The zinc product was given a second pass at 0.40 amperes. The magnetic product was also re-passed to separate the zinc entangled in and carried off by the cloud of magnetic material and the zinc so obtained added to the first zinc product. Result:

66.09 tons blende	@ 50.8% =67,147.44 lbs. zinc,	saving	84.83%
33.91 tons siderite	@ 17.7% =12,004.14 lbs. zinc,	loss	15.17%
100.00 tons	@ 39.57%=79,151.58 lbs. zinc.		

The 33.91% of magnetic material was next crushed to pass 0.02 in. screen aperture, to free the particles of zinc blende included in or attached to the roasted siderite. Result:

23% of zinc blende assaying........50.4% zinc.
77% of siderite assaying........... 8.6% zinc.

Combining these results we have for total products and recovery the following:

73.89 tons of blende	@ 50.75%=74,998.35 lbs. zinc,	saving	94.37%
26.11 tons of siderite	@ 8.60%= 4,490.92 lbs. zinc,	loss	5.63%
100.00 tons	@ 39.75%=79,489.27 lbs. zinc.		

This is a very high recovery of zinc from medium grade ore, and is obtained by the simple method of crushing the coarse magnetic product recovered in the first pass through the machine, which amounts in this case to one-third of the weight of the ore. The process consists of roasting the ore as it comes from the concentrator, passing it through a low intensity machine, producing marketable zinc ore and highly magnetic iron, which is crushed to pass a screen of 0.02 in. aperture, and re-treated by the low intensity magnetic separator, by giving a further quantity of marketable zinc blende and an iron product which will be valuable as a smelting ore, when rich enough in silver and lead. Both the silver and the lead are low in the original concentrates and also in the various products; the following assays, show the distribution of these metals:—

	Silver	Lead	Zinc
1. Original concentrates	5.00 ozs.	1.40%	35.0%
2. First zinc (roasted ore)	7.20 ozs.	2.40%	50.8%
3. Second zinc (roasted ore)	5.80 ozs.	1.90%	50.4%
4. First magnetic (roasted ore)	4.80 ozs	1.90%	17.7%
5. Second magnetic (roasted ore)	4.40 ozs.	1.80%	8.6%

These various products are, as we have previously seen, reduced to 73.89 tons of 50.75% zinc ore and 26.11 tons of magnetic material assaying as per No. 5 in the table above.

These tests show two feasible methods of treating the Jackson ores, viz., in the raw state by high intensity machines or in the roasted state by low intensity machines. Further comparisons between these machines will be made in summing up the conclusions of the ore tests; here it is necessary, only to point out that if zinc ore exceeding 50% metal is required, the roasting process must prevail.

LOT NO. 2—RUTH MINE.

Concentrates from the ores of the Ruth mine, made in the Ruth mill, Sandon, B.C.

Test A. — The concentrate consisted of zinc blende, siderite, pyrites and quartz. It assayed: gold 0.02 ozs., silver 9.4 ozs., lead 1.4%, zinc 37%. It was crushed to 20-mesh, 0.03 in. aperture, and passed in the raw state over the Wetherill separator. Current 120 volts, 1.85 amperes.

73.69 tons blende	@ 47.8%=70,447.64 lbs. zinc,	saving	93.50%
26.31 tons siderite	@ 10.0%= 5,262 lbs. zinc,	loss	6.95%
100.00 tons	@ 38.35%=75,709.64 lbs. zinc.		

The zinc recovery is quite high, but the tenor of the zinc blende is not perhaps high enough to give the best commercial results, and furthermore it carries 10 ozs. of silver per ton.

The next test was made on a Blake electrostatic separator to throw out the lead and it was hoped a large portion of the silver; then treating the zinc product from the Blake machine in a Wetherill to remove the siderite as in Test (A).

Test B.—The lead product repelled by the Blake separator assayed 14.6 ozs. silver and 4.40% lead. The siderite 5 ozs. silver and 1.20% lead. The zinc 10.20 ozs. silver and 1.10% lead. The last two products are of course from the re-treatment of the zinc product of the Blake machine on the Wetherill separator. Figuring the zinc in the usual manner, we find the following:—

68.25 tons blende	@ 49.6 %=67,704.0 lbs. zinc,	saving	87.65%
25.40 tons siderite	@ 11.6 %= 5,892.8 lbs. zinc,	loss	7.63%
6.35 tons lead	@ 28.7 %= 3,644.9 lbs. zinc,	loss	4.72%
100.00 tons	@ 38.62%=77,241.7 lbs. zinc.		

The lead product from the Blake separator contains only 4.40% lead, with 28.7% zinc, and is practically an unsalable product. Consequently this test is unsatisfactory.

Test C.—A portion of the ore was next roasted in the muffle for a few minutes; the roasted product assaying 40.4% zinc was treated on the Wetherill machine at 120 volts, 0.30 amperes, to remove the highly magnetic material.

Result:

64.50 tons of blende	@ 53.30 %=68,757 lbs. zinc,	saving	85.07%
35.50 tons of siderite	@ 17.00 %=12,070 lbs. zinc,	loss	14.93%
100.00 tons	@ 40.41%=80,827 lbs. zinc.		

The ore lost 12.28% of its weight in roasting, or 1.14 tons of raw gave one ton of roasted ore. It will be noted that the zinc recovered is but 85.07% of that present, but that it assays 53.3% metal. The complete assays on the two separated products are as follows:—

Zinc blende, silver 10 ozs.	Lead 0.10%	Zinc 53.30%
Siderite , silver 10 ozs.	Lead 2.20%	Zinc 17.00%

The silver it will be seen is equally divided as to value between the zinc blende and the siderite, while the lead favours the siderite.

Test D.—The raw concentrate was next crushed to pass 0.04 in. screen aperture and passed through the International machine with the dust made in crushing. The representative in charge of the machine objected to the dust, just as in the test on Lot No. 1; objection allowed and test not classed official. The feed to the machine assayed 37.59% zinc. Current 110 volts 3.0 amperes, 93 revolutions of armature per minute. Result:—

68.50 tons blende	@ 47.80%=65,486 lbs. zinc,	saving	87.09%
22.58 tons siderite (1st)	@ 13.20%= 5,961.1 lbs. zinc,	loss	7.92%
8.92 tons siderite (2nd)	@ 21.00%= 3,746.4 lbs. zinc		4.99%
100.00 tons	@ 37.59%=75,193.5 lb.s zinc.		

The zinc product obtained in the first pass through the machine was given a second pass, to free it from siderite, producing a 47.8% zinc and a middling product, assaying 21% zinc that should be ground finer and repassed.

Test E.—Official. Wetherill machine. Ore reduced to pass 0.04 in. screen aperture. The same ore in fact was used in test (D) on the International machine. Current 120 volts and 2.3 amperes. One double-pole, Wetherill machine. One pass with the dust in the ore.

71.04 tons blende	@ 48.40%=68,766.7 lbs. zinc, saving 94.96%
28.96 tons siderite	@ 6.30%= 3,648.9 lbs. zinc, loss 5.04%
100.00 tons	@ 36.20%=72,415.6 lbs. zinc.

Test F.—Official. International machine. Made on the concentrates as received from the mill, divided however by screening into the following sizes.

+ 0.07 in. screen aperture	28.1%
+ 0.03 in. screen aperture	40.7%
— 0.03 in. screen aperture	31.2%
	100%

Each size was treated separately on the machine and each zinc product was given two additional passes, making three passes in all for each finished product.

+ 0.07 in. product treated at 75 revolutions, 6 amperes.
+ 0.03 in. product treated at 75 revolutions, 4 amperes.
— 0.03 in. product treated at 90 revolutions, 4 amperes.

The voltage in each instance was 110. The combined products gave the following computed results:—

65.8 tons of blende	@ 49.7% zinc = 65,405.2 lbs. zinc, saving 90.44%
34.2 tons of siderite	@ 10.1% zinc = 6,908.4 lbs. zinc, loss 9.56%
100.0 tons	@ 36.15% zinc = 72,313.6 lbs. zinc.

One test was made on this Lot No. 2 roasted, and passed through the International machine, but owing to the loss of the magnetic product through an accident, it cannot be reported fully. The zinc product, however, assayed 53.7% zinc, showing 93% recovery.

This Ruth ore having almost the same mineral composition as the Jackson ore, further tests were not considered necessary, more especially as one general roasting test will later be reported on a mixture of lots 1, 2 and 3, which are typical of the Slocan blende-siderite ores of medium silver value.

LOT NO. 3—PAYNE MINE.

Concentrates from the Payne mill, Sandon, produced from the Payne mine dumps. The concentrates assayed gold 0.02 oz., silver 18 oz., lead 4.9%, zinc 42%.

This ore contained nearly 5% lead; hence an effort was made to throw out the lead on the Blake electrostatic machine, after removing the siderite by treatment in the Wetherill machine. The concentrates were first crushed to pass 0.03 in. screen aperture. The siderite removed by the Wetherill at 2.10 amperes, amounted to 12.70%, assaying, silver 5.20 ozs., lead, 1.60%, zinc 5.40%. The non-magnetic product at 2.10 amperes, was then passed over the Blake machine, with unsatisfactory results; the portion of the ore that should have contained the bulk of the lead amounted to 7.04%, assaying, lead 4.10%, while the zinc product amounting to 80.26% of the ore assayed 4.20% lead. The assays on the two products were as follows:—

First:	Silver, 25.4 ozs.	Lead, 4.10%	Zinc, 46.20%	Gold, 0.06 ozs.
Second:	Silver, 19.0 ozs.	Lead, 4.20%	Zinc, 49.10%	

From which it will be observed that there is a slight concentration of the silver in the first product and of the zinc in the second, but neither on a commercial basis. I therefore combine the products to obtain the result of the blende-siderite separation in the first instance.

Test A.—Wetherill separation at 120 volts, 2.10 amperes.

80.26 tons blende	@ 49.10% zinc	= 78,815.32 lbs. zinc	saving	98.42%
7.04 tons blende	@ 46.20% zinc	= 6,504.96 lbs. zinc		
12.70 tons siderite	@ 5.40% zinc	= 1,371.60 lbs. zinc,	loss	1.58%
100.00 tons	@ 43.35% zinc	= 86,691.88 lbs. zinc.		

The zinc saving is, of course, high on account of the small quantity of siderite in the ore. The combined zinc product, however, assays only 48.8% zinc, and contains 96% of the silver in the ore.

Test B.—A portion of the ore was then roasted at a dull red heat and subjected to the Wetherill separator with a current of 120 volts and the following amperes:—

	Silver	Lead	Zinc
At 0.30 amperes, 13.45% assaying	21.40 ozs.	5.00%	8.30%
At 0.90 amperes, 7.24% assaying	32.20 ozs.	10.90%	30.80%
At 4.00 amperes, 67.47% assaying	17.00 ozs.	3.10%	53.00%
Non-magnetic 10.84% assaying	22.00 ozs.	10.80%	48.50%

The 13.45% by weight removed at 0.30 amperes would make a satisfactory lead smelting product, containing 21 oz. silver and 5% lead; the product at 0.90 amperes carries too much zinc for smelting and not enough for a zinc ore. By crushing this product to pass a 0.02 in. screen aperture the zinc can be liberated as shown in Lot No. 1, Test *H*, and a high zinc recovery attained. A good treatment for this ore is, however, a proper roast, then take off one product at not exceeding, 0.9 amperes, crush this magnetic product and separate the released zinc blende. A stronger roast would doubtless leave the lead in such condition that most of it would pass off with the highly magnetic product, leaving the zinc blende with less than 5% lead.

Test C.—Official. Wetherill machine. The ore was reduced to pass a screen with 0.04 in. aperture and passed over a Wetherill bi-polar magnetic separator, with a current of 2.50 amperes, 120 volts.

88.06 tons blende	@ 47.10%	= 82,952.5 lbs. zinc,	saving	98.9%
11.94 tons siderite	@ 3.30%	= 911.2 lbs. zinc,	loss	1.1%
100.00 tons	@ 41.93%	= 83,863.7 lbs. zinc.		

Test D.—Official. The above products were next thoroughly mixed together once more, and passed over the International machine; speed of armature 100 r.p.m., current 2.5 amperes on first pass and 3.0 amperes on second pass, 110 volts; the dust resulting from the crushing remained in the ore, which gave the following results:—

90.1 tons blende	@ 47.50%=85,695 lbs. zinc, saving	95.85%
6.6 tons siderite	@ 14.00%= 1,848 lbs. zinc, loss	2.07%
3.3 tons middlings	@ 28.20%= 1,861 lbs. zinc,	2.07%
100.0 tons	@ 44.70%=89,404 lbs. zinc.	

Test E.—Official. International machine. The same ore was again mixed thoroughly together and the dust taken out. It gave the following screen analysis:—

+0.03 in. aperture	56.40%
—0.03 in. aperture	33.06%
Dust	10.54%
	100.00%

The dust assayed 32.4% zinc and was not passed through the machine.

The zinc product obtained on the first pass through the machine was given two other passes, making three in all, the first two at 2.5 amperes, the third at 3.0 amperes, 110 volts. Armature 95 r.p.m. The results were as follows.

95.5 tons blende	@ 50.3% zinc = 96,073 lbs. zinc, saving	99.14%
4.5 tons siderite	@ 14.2% zinc = 828 lbs. zinc, loss	0.86%
100.0 tons	@ 48.45% zinc=96,901 lbs. zinc.	

The removal of the dust raised the assay value of the crude ore nearly 4%. The ore was fed to the separator in the two sizes indicated by the screening tests, the results combined for the 99.14% saving as above.

LOTS 1, 2 AND 3, MIXED.

It being assumed that Lots 1, 2, and 3 represented the average medium grade blende-siderite ores, equal proportions of each lot were taken and well mixed; the mixture assayed 38.2% zinc, and, was charged, just as it came from the mills into a red hot muffle. A strong smell of sulphur was given off and the ore reached a good red heat at 20 minutes. It was kept at this temperature for five minutes more and was drawn after 25 minutes total treatment in the muffle. The loss of weight in roasting was 5%. The roasted ore was next passed through a Dings magnetic separating machine, 120 volts, 0.35 amperes. The zinc product was re-passed at 0.40 amperes; lastly the magnetic product, was re-passed at 0.30 amperes to separate entrained zinc and the product so obtained was added to the first zinc, and gave the following result:—

Test F.—Official.

77.25 tons blende	@ 52.2% zinc = 80,649.0 lbs. zinc, saving	95.1%
22.75 tons siderite	@ 9.3% zinc = 4,231.5 lbs. zinc, loss	4.9%
100.00 tons	@ 42.44% zinc = 84,880.5 lbs. zinc.	

The 22.75 tons of magnetic material assaying 9.3% zinc was next ground to pass a screen of 0.02 in. aperture and re-treated on the Dings machine in the same manner as the first ore products. Result:—

7.62% of blende assaying..............40.7% zinc
92.34% of siderite assaying............ 7.4% zinc

This second concentrates, combined with the first gives:—

78.98 tons of zinc blende	@ 52.0% zinc = 82,060.2 lbs. zinc, saving	96.3%
21.02 tons of siderite	@ 7.4% zinc = 3,110.9 lbs. zinc, loss	3.7%
100.00 tons	@ 42.58% zinc = 85,171.1 lbs. zinc.	

The crude mixed ores, Lots 1, 2 and 3, assayed 9.60 ozs. silver per ton, 2.6% lead and 38.2% zinc before roasting. The 22.75 tons of roasted siderite first separated assayed: silver 5.0 ozs. per ton, lead 1.70%, zinc 9.30%.

The 77.25 tons of zinc separated from the roasted ore assayed: silver 15.2 ozs. per ton, lead 3.60%.

Test G.—Official. A second lot of this mixed product was roasted for 15 minutes only, in a red hot muffle. It gave the following screen analysis:—

Before roasting			After roasting		
+	0.05 in.	53.8%	46.7%	+	0.05 in.
—	0.05 in.	46.2%	24.6%	+	0.02 in.
			28.7%	—	0.02 in.
		100%	100%		

The siderite decrepitates in roasting, hence the roasted ore is much finer than the raw ore as indicated in the foregoing analysis. The loss in roasting amounted to 8%.

The roasted ore was next passed through a Dings separator at 120 volts and 0.35 amperes, for the following result:—

81.50 tons of blende	@ 50.00% = 81,500 lbs. zinc, saving	95.82%
18.50 tons of siderite	@ 9.60% = 3,552 lbs. zinc, loss	4.18%
100.00 tons	@ 42.52% = 85,052 lbs. zinc.	

This test indicates that the siderite can be rendered magnetic at a much lighter roast than that used in test (*F*), but inasmuch as the pyrites is not changed to the magnetic state, it remains with the blende which in test (*F*) assayed 52.2% zinc with a 25 minute roast against 50% zinc with a 15 minute roast in test (*G*). Either roast will, however, give a good marketable product. A higher zinc recovery could be made in test (*G*) by recrushing the 18.50 ton product assaying 9.6% zinc and re-treating it on the Dings separator as was done in the (*F*) test, but inasmuch as the mixed blende product would thus be lowered to a point below 50% zinc; it was not done. On the whole, therefore, the stronger roast appears preferable.

These three lots consisting of concentrates from the Jackson, Ruth and Payne mines, being fairly representative of the low silver-blende-siderite mill products, received it will be noticed considerable attention, and the results of the various tests show clearly three methods by which the ore can be successfully beneficiated.

(A) Treatment in the raw state, over high intensity machines, of the sized, but un-crushed, concentrates for a 47 to 48% zinc product, with a 93% zinc saving.

(B) Treating the lightly roasted, but un-crushed, concentrates in a low intensity machine for a 50% zinc product and 95% saving.

(C) Roasting the ore to obtain a magnetic pyrites and treating in a low intensity separator, re-crushing the highly magnetic product and re-treating it for a 52% zinc product and 96% recovery.

LOT NO 4—HARTNEY GROUP.

This was crude ore from the Hartney Group, which averaged: silver 22.7 ozs., lead 8.10%, zinc 27%.

Test A.—Preliminary. Crushed to pass a screen of 0.03 in. aperture and then treated on Wetherill magnetic separator to remove the siderite. With 1.85 amperes, 18.21% was removed, assaying silver 3.20 ozs., lead 0.30%, zinc 5.90%, leaving a non-magnetic product amounting to 81.79%, which was passed over a concentrating table to separate the lead and zinc. Result, figured from the original ore:—

8.10% lead ore, assaying, silver 67.5 ozs., lead 49.8%, zinc 16.2%
73.69% zinc ore, assaying, silver 13.4 ozs., lead 2.4%, zinc 34.2%

From which it will be noticed that while the galena is fairly well separated, with a tendency of the silver to follow the lead in part, the zinc separation is bad, and the zinc product carries only 34.2% metal. A second concentration test was decided on, looking toward a higher lead product, and it was hoped higher zinc also.

Test B.—All of the ore was passed through 0.03 in. screen aperture; 58% by weight remained on a screen of 0.008 in. aperture, and 42% passed through. The ore was treated on a concentration table with following results:—

Weight	Gold	Silver	Lead	Zinc
Lead concentrate, 7.30% assaying......	0.05 ozs.	82.6 ozs.	64.2%	7.2%
Residue 90.60% assaying......	......	17.3 ozs.	1.4%	27.8%
Loss 2.10%	......	......	......	......
100.00%				

The residue amounting to 90.60% of the ore was then passed over the Wetherill separator, at 2.10 amperes, 120 volts, to remove the siderite, which in Test A. was taken out before concentration. Result:—

Weight	Silver	Lead	Zinc
11.60% magnetic assaying	4.0 ozs.	0.40%	5.70%
79.00% non-magnetic assaying	20.0 ozs.	4.10%	31.60%
90.60%			

The non-magnetic material at 2.10 amperes, contained only 31.6% zinc, which is again unsatisfactory.

Test C.—A portion of the residue (90.6%) from the concentration test was next subjected to a slight roast. The roasted product was then passed over the Wetherill separator, with the following result:—

Weight		Silver	Lead	Zinc
0.30 amperes	18.48% assaying	7.0 ozs.	2.7%	8.2%
0.70 amperes	8.62% assaying	16.5 ozs.	4.8%	19.3%
4.00 amperes	46.02% assaying	23.6 ozs.	3.3%	41.0%
Non-magnetic	17.48% assaying	16.7 ozs.	4.7%	35.3%

The highest zinc product is scarcely suitable as zinc ore, while it also contains the bulk of the silver. The highly magnetic (iron) products are low in both silver and lead. Consequently the results of this test are quite unsatisfactory.

Test D.—A portion of the raw ore reduced to pass a screen of 0.03 in. aperture was next roasted at low temperature, resulting in an 18.7% loss of weight. The following results were then obtained from the Wetherill machine:—

Weight		Silver	Lead	Zinc
At 0.45 amperes	31.17% assaying	8.5 ozs.	2.4%	12.8%
At 4.00 amperes	54.52% assaying	20.4 ozs.	4.5%	41.6%
Non-magnetic	14.31% assaying	35.8 ozs.	21.8%	31.8%

From which it is evident that if roasting is of benefit it must be a high roast to render the pyrites magnetic.

Test E.—Official. The ore was ground to pass a screen of 0.02 in. aperture, then sized and passed over a concentrating table to separate the galena from the blende. It was noticed that considerable silica passed over the table with the blende and that on a large scale operation this silica could be separated, making three products from one table.

A. Silver-lead—headings.

B. Zinc-blende—middlings.

C. Silicious—tailings (valueless).

The ore assayed: gold, 0.02 ozs.; silver, 23.00 ozs.; lead, 8.4%; zinc, 26.10% and gave the following screen analysis:—

+ 0.008 in.	45.71%
+ 0.0055 in.	15.72%
— 0.0055 in.	38.57%
	100.00%

The ore was then treated at each size on a concentrating table and the various products mixed for a general result of the operation.

	Weight.	Silver	Lead	Zinc
Lead concentrates	11.13% assaying	70.50 ozs.	53.20%	12.20%
Tailings	82.86% assaying	16.40 ozs.	2.30%	28.20%
Loss	6.01%	......	......	......
	100.00%			

The recovery and distribution of the metals after this concentration are as follows:—

	Weight	Silver	Lead	Zinc
Lead concentrates	11.13%	34.17%	70.49%	5.20%
Tailings	82.86%	59.43%	22.68%	89.14%
Loss	6.01%	6.50%	6.83%	5.66%

The lead recovery it will be noticed is quite high, 70.49% of the 8.4% lead in the original ore. The saving of silver in the lead concentrate is not good. However, as both the lead and the zinc are low in the crude ore, the results obtained by concentration are on the whole satisfactory.

The 82.86% tailing product was next sized, then roasted at red heat for 35 minutes, allowed to cool and passed through the Wetherill magnetic machine. The screen analysis was as follows:

Before roasting	After roasting
+ 0.008 =45%	56%
+ 0.0055=18%	22%
— 0.0055=37%	22%

showing a considerable increase in the size of the roasted particles on account of the amount of sulphides and lead product. The loss in roasting was 6%. The Wetherill results, at 120 volts, were as follows:—

	Weight	Silver	Lead	Zinc
0.15 amperes	25.00% assaying	19.00 ozs.	3.60%	9.60%
0.30 amperes	7.60% assaying	16.20 ozs.	2.90%	18.30%
0.45 amperes	14.50% assaying	21.40 ozs.	2.50%	30.10%
4.00 amperes	40.40% assaying	18.60 ozs.	0.40%	44.90%
Non-magnetic	12.50% assaying	12.50 ozs.	0.60%	41.10%

These results figured in percentages show the following distribution of metals in the separated products in 100 tons of roasted ore:—

Tons	Silver	Lead	Zinc
25.00	26.15%	52.34%	7.63%
7.60	6.78%	12.80%	4.42%
14.50	17.09%	21.07%	13.88%
40.40	41.33%	9.39%	57.71%
12.50	8.65%	4.40%	16.36%
100.00	100.00%	100.00%	100.00%

The computed value of the roasted ore being: silver, 18.16 ozs.; lead, 1.719%; zinc, 31.43%.

The product 7.60% by weight assaying 18.3% zinc was next crushed to pass a screen of 0.008 in. aperture and treated on the Wetherill machine at 0.15 amperes. Result:—

50% assaying....................15.50% zinc.
50% assaying....................23.90% zinc.

The product 14.50% by weight was also ground to pass an 0.008 in. screen aperture and treated on the Wetherill at 0.45 ampere with the result:—

69% assaying....................30% zinc
31% assaying....................34% zinc

It is therefore quite apparent that the zinc tenor of the two foregoing products cannot be further reduced on a commercial basis, at least not by magnetic separation, the zinc being evidently intimately associated with the iron.

By referring to the magnetic separation tests it will be seen that there is no object in making any separation above 0.45 amperes; that by mixing the two products obtained at 4.0 amperes, we obtain 86% of the zinc contained in the roasted ore in a product assaying about 44% zinc. This it will be observed is the product non-magnetic at 0.45 amperes. Hence by running the roasted ore through the separator at 0.45 amperes we obtain a magnetic product amounting to 47 tons containing 50% of the silver and 86% of the lead; and 53 tons non-magnetic product assaying 44% zinc. This is the best that can be done with this ore.

LOT NO. 5—SLOCAN STAR MINE.

This is a concentrate, consisting of zinc blende, siderite, pyrites and pyrrhotite, assaying, silver 38.6 oz.; lead 1.80%; zinc 33.80%.

Test A.—Preliminary. The first plan suggested consisted in passing the ore over a Wetherill separator to remove the siderite, then concentrating the remainder of the ore on a table to see what amount of silver could be removed from the blende, in a lead or smelting product. A current of 120 volts 1.85 amperes gave the following result on the Wetherill machine:—

	Weight	Silver	Lead	Zinc
Siderite	30.00% assaying............	10.80 ozs.	0.20%	6.60%
Lead concentrate	2.05% assaying............	86.50 ozs.	35.40%	17.40%
Blende	66.70% assaying............	49.40 ozs.	1.20%	45.60%
Loss	1.25%....................	......	......	......

Showing a metal distribution as follows:—

Tons	Contents	Silver	Lead	Zinc
66.7	blende.............................	85.35%	44.4 %	90.00%
30.0	siderite............................	8.40%	3.33%	5.86%
2.05	lead...............................	4.60%	40.30%	1.05%
1.25	loss...............................	1.65%	11.97%	3.09%
		100.00%	100.00%	100.00%

Mixing the siderite and lead product would give a good smelting ore: 32 tons, containing 13% of the silver, 43.6% of the lead in the original ore. The bulk of the silver, however, 85.3%, remains with the blende, the zinc tenor of which is low, so the method cannot be pronounced satisfactory.

Test B.—Official. The ore was sifted through a 12-mesh screen, 0.05 in. aperture, and the oversize crushed to pass the same screen aperture. It was next passed through a Wetherill separator at 120 volts and 2.15 amperes, but owing to the amount of pyrrhotite in the ore the amperes were reduced to 1.85. Result:—

70.90 tons zinc blende	@ 44.80% zinc	=63,526.0 lbs. zinc,	saving	94.9%
29.10 tons siderite	@ 6.00% zinc	= 3,492.0 lbs. zinc,	loss	5.1%
100.00 tons	@ 33.5 % zinc	=67,018.5 lbs. zinc		

Test C.—Official. These products were then thoroughly mixed and passed through the International machine under the following conditions:—

Speed of armature, 90 revolutions, voltage, 110, amperes, 3.1. Three passes: The result was as follows:—

64.66 tons blende	@ 44.3 %	=57,288.7 lbs. zinc,	saving	82.57%
9.67 tons middlings	@ 24.3 %	= 4,699.6 lbs. zinc,	saving	6.79%
25.67 tons siderite	@ 14.4 %	= 7,392.9 lbs. zinc,	loss	10.64%
100.00 tons	@ 34.69%	=69,381.2 lbs. zinc.		

The tests (B) and (C) show conclusively that high intensity machines cannot raise the ore crushed to pass an 0.05 in. screen aperture above 45% zinc; hence roasting is indicated.

The middling product in test (C) should have been crushed finer and re-passed in the machine, but as the product so obtained would in all probability be no higher in zinc than the first headings, 44%, it was deemed unnecessary to carry the test further.

Test D.—Official. A portion of the concentrates was charged into a red hot muffle and kept at a dull red heat for 30 minutes. Sulphur was given off freely and the ore gave a very strong sulphur fume when being drawn from the muffle.

SIZING TEST.

	Before roasting	After roasting
All passed 8-mesh.	+ 0.05 in. 46.00% + 0.01 in. 20.00% — 0.01 in. 34.00%	40.00% 20.00% 40.00%

Roasting loss 10%. The roasted ore was next passed through the Dings machine, 120 volts and 0.30 amperes, with the following result:—

60.00 tons blende	@ 50.6%=60,720. lbs. zinc, saving	84.14%
40.00 tons iron product	@ 14.3%=11,440. lbs. zinc, loss	15.86%
100.00 tons	36.0%=72,160. lbs. zinc.	

The blende product assayed, silver 46.8 ozs.; lead 2.40%
The iron product assayed, silver 35.5 ozs.; lead 2.70%

The separated products therefore show the following distribution of metals.

	Silver	Lead	Zinc
60 tons containing	66.4%	57.2%	84.14%
40 tons containing	33.6%	42.8%	15.86%

By roasting, a 50.6% zinc product can be obtained carrying 46.8 ozs. silver per ton, or 66.4% of the silver in the roasted ore, and an iron product carrying 35.5 ozs. of silver per ton and accounting for 33.6% of the total silver in the roasted ore. This latter is a first-class smelting ore, carrying a large excess of iron over silica, but the zinc is too high, viz., 14.3%. This, however, can be reduced to about 7% by re-crushing and treating the crushed ore on a low intensity machine, just as in the cases of blende-siderite ore previously treated in these testing operations. The result being so obvious, further demonstration was considered superfluous.

It would be very desirable to separate the silver from the zinc ore so that the miner would be paid for the full value of the silver; tests were therefore made along the line of cyaniding and amalgamation.

Test E.—Official. A portion of the ore was crushed to pass an 0.005 in. screen aperture and agitated for 8 hours in a 0.30% cyanide solution with 10 lbs. of lime per ton of ore. The ore before treating assayed 40.9 oz. silver per ton; after treatment, 32.0 ozs. silver per ton; showing an extraction of 20.05% of the silver.

Test F.—Official. A second agitation test was made with ore ground to the same size as test E., but using a 0.15% KCN solution and mercury. Agitation was continued for four hours. The ore was then washed. It assayed 32.8 ozs. of silver per ton, which was practically the same result as in test E., showing that there is no silver present to remove by plain amalgamation.

LOT NO. 6—MONITOR AND AJAX MINE.

This is a lot of zinc concentrates obtained from running the dumps of the Monitor and Ajax Fraction mine in the new mill at Roseberry. It contains a large amount of pyrites, and some pyrrhotite and siderite, together with a magnetic and a non-magnetic zinc blende, that is to say non-magnetic at 4 amperes. This concentrate also contains some galena and gold, the former in very fine grain and closely associated with the zinc blende. The concentrates as received assayed: silver, 14 ozs.; lead, 3.60%; zinc, 34.0%.

Test A.—Preliminary. Crushed to pass a screen aperture of 0.03 in. and passed over a Wetherill magnetic separator, with a current of 120 volts and amperage as indicated:—

	Weight	Silver	Lead	Zinc
1.90 amperes	5.44% assaying.........	5.50 ozs.	0.30%	2.10%
4.00 amperes	49.00% assaying.........	11.50 ozs	0.90%	45.00%
Non-magnetic	45.56% assaying.........	18.40 ozs.	5.20%	25.50%

This test is unfavourable; although 69.8% of the contained zinc is obtained in the 4-ampere magnetic product, it runs only 45% metal, and the non-magnetic, on account of the resin blende present, runs 25.5% zinc, thus shutting it out as a smelting product.

Test B.—The crushed ore was next given a light roast, the loss in weight being equivalent to a concentration of 1.03 tons into one ton,—about 3%.

The roasted ore was passed over the Wetherill machine, current 120 volts and amperage as indicated.

	Weight	Silver	Lead	Zinc
0.55 amperes	10.64%................	25.50 ozs.	3.20%	8.10%
1.35 amperes	14.45%................	19.50 ozs.	3.50%	16.20%
4.00 amperes	53.71%................	10.60 ozs.	1.10%	46.80%
Non-magnetic	21.20%................	13.50 ozs.	7.60%	28.70%

The zinc product assaying 46.80% zinc accounts for 73.24% of the zinc in the roasted ore, but the test is on the whole unsatisfactory. It is evident, however, that a strong roast is necessary to the successful treatment of this ore, so as to render the pyrites magnetic and removable as a salable product for lead smelting, thus materially raising the tenor of the zinc blende.

Test C.—A portion of the raw ore was next treated on a Blake electrostatic machine, with the following results:—

Weight		Silver	Lead	Zinc
Lead and iron product	20%...............	22.80 ozs.	5.40%	19.20%
Zinc product	80%...............	11.40 ozs.	2.90%	37.20%

This test is also unfavourable, due largely to the fact that the lead is so fine grained and intimately associated with the blende and other minerals. To reduce the ore fine enough to liberate the zinc would mean a high dressing loss, and without very fine crushing no separations can be made.

Test D.—Official. A portion of the ore was sifted through a screen of 0.05 in. aperture, and the coarse product crushed to pass the same mesh, this in order to have a granular product with little dust. It was then passed over a Wetherill separator with a current of 120 volts and 2.15 amperes. Result:—

95 tons blende	@ 35.3% zinc =	67,070 lbs. zinc, saving	99.6%
5 tons siderite	@ 2.4% zinc =	240 lbs. zinc, loss	0.4%
100 tons	@ 33.65% zinc =	67,310 lbs. zinc.	

The above confirms preliminary test (A) and proves conclusively that the ore cannot be successfully treated in the raw state.

Test E.—Official. It was therefore decided to give a strong roast and produce a magnetic pyrites if possible. The ore reduced to pass a screen of 0.05 in. aperture was placed in a red hot muffle and kept at a dull red heat for 35 minutes; the ore when drawn was giving off sulphur gases freely and just beginning to soften and ball. The loss weight in roasting was 7%.

The cooled ore was next passed over the Dings machine at full power of the magnets, 0.50 amperes, but only 2% was removed; this was returned and mixed in with the roasted ore and passed through the Wetherill separator, with current of 120 volts and varying amperes as indicated.

At 0.45 amperes	33.70%, assaying, zinc	10.10%
At 0.90 amperes	7.24%, assaying, zinc	37.10%
At 4.10 amperes	40.84%, assaying, zinc	49.30%
Non-magnetic	18.22%, assaying, zinc	56.60%
	100.00%	

In test (B) with a roast that gave good results on the blende-siderite ores, only 10.64% of the net weight of the roasted ore was taken up at 0.55 amperes, while with the higher roast, in test (E) now under discussion 33.7% was picked up by the magnets with 0.45 amperes. This great change is due to roasting the pyrites up to the magnetic stage as indicated and it proved a complete solution for the treatment of this rather complex ore (concentrate).

Taking first the zinc only, we find it distributed in the various products as follows:—

18.22 tons blende	@ 55.6 %=20,260.64 lbs. zinc,	saving	27.84%
40.84 tons blende	@ 49.3 %=40,268.24 lbs. zinc,	saving	55.38%
7.24 tons blende	@ 37.1 %= 5,372.80 lbs. zinc,	saving	7.42%
33.70 tons iron	@ 10.10%= 6,807.40 lbs. zinc,	loss	9.36%
100.00 tons	36.35%=72,709.08 lbs. zinc.		

Taking the first three products for zinc ore we have 66.3 tons assaying 49.7% zinc, equal to a saving of 90.64% of the zinc present in the ore. The third product, however, should be ground finer and re-treated so as to liberate and remove more of the lead and iron from the zinc ore as demonstrated in other tests. Taking, however, the first two products only, they amount to a saving of 83.22% of the zinc present in a product that assays 51.2% zinc, which is a very satisfactory saving in a complex ore like this one.

The complete assays of the product are as follows:—

	Gold	Silver	Lead	Zinc
18.22 tons blende	0.05 ozs.	6.4 ozs.	2.8%	55.6%
40.84 tons blende	0.16 ozs.	10.2 ozs.	3.4%	49.3%
7.24 tons blende	0.28 ozs.	15.7 ozs.	8.8%	37.1%
33.70 tons iron	0.34 ozs.	24.9 ozs.	5.1%	10.1%

	Gold	Total	Silver	Total
18.22 tons blende	0.5 ozs.	0.911 ozs.	6.4 ozs.	116.608 ozs.
40.84 tons blende	0.16 ozs.	6.534 ozs.	10.2 ozs.	416.568 ozs.
7.24 tons blende	0.28 ozs.	2.027 ozs.	15.7 ozs.	113.663 ozs.
33.70 tons iron	0.34 ozs.	11.458 ozs.	24.9 ozs.	839.13 ozs.
100.00 tons	0.209 ozs.	20.930 ozs.	14.86 ozs.	1485.969 ozs. silver.

Figured on a percentage basis the distribution of the metals in the various products is as follows:—

	Gold	Silver	Lead	Zinc
18.22 tons blende	4.25%	7.85%	12.00%	27.84%
40.84 tons blende	31.12%	28.00%	32.63%	55.38%
7.24 tons blende	10.00%	7.70%	15.00%	7.42%
33.70 tons iron	54.63%	56.45%	40.37%	9.36%
100.00 tons	100.00%	100.00%	100.00%	100.00%

Taking the first two products for zinc ore, the latter two for lead smelting ore we have,—59.06 tons assaying 51.2% zinc and containing 35.7% of the gold, 35.85% of the silver, 44.63% of the lead and 83.22% of the zinc; and 40.94 tons of smelting ore containing, 64.63% of the gold, 64.15% of the silver, 55.37% of the lead and 16.78% of the zinc.

It will be seen that the bulk of the gold, silver and lead is contained in the smelting ore where it is of most value to the seller, while the zinc blende is fairly low in lead and silver.

This test (E) is instructive from two view points. First, studied in connection with test (B) it shows that the nature and extent of the roast is the difference between an unsatisfactory result and a brilliant success; hence great care is necessary in roasting these ores in order to obtain the best results. Secondly, that low intensity machines are useless on roasted ore of this type.

The proper treatment of this ore would work out as follows:—

A. Roast the concentrates, as they come from the mill, for the production of the magnetic sulphide.

B. Treat the roasted ore at about 0.40 amperes to remove the strongly magnetic material.

C. Draw off the next weaker magnetic product (pyrites) at about 1 ampere; the residual ore is first class blende exceeding 50% zinc.

D.—Crush the product, magnetic at 1 ampere, (C) to pass a screen aperture of 0.02 in. and re-treat on the separators; the zinc product to be added to the residual ore (blende) from (C) and the magnetic product added to and mixed with the 0.40 ampere magnetic product from (B), to form the smelting ore. There is thus only two classes of ore produced from the combined operation, viz., a smelting ore, containing the bulk of the gold, silver and lead, and a zinc ore of about 50% content of zinc.

LOT NO. 7—ENTERPRISE MINE.

A zinc blende concentrate produced from the ores of the Enterprise mine, Slocan Lake. The concentrates run high in silver and contain along with the zinc blende, galena, siderite, quartz and pyrites. The assay of the ore as received was: silver 115 ozs.,lead 4.8%, zinc 43.70%, and it should be classed as a silver ore, high in zinc. Some attempts were, however, made to separate the silver ores from the zinc blende, but not with unqualified success.

Test A.—Based on removing the siderite on a high intensity machine, and then separating the lead from the zinc by wet concentration. Ore reduced to pass a screen of 0.03 in. aperture. The ore was first passed over the Wetherill machine at 120 volts and 1.85 amperes. Result, 15.5% of the weight treated was removed. The remainder was passed over a concentrating table. The combined results were as follows:—

	Weight	Silver	Lead	Zinc
At 1.85 amperes	15.50%, assaying	67.0 ozs.	2.8%	7.50%
Lead concentrates	3.70%, assaying	293.0 ozs.	45.2%	14.20%
Zinc concentrates	77.10%, assaying	106.0 ozs.	3.0%	51.00%
Loss	3.70%	....	....	....
	100.00%			

The silver to a large extent followed the lead, but the zinc product still contained a higher silver than zinc value, the distribution of the metals in the various products being as follows:—

	Silver	Lead	Zinc
15.50% siderite contained	9.03%	9.00%	2.66%
3.70% galena contained	9.59%	34.83%	1.20%
77.10% zinc blende contained	71.06%	48.14%	90.00%
3.70% loss contained	10.32%	8.03%	6.14%
100.00%	100.00%	100.00%	100.00%

From which it is apparent that the zinc blende, carrying 90% of the total zinc present in the ore, carries also 71% of the silver, and the galena concentrated out, though of high silver tenor, accounts for only 9.59% of the total silver in the ore; this is because there is less than 3.75% by weight of galena removed from the ore. By combining the first two products,—siderite and galena,—a mixture is obtained containing 18.60% of the silver; 43.80% of the lead; 3.80% of the zinc; and assaying, silver 111.5 ozs., lead 10.90% and zinc, 12.8%. This is a good smelting ore, but inasmuch as the object of the test was to separate if possible the silver from the zinc, it did not result successfully. However, looking at it as a zinc ore, we see the percentage of zinc has been raised from 43.7% in the crude concentrates to 51% in the finished results, thus giving a first class zinc ore containing only 3% of lead. On this basis the test is quite satisfactory.

Test B.—This test was designed to (1) remove the lead by means of the Blake electrostatic separator; (2) remove the siderite from the tailings by means of the Wetherill machine; (3) the tailings from the Wetherill machine to be concentrated on a table to make an additional lead concentrate to remove silica and make a cleaner zinc middling. The Blake products, however, were so unsatisfactory that the remainder of the tests were abandoned. The result of the Blake test was as follows:—

Weight	Silver	Lead	Zinc
Lead product 14% assaying	27.3 ozs.	4.31%	44.00%
Zinc product 86% assaying	82.0 ozs.	3.7%	37.20%

Better results could no doubt be obtained by crushing the ore finer, but then the dust question would prove too serious for a commercial undertaking.

Test C.—A second portion of the ore was given a very light roast, the loss in weight being 8.25%. The roasted ore was next passed over the Wetherill machine at 120 volts and the following amperes:—

Weight	Silver	Lead	Zinc
0.45 amperes 26.30% assaying	221.0 ozs.	6.40%	21.0%
0.90 amperes 13.50 % assaying	187.5 ozs.	5.80%	47.2%
4.00 amperes 40.80% assaying	62.6 ozs.	3.20%	56.8%
Non-magnetic 19.40% assaying	17.6 ozs.	4.80%	57.2%

Showing the following distribution of values.

Weight	Silver	Lead	Zinc
26.30% contained	51.7%	35.8%	12.2%
13.50% contained	22.5%	16.6%	14.1%
40.80% contained	22.7%	27.8%	51.3%
19.40% contained	3.1%	19.8%	22.4%
100.00%	100.0 %	100.0 %	100.0 %

The iron product removed at 0.45 amperes contained nearly 52% of the total silver in the roasted ore against only 9% of the silver removed in the raw siderite, test (A), showing a very material change in the silver combination due to the roasting and pointing clearly to the solution of this ore treatment. It will be noticed that the resin blende, non-magnetic at 4.0 amperes contains only a little silver (17.6 ozs.), that the blende magnetic at 4.0 amperes contained 62 ozs. of silver, and so on up the line; the more highly magnetic the material the higher the silver value it contained. In other words the more iron present, the higher the silver values and conversely, the higher the zinc values in the various products, the less silver they contained.

The lead content of the zinc products can be reduced by separating the galena of the ore by concentrating previous to roasting, the treatment working out as follows:—

(A) Crush to about 25-mesh and concentrate out the galena.

(B) Roast the tailings from the galena concentration.

(C) Magnetic separation of the iron from the zinc at about 0.50 amperes.

(D) Mixing the galena concentrate with the iron product for a smelting ore.

Test D.—A portion of this ore was crushed to pass a screen aperture of 0.005 in. and agitated in 0.3% KCN solution for eight hours.

Assay of ore before treatment.......silver	102 0 ozs.
Assay of ore after treatment ....silver	85.5 ozs.
Silver extraction	16.1%

Test E.—Ore ground as in last test and agitated in 0.15% KCN solution for four hours in the presence of mercury.

Assay of ore before treatment.......silver	102.0 ozs.
Assay of ore after treatment ....silver	83.9 ozs.
Silver extraction......................	17.7 %

These tests show that there is but very little native silver in the ore, and from other lines of investigation, it is fair to assume that the bulk of the silver is held in combination with copper minerals, e.g. freibergite.

LOT NO. 8—MOLLY GIBSON MINE.

This is crude ore from the Molly Gibson mine, near Nelson, sent in by the owners as average vein rock. It contains considerable siderite, galena, silica, pyrites and zinc blende, and assayed: gold 0.02 ozs., silver 71.5 ozs., lead 43%, zinc 11.30%. This is in no sense a zinc ore, though the first test was based on that assumption and carried out before the lead value was determined.

Test A.—The ore was given a light roast which resulted in a reduction of weight amounting to 20%. The roasted ore was passed through a Wetherill magnetic separator, for the following result:—

	Weight	Silver	Lead	Zinc
At 0.30 amperes	36.30% assaying........	60.50 ozs.	22.7%	21.00%
At 0.65 amperes	7.96% assaying.........	93.60 ozs.	33.7%	12.10%
At 3.00 amperes	20.72% assaying.........	90.00 ozs.	42.1%	21.2%
Non-magnetic	35.02% assaying.........	69.50 ozs.	67.6%	5.90%

The magnetic separation on the crude roasted ore, as above, makes practically no separation of the silver or of the zinc and is consequently unfavourable.

Test B.—Treating the raw ore crushed to pass 0.03 in. screen aperture on the Wetherill magnetic separator, current 120 volts. Result:—

	Weight	Silver	Lead	Zinc
At 1.90 amperes	23.70%	27.6 ozs.	2.70%	11.0%
At 4.00 amperes	17.10%.................	102.4 ozs.	6.00%	32.6%
Non-magnetic	59.20%.................	69.2%	62.00%	5.1%

By this method a 32% zinc product is obtained carrying 102ozs. of silver, which is not satisfactory.

Test C.—Ore reduced to pass 0.03 in. screen aperture and sized, giving +.008 in. screen aperture 57% and —0.008 in. 43%. Each product was treated separately on concentrating tables and like products were mixed for the general result, the aim being to produce (A.) a silver-lead concentrate, (B.) a zinc concentrate, and (C.) tailings carrying the silica, etc.

The results were as follows:—

	Weight	Silver	Lead	Zinc
Lead concentrate	55.40% assaying.........	81.0 ozs.	74.2%	4.8%
Zinc concentrate	19.40% assaying.........	63.6 ozs.	12.0%	17.6%
Tailings	23.10% assaying.........	56.6 ozs.	5.0%	13.6%
Loss	2.10%	....	....	....

The lead concentrate is a first class product. The zinc concentrate runs too high in lead, owing to the lead middlings going with the zinc. In working on a large scale, however, this lead-zinc middling would be ground finer and re-treated. Lastly the silicious tailings run 56.6 ozs. in silver. I am doubtful if 11% zinc can be separated profitably from a straight silver-lead ore, such as this Molly Gibson ore turns out to be. It being evident that smelting is the proper treatment for ore of this composition and grade, no further tests were made.

LOT NO. 9—BIG LEDGE.

This ore is from the "Big Ledge" on Arrow Lake, near the Hot Springs. Three lots of this ore were shipped to the Zinc Commission by persons interested in the property. Lot 9 under consideration is said to have been taken from the portal to the face of a 30—foot tunnel in the ledge, all in solid ore, "Empress Claim." The ore consisted of massive pyrites and pyrrhotite, scattered through which occurs resin blende and ferruginous blende, the latter occurring in small quantities. There is but little gangue in this ore. It assayed: gold, trace, silver 0.70 ozs., lead, trace, zinc 19.4%.

Test A.—The ore reduced to pass a screen of 0.03 in. aperture and passed through the Wetherill separator, with a current of 120 volts exciting the magnets. The magnetic material removed was as follows:—

Weight		Silver	Lead	Zinc
At 0.90 amperes	16.95% assaying........	trace	trace	4.10%

The product non-magnetic at 0.90 amperes was next passed through a Blake electrostatic machine for the following separation:—

Weight	Silver	Lead	Zinc
16.40% assaying	1.40 ozs.	trace	18.20%
66.65 assaying	trace	trace	24.60%

There is no commercial product obtained by this method of treatment.

Test B.—A roast before magnetic separation was tried. Loss in roasting, 12.5%. The roasted ore was passed through a Wetherill separator at 120 volts and amperes as indicated. Result:—

Weight		Silver	Lead	Zinc
At 0.45 amperes	37.0% assaying..........	0.40 ozs.	trace	8.4%
At 3.00 amperes	54.8% assaying..........	trace	trace	32.7%
Non-magnetic	8.2% assaying..........	1.20 ozs.	0.20%	6.0%

The 54.8% zinc product accounts for 83.3% of the zinc in the ore, but inasmuch as it assays only 32.7% zinc, the product is too low in grade to pay under the conditions that now obtain in British Columbia.

LOT NO. 10—BIG LEDGE.

This ore is from the "Big Ledge" and said to be taken from an "open cut showing 10 feet of solid ore." The ore is practically identical with that of Lot No. 9, and assayed: gold, trace, silver 0.70 ozs., lead, trace, zinc 20.6%.

Test A.—The ore was crushed to pass a screen of 0.03 in. aperture and treated in the raw state on a Wetherill magnetic separator, current 120 volts. The magnetic material removed was as follows:—

At 0.90 amperes	5.76% of weight assaying, zinc	5.60%
At 1.80 amperes	8.89% of weight assaying, zinc	5.90%
Non-magnetic	85.35% of weight assaying, zinc	22.40%

The non-magnetic product at 1.80 amperes was next passed over a Blake electrostatic machine, yielding the following products:—

	Silver	Lead	Zinc
A. 7.35% by weight assaying............	1.40 ozs.	0.30%	14.5%
B. 78.00% by weight assaying............	....	....	23.2%

These results are practically identical with those obtained with Lot No. 9 and are quite unsatisfactory.

Test B.—The same ore passing 0.03 in. screen aperture was given a light roast, the loss of weight in roasting being 16.6%. The roasted ore was next treated on a Wetherill separator at 120 volts and amperes as stated below, the result being:—

At 0.45 amperes	26.83% of weight assaying	8.10% zinc
At 4.00 amperes	58.51% of weight assaying	31.6 % zinc
Non-magnetic	14.66% of weight assaying	5.1 % zinc

The results again confirm those of Lot No. 9.

Test C.—The same ore treated on a concentrating table to separate the gangue from the mixed sulphides gave the following result:—

A. Zinc concentrates	84.40% of weight assaying	22.4% zinc
B. tailings	13.80% of weight assaying	7.3% zinc
Loss	1.80%	

A portion of the concentrate (A) was next passed through the Wetherill separator, current 120 volts.

At 0.90 amperes } At 1.95 amperes }	15.20% of weight assaying	10.5% zinc
Non-magnetic	69.20% of weight assaying	25.7% zinc

Another portion of the concentrate (A) was given a slight roast and then treated on the Wetherill magnetic separator, current 120 volts.

At 0.30 amperes	23.20% of weight assaying	7.90% zinc
At 4.00 amperes	57.00% of weight assaying	31.60% zinc
Non-magnetic	4.20% of weight assaying	0.70% zinc

It will be noticed that the highest zinc product obtained is 31.6% zinc, identical with the similar product obtained from the roasted ore, without previous concentration. It is evident therefore that 31% zinc is the best that can be obtained from these low grade, heavy sulphide ores without a higher roast, looking toward the formation of magnetic pyrites in the roasting process.

LOT NO. 11—BIG LEDGE.

This is from the "Big Ledge" deposit, said to be taken from a cut 60 feet long by six feet in width on the "Monarch Claim", and to be a general sample for the width given.

Test A.—The ore was crushed to pass a screen aperture of 0.03 in. sampled and assayed as follows: gold, trace, silver 0.80 ozs., lead, trace, zinc 22.7%. This lot contains a higher grade of zinc sulphide than the others as it assays higher, though containing almost double the amount of gangue occurring in Lot No. 10; otherwise the composition of the ores and gangue are similar. The raw ore was treated on the Wetherill separator, current 120 volts. Result:—

At 1.85 amperes	17.10% of weight assaying	9.20% zinc
At 4.00 amperes	56.10% of weight assaying	35.30% zinc
Non-magnetic	26.80% of weight assaying	6.10% zinc

Test B.—The same ore was lightly roasted. The loss of weight was 15%. The roasted ore assayed 24.1% zinc. It was treated on the Wetherill machine, current 120 volts. Result:—

At 0.45 amperes	30.4% of weight assaying	10.10% zinc
At 4.00 amperes	62.2% of weight assaying	33.60% zinc
Non-magnetic	7.4% of weight assaying	2.60% zinc

The zinc product obtained in both A. and B. tests is of too low grade to be of commercial value, under the conditions that now obtain in British Columbia.

Test C.—A portion of the ore was concentrated on a table giving the following products:—

A. Zinc concentrates	65.4% of weight assaying	29.4% zinc
B. Tailings	29.3% of weight assaying	12.6% zinc

The product A. was next treated on the Wetherill machine with a current of 120 volts. Result:—

At 1.00 amperes	5.24% of weight assaying	6.8% zinc
At 4.00 amperes	47.06% of weight assaying	38.6% zinc
Non-magnetic	13.10% of weight assaying	3.6% zinc
	65.40%	

The above products were then thoroughly mixed and subjected to a light roast, followed by Wetherill treatment.

At 0.15 amperes	13.4 %	of weight assaying	6.30% zinc
At 0.55 amperes	6.75 %	of weight assaying	18.70% zinc
At 4.00 amperes	40.80%	of weight assaying	42.50% zinc
Non-magnetic	4.45%	of weight assaying	2.60% zinc
	65.4 %		

The above would indicate that the best method to pursue is first to concentrate the ore to eliminate the gangue. Second, to roast the zinc concentrate for magnetic pyrites. Third, to treat on high intensity machine.

The three lots of "Big Ledge" ore (Nos. 9, 10 and 11) being of almost identical composition and acting alike in the magnetic separators, it was decided to mix them together and treat as one lot in the official tests.

Test D.—Official—On Lots No. 9, 10 and 11 the ore was mixed and ground to pass a screen of 0.02 in. aperture. A sample of the mixed lots was then cut out. It assayed 20.40% zinc.

The ore was next sized, giving the following result:—

+ 0.008 in. aperture	65.0%
+ 0.0055 in. aperture	13.8%
— 0.0055 in. aperture	21.2%

Each size was treated separately on a concentrating table, but the corresponding products were combined to form one general result.

A. Zinc concentrates	67.24%	of weight assaying	26.40% zinc
B. Tailings	28.50%	of weight assaying	7.40% zinc
Loss	4.26%		

The zinc concentrate, product (A), was then sized, roasted and again sized. Result:—

	Before roasting	After roasting
+ 0.008 in. aperture	39.5%	49.70%
+ 0.0055 in. aperture	21.0%	19.50%
— 0.0055 in. aperture	39.5%	30.8 %

The ore was charged into a red hot muffle and gradually brought up to a dull red heat and kept at that temperature for about 20 minutes. The loss of weight in roasting amounted to 5.7%.

The roasted ore was next treated on a Wetherill magnetic separator with a current of 120 volts. Result:—

At 0.30 amperes	25.56%	of weight assaying	11.30% zinc
At 0.90 amperes	9.90%	of weight assaying	26.60% zinc
At 4.00 amperes	57.90%	of weight assaying	40.70% zinc
Non-magnetic	6.64%	of weight assaying	3.7 % zinc

There is but one product in the above that is of possible value in British Columbia to-day, viz. the 40.7% zinc, though it may be possible by grinding finer to obtain part of the zinc in the 29.6% product as a 40% zinc ore. However, we can only figure on the 57.9 tons of 40.7% zinc.

Going back to the original ore before concentration, we find that

100.00 tons	@ 20.4%=40,800 lb. of zinc	
67.24 tons concentrate	@ 26.4%=35,502.7 lb. of zinc, saving	87%
32.76 tailings and loss	5,297.3	loss 13%
	40,800	

The 57.9 tons of roasted product assaying 40.7% zinc accounts for 79.5% of the zinc in the roasted ore.

An attempt was next made to reach the highest possible zinc tenor from the 4-ampere product assaying 40.7% zinc. It was, therefore re-treated on the Wetherill separator with the following results:—

At 1.35 amperes	5.18%	of weight assaying 27.5% zinc
At 1.80 amperes	6.12%	of weight assaying 31.4% zinc
At 4.00 amperes	46.60%	of weight assaying 42.1% zinc
	57.90%	

It is therefore evident that by this method of treatment a higher grade of zinc than 42% cannot be made; and very likely when working on a commercial basis, a 40% zinc blende with a recovery of 70% to 75% of the zinc in the raw ore, will be about the best result that can be hoped for.

LOT NO. 12—GOODENOUGH MINE.

This lot is high grade zinc ore from the Goodenough mine, almost good enough for direct marketing without further treatment. The ore is a fine grained blende, containing galena sprinkled through it, siderite and a little pyrites, but scarcely any gangue.

The ore was crushed to pass a screen of 0.03 in. apertures, sampled and assayed with the following result:

Gold 0.02 ozs.; silver 22 ozs.; lead 10.8%; zinc 45%.

This is in fact a very fair zinc ore just as it is, except for the amount of lead it contains.

Test A.—The raw ore was passed through the Wetherill separator, current 120 volts, 40 amperes. Result:

Weight	Silver	Lead	Zinc
29.8% magnetic assaying....................	11.0 ozs.	0.50%	56.7%
70.2% non-magnetic assaying.............	25.6 ozs.	14.40%	41.5%

29.8 tons blende	@ 56.7% zinc	=33,793 lbs. zinc, saving	36.7%
70.2 tons siderite	@ 41.5% zinc	=58,266 lbs. zinc, loss	63.3%
100.0 tons	@ 45.0%	=92,059 lbs. zinc	

It would appear from the above that about two-thirds of the zinc is non-magnetic (resin blende) and that the ferruginous zinc is not associated with the resin blende and only slightly with the silver; furthermore, as the 29.8 tons of 56.7% zinc is a first-class zinc product in every way the above treatment might very well form one step in a combination process for the beneficiation of these ores. The 70.2 ton product, however, is a rather difficult problem, on account of the high lead and medium grade zinc in the non-magnetic product. This indicates the removal of the lead prior to the magnetic treatment.

Test B.—The ore was first roasted, which resulted in 12.3% loss of weight. The roasted ore assayed 46.7% zinc. It was next treated on the Wetherill separator with a current of 120 volts. Result:

Weight	Silver	Lead	Zinc
Amperes 0.45; 11.76% assaying............	13.8 ozs.	7.5%	32.7%
Amperes 4.00; 79.77% assaying............	17.4 ozs.	5.8%	52.8%
Non magnetic; 8.47% assaying............	72.4 ozs.	64.8%	9.4%
100.00%			

The distribution of the zinc in the above products, figured in percentages of the whole is as follows:

11.76 tons, 32.7% zinc = 8.2% of total zinc
79.77 tons 52.8% zinc = 89.8% of total zinc
8.47 tons, 9.4% zinc = 2.0% of total zinc

This test gives a very fair zinc recovery, but the lead in the zinc product, 5.8%, is still high, as is also the zinc in the first and last products, of which only the last is a good smelting ore.

Test C.—The raw ore passing 0.03 in. screen aperture was treated on concentration tables to remove the galena. Result:

Weight	Silver	Lead	Zinc
Lead concentrate 10.70% assaying	81.6 ozs.	62.4%	11.10%
Tailings (zinc) 89.30% assaying...........	15.0 ozs.	4.1%	48.7 %

This test demonstrates the best solution for the treatment of this ore. The 10.7% concentrate is a first-class smelting ore containing 40% of the silver and 62% of the lead in the raw ore. The zinc saving works out as follows:—

89.3 tons zinc blende	@48.70% zinc	= 86,978.2 lbs. zinc, saving	97.4%
10.7 tons lead concentrate	@11.10% zinc	= 2,375.4 lbs. zinc, loss	2.6%
100.0 tons	@44.67%	= 89,353.6 lbs. zinc	

By passing the zinc product through a high intensity machine in the raw state it could easily be raised to 50% zinc content, or by a previous roast to to about 55% zinc.

Test D.—Official. The ore reduced to pass a screen with 0.03 in. aperture gave the following:—

+ 0.008 in., raw 69.8%, roasted 67.10%
+ 0.0055 in., raw 8.3%, roasted 10.70%
— 0.0055 in., raw 21.9%, roasted 22.20%

This mixture assayed: gold 0.02 ozs., silver 22.0 ozs., lead 10.80%, zinc 45%. It was charged into a red hot muffle and roasted at a dull red heat for 30 minutes. Loss in weight, 3.4%.

The roasted ore was then passed over a Wetherill separator, with a current of 120 volts. Result:

Weight		Silver	Lead	Zinc
Amperes 0.15	5.45% assaying...........	15.2 ozs.	7.6%	25.9%
Amperes 0.30	6.00% assaying...........	17.2 ozs.	8.2%	35.7%
Amperes 4.00	82.67% assaying...........	16.0 ozs.	5.1%	52.1%
Non-magnetic	5.88% assaying...........	66.6 ozs.	53.8%	13.8%

This test confirms (B) and proves conclusively that the galena should be removed before roasting; in fact that the proper treatment of the ore is,

A. Concentrate out the galena, saving all the tails.

B. Sell the tailings for zinc ore direct; or raise their value above 50% zinc by running them through a high intensity machine for a 50% zinc product; or roast and treat on a low intensity machine for a 55% zinc product; in either case the product removed by the separators can be added to the lead concentrates for smelting ore.

LOT NO. 13—HEWITT MINE.

This ore is from a lot of zinc concentrates made during a trial run of Hewitt ore in the Wakefield mill last fall. It assayed: gold 0.03 ozs., silver 80.2 ozs., lead 11.5%, zinc 32.8%, and contained zinc blende, siderite, pyrites and gangue.

Test A.—The scheme followed in this test is to remove the siderite on the Wetherill separator, then separate the lead from the zinc product on the Blake electrostatic machine, it having been previously determined that only a small amount of the zinc blende was magnetic.

Wetherill separator, current 120 volts. Result:

Weight		Silver	Lead	Zinc
Amperes 2.20	26.10% assaying............	13.2 ozs.	1.10%	5.6%
Non-magnetic	73.9 % assaying...........	97.5 ozs.	14.90%	42.2%

The non-magnetic product, 73.9% by weight, containing 95% of the zinc in the crude ore, was next passed over the Blake electrostatic machine with the following results:—

Weight	Silver	Lead	Zinc
A. Lead product 10.19% assaying..........	198.4 ozs.	35.6%	24.8%
B. Zinc product 63.71% assaying..........	81.0 ozs.	11.6%	45.0%

Test B.—A repetition of A. for confirmation.

Weight	Silver	Lead	Zinc
Siderite at 2.2 amperes 26.2% assaying.....	11.0 ozs.	0.80%	6.5%
A. Lead product (Blake) 15.0% assaying	207.2 ozs.	35.40%	24.7%
B. Zinc product (Blake) 58.8% assaying.....	77.2 ozs.	11.60%	45.6%

All the zinc products are low in zinc and high in lead, and further, the lead is too finely divided, or too intimately mixed with the zinc to be cleanly separable on the Blake machine.

Test C.—The ore was given a light roast resulting in a loss of weight amounting to 15%; the roasted ore was then passed over the Wetherill separator at 120 volts. Result:

Weight	Silver	Lead	Zinc
Amperes 0.30; 36.00% assaying............	116.6 ozs.	9.9%	16.2%
Amperes 0.60; 9.33% assaying............	158.8 ozs.	16.2%	40.0%
Amperes 4.00; 49.34% assaying............	51.0 ozs.	10.9%	49.7%
Non-magnetic; 5.33% assaying............	49.0 ozs.	53.2%	16.1%

The zinc product contains 63% of the zinc in the original ore, magnetic at 4 amperes, but the lead is too high for a zinc ore.

Test D.—Official. The large amount of galena included in the particles of zinc blende clearly indicates that the ore should be crushed finer and the galena concentrated from the blende as a condition precedent to good magnetic work The ore was therefore reduced to pass a screen aperture of 0.03 in. and after sizing was passed over a concentrating table. Each size was concentrated separately, but the various like products were combined for one general result. The ore contains considerable clear quartz and it was noticed while making this test that an apparently worthless quartz gangue could be cut out in the concentration on a commercial scale; but for obvious reasons the test was carried through as designed, producing· (A) High grade galena; (B) high grade zinc concentrates; (C) a zinc-silica tailing, all of which was saved for further treatment.

Screen analysis of crushed ore, all passed 0.03 in.

+ 0.008 in. aperture, 17.88%
+ 0.005 in. aperture, 27.06%
— 0.005 in. aperture, 55.06%

The ore assayed, silver 77·6 ozs. lead 11·1%, zinc 32%

The products of concentration are as follows:—

Weight	Gold	Silver	Lead	Zinc
14.48% lead concentrate, assaying	0.04 ozs.	192.50 ozs.	67.40%	7.20%
61.62% zinc concentrate assaying......	......	......	......	40.60%
21.52% tailings assaying.............	......	......	......	23.20%
2.38% loss.......................	......	......	......	......
100.00%				

The lead concentrate contains only 7.2% zinc and has a high silver and lead tenor, as well as accounting for a large proportion of these metals contained in the original ore, namely 88% of the lead and 35.9% of the silver. The zinc concentrate contains 77.93% of the zinc in the original ore.

The concentration process is consequently the means necessary for the separation of the bulk of the lead and over one-third of the silver from the crude concentrates, herein called the original ore.

The zinc concentrate, 61.62% of the original ore, was then sized, and charged into a red hot muffle, drawn in 30 minutes, allowed to cool gradually, and again sized. The loss in weight during the roast was 4%.

Screen analysis	Before roasting	After roasting
+ 0.008 in.	15.20%	21.80%
+ 0.005 in.	13.00%	19.20%
— 0.005 in.	71.80%	59.00%

The roasted zinc concentrates were then treated on the Wetherill separator at 120 volts and 0.30 amperes, which was subsequently found to be too strong; consequently the 0.30 ampere product was re-passed at 0.15 amperes The following results are, however, given, in the order in which they were made:

Weight	Silver	Lead	Zinc
At 0.30 amperes 40.50% assaying..........	132.60 ozs.	4.90%	25.2%
Non-magnetic 59.50% assaying..........	26.50 ozs.	0.60%	57.3%

The 0.30 ampere product was next re-passed at 0.15 amperes.

Weight	Silver	Lead	Zinc
0.15 amperes 34.2% assaying..............	139.0 ozs.	5.7%	22.5%
Non-magnetic 6.3% assaying..............	63.8 ozs.	2.1%	48.6%

It is evident, however, that the 40.5 tons of 0.15 ampere product should have been ground finer before passing it through the separator in order to liberate the zinc from the iron; however, by mixing the non-magnetic products at 0.15 and 0.30 amperes, a product is obtained amounting to 63% by weight of the zinc concentrates from the tables and assaying: silver 30. oz.;, lead 0.74%, zinc 56.46%, or a saving of 86% of the zinc in the roasted concentrates or 67% of the zinc in the original ore.

The tailings from the concentrating table, 21.52% by weight, assaying 23.2% zinc were roasted under the same conditions as the zinc concentrates, the loss of weight in roasting being 2%. The roasted product was then passed over a Wetherill separator at 120 volts current, giving the following products:—

Weight		Silver	Lead	Zinc
0.15 amperes	45.8%	23.20 ozs.	2.10%	6.80%
0.60 amperes	10.4%	....	....	23.20%
4.00 amperes	18.7%	....	....	45.40%
Non-magnetic	25.1%	....	....	49.90%

Combining now	59.50%	at	57.3%	zinc
	6.30%	at	48.6%	zinc
	18.70%	at	45.4%	zinc
	25.10%	at	49.9%	zinc

a zinc blende product is obtained amounting to 48.17% of weight of the crude ore and assaying 54.8% zinc, a saving of 83.3% of the zinc in the raw concentrates.

It is also seen by that by combining

Lead concentrates	14.48%
Iron product	34.20%
Iron product	45.80%

a mixture is obtained, containing 44.36% by weight of the initial ore, assaying silver 131.27 oz., lead 25.04%, zinc 14.08%, iron 23%, silica 3.6%, a recovery of 75.0% of the silver and 100% of the lead in the initial ore.

On a large working scale, however, the proper treatment of this Lot No. 13 would work out something like this:

A. Crush to pass 0.02 in. screen aperture.

B. Concentrate out the lead, making a middling product of zinc and saving all the tailings.

C. Roast the zinc middlings and treat on magnetic separator.

D. The tailings from the concentrating tables would probably not pay for further magnetic treatment, particularly if re-concentrated before impounding, but might be mixed with the lead concentrates for a smelting ore, or with some of the iron products from the magnetic separators.

LOT NO. 14—EMILY-EDITH MINE.

This is a lot of concentrates made in the Wakefield mill from Emily-Edith ore.

The concentrates were reduced to pass a screen of 0.03 in. apertures, sampled and assayed, giving the following results:—Gold 0.02 ozs.; silver, 24.0 ozs.; lead, 8.0%; zinc, 38%.

Test A.—Preliminary. Wetherill treatment to remove the siderite, followed by Blake electrostatic treatment to separate the lead from the zinc. Wetherill separator at 120 volts and 1.85 amperes. Results:

Weight		Silver	Lead	Zinc
At 1.85 amperes	2.52% assaying..........	32.6 ozs.	3.0%	6.8%
Blake lead	9.68% assaying..........	49.2 ozs.	17.1%	27.3%
Blake zinc	87.80% assaying..........	20.5 ozs.	6.5%	39.6%

This line of treatment not indicating any favourable result, roasting was next tested.

Test B.—The ore reduced to pass an 0.03 in. screen aperture was roasted and sampled. It assayed 39.8% zinc after roasting and was passed over a Wetherill separator with a current of 120 volts. Result:

Weight		Silver	Lead	Zinc
At 0.30 amperes	16.40% assaying.........	49.6 ozs.	10.7%	24.1%
At 0.45 amperes	15.60% assaying.........	32.4 ozs.	7.6%	26.6%
At 4.00 amperes	62.50% assaying.........	16.2 ozs.	4.8%	45.6%
Non-magnetic	5.50% assaying.........	18.4 ozs.	28.8%	22.2%

The results are entirely unfavourable. The Emily-Edith concentrates as represented by this Lot No. 14 are very base, containing both resin blende and ferruginous blende in more or less intimate mixture with galena, pyrites, siderite and quartz. It is evident that the lead must be removed, so far as possible, by concentration, inasmuch as the Blake electrostatic machine does not effect sufficient separation of the lead from the zinc.

Test C.—Official. Based on removing the lead by concentration, roasting the zinc middlings and treating the roasted product on magnetic separators. The ore crushed to pass 0.03 in. screen aperture assayed as follows: Silver, 23.50 ozs.; lead, 8.3%; zinc, 37.1%. It gave the following sizes:

+ 0.008 in.........................	34.01%
+ 0.005 in..........................	29.70%
— 0.005 in.........................	36.29%

Each size was treated separately on a concentration table, and like products combined for the general result.

Weight		Silver	Lead	Zinc
Lead concentrate	7.81% assaying..........	58.50 ozs.	54.6%	12.5%
Zinc concentrate	80.55% assaying..........	....	....	42.0%
Tailings	9.81% assaying..........	....	....	18.6%
Loss	1.83%..................	....	....	......

The 7.81% of lead concentrates obtained from this wet concentration contain 19.43% of the silver and 51.38% of the lead in the original ore.

The zinc concentrate, 80.55% of the original ore, was charged into a red hot muffle, raised to dull red heat and drawn while smoking freely at 35 minutes from time of charging.

Sizing test: Zinc concentrates:

	Before roasting	After roasting
+ 0.008 in.......	26.6%	38.7%
+ 0.005 in.......	18.5%	25.1%
— 0.005 in.......	54.9%	36.2%
	100.0%	100.0%

The roasted ore was then passed over a Wetherill separator at 120 volts and 0.30 amperes. Result:—

Weight	Silver	Lead	Zinc
31.73% magnetic, assaying	32.4 ozs.	5.3%	36.4%
68.27% non-magnetic, assaying	16.2 ozs.	2.8%	47.9%

The magnetic product was then re-treated at 0.12 amperes, on the assumption that it was sufficiently fine to liberate the blende, which, however, proved incorrect; the grade of the blende removed shows clearly that a much higher recovery of marketable blende could be effected from this magnetic material, if crushed fine enough to liberate the minerals:

Weight		Silver	Lead	Zinc
0.15 amperes	19.42% assaying..........	36.4 ozs.	7.6%	23.9%
Non-magnetic	12.31% assaying..........	30.0 ozs.	4.5%	44.3%

By mixing the above two non-magnetic products, 68.27% and 12.31%, a zinc ore is obtained amounting to 62% of the weight of the original ore and assaying; silver 18.30 ozs., lead 3.06%, zinc 47.29%, showing a recovery of 88.9% of the zinc in the roasted ore, or 81% of the zinc in the original ore.

The 9.81% tailings from the concentration test were roasted under the same conditions as the zinc concentrates and treated on the Wetherill separator at 120 volts. Loss of weight in roasting, 1.5%. Result:

	Weight	Silver	Lead	Zinc
At 0.15 amperes	8.00% assaying..........	....	....	12.00%
At 0.60 amperes	32.00% assaying..........	....	....	17.00%
At 4.00 amperes	36.00% assaying..........	....	....	24.00%
Non-magnetic	24.00% assaying..........	....	....	16.40%

All these products are possibly too low grade to work up for marketable zinc, except perhaps that product magnetic at 4 amperes, assaying 24% zinc. This work on the tailings is very instructive, showing as it does an almost uniform zinc value in three of the four products, demonstrating clearly the intimate mixture of the zinc blende with the other minerals.

Test D.—Official. A portion of the roasted zinc concentrate was given two passes through a Dings magnetic separator at 120 volts and 0.30 amperes. Result:

	Weight	Silver	Lead	Zinc
Magnetic product	22.2% assaying........	25.5%	4.6%	39.2%
Non-magnetic product	77.8% assaying........	20.2%	3.2%	46.8%

The non-magnetic product contains 80.8% of the zinc in the roasted ore, or 73.72% of the zinc in the original ore. Owing to the construction of the Dings machine used in these tests the rotating magnet cannot be brought as close to the ore as those in the Wetherill machine; consequently the latter will give a little better results at similar amperes. The Dings machine, however, has greater capacity, and for roasted ore requiring no higher power than 0.40 amperes is an excellent machine.

The proper treatment of these mill concentrates should begin with finer crushing, say to 0.02 in. screen aperture. In this way a higher recovery of zinc and lead would be made and a tailing of lower zinc tenor. The scheme would be about as follows:—

A. Crush to pass screen of 0.02 in. aperture.

B. Concentrate on table for (1) lead concentrate, (2) zinc concentrate, (3) tailings to be filtered and saved for mixing with the magnetic product for a smelting ore, or sold as they are.

C. Roast the zinc middlings and treat on low intensity magnetic separator. (2) Re-grind the magnetic product and re-treat it on the magnetic separators. (3) Mix the final magnetic product (a) with the lead concentrate, (b) with the tailings or (c) mix together the lead concentrate, tailings and magnetic product for a lead smelting ore, just as found preferable.

LOT NO. 15—LUCKY JIM MINE.

This is crude concentrating ore from the "Lucky Jim" mine, consisting of zinc blende, galena, pyrites and quartz gangue.

The ore was crushed to pass a screen of 0.05 in. apertures, sampled and assayed, giving silver 11.50 ozs.; lead, 9.80%; zinc, 33.20%.

Sizing test—	+	0.02 in.	58.42%
	+	0.006 in.	23.59%
	—	0.006 in.	17.99%
			100.00%

Test A.—Preliminary. These various sizes were fed separately to a concentration table and similar products combined for the general result:

	Weight	Silver	Lead	Zinc
Lead concentrate	17.46% assaying	49.4 ozs.	49.4%	15.6%
Zinc concentrate	60.56% assaying	6.5 ozs.	4.4%	42.3%
Tailings	16.43% assaying	2.8 ozs.	0.1%	19.6%
Loss	5.55	....	....	....

The concentration of the lead is quite satisfactory, with a saving of 88%, containing 75% of the silver present in the crude ore. The zinc, however, is too high for a smelting ore, but it could be reduced by making a lead-zinc middling on the concentration table, to be ground finer and re-concentrated to separate the lead from the zinc. In testing on a small scale, however, it is preferable to grind the ore finer before concentration. The zinc product is also of fair grade, and accounts for 77.2% of the zinc in the crude ore, a very fair saving.

The zinc product was next passed over a Wetherill separator, to remove the siderite; the non-magnetic product from the Wetherill was then passed over the Blake electrostatic machine.

Wetherill treatment at 120 volts and 2.15 amperes, gave:

	Weight	Silver	Lead	Zinc
	10% assaying	....	....	5.70%
Blake A.	65% assaying	....	....	44.60%
Blake B.	25% assaying	....	....	20.00%

The test as a whole is not quite satisfactory, but as a preliminary test it furnished needful data, pointing toward the correct method of treatment.

Test B.—To remove the siderite on the Wetherill machine, without previous roasting. Second, pass the concentrates freed from the siderite, over Blake electrostatic separator to repel the galena from the non-conducting blende and silica. Third, separate the silica from the blende on a wet concentrating table.

Ore taken at 0.03 in. screen aperture unsized; Wetherill treatment at 120 volts gave the following result:

	Weight	Silver	Lead	Zinc	Iron	Silica
1.60 amperes	7.25%.	2.30 ozs.	2.10%	6.10%	20.5%	3.4%
Blake A	9.75%.	17.00 ozs.	16.70%	25.10%	......	......
Zinc concentrate	62.00%.	4.00 ozs.	1.90%	42.6 %	......	......
Lead concentrate	10.12%.	54.50 ozs.	54.80%	10.7 %	......	......
Loss	11.88...	......	......	......	......	......

This combination method recovers 79.7% of the zinc in the original ore in a product assaying 42.6% zinc, fairly free from lead. The lead concentrate shows a saving of 56.6% of the lead in the original ore and 48% of the silver. The method is, however, not good; the product A. is too high in zinc, and the loss through dust, etc. is heavy. Better results could be made on sized products except for the fact that the very fine material and dust cannot be fed independently to either the Wetherill or the Blake machines.

Test C.—Another wet and dry combination method, based on, first, recovery, of the siderite on magnetic separator in the raw state; second, concentrating out the lead by the wettable process, saving all the tailings, including slimes, as a zinc product. Wetherill treatment at 120 volts. Result:

Weight		Silver	Lead	Zinc
1.85 amperes	9.4% assaying..........	3.80 ozs.	1.60%	7.10%
Lead concentrate	11.8% assaying..........	65.00 ozs.	67.40%	4.70%
Tailings (blende)	14.5% assaying..........	6.00 ozs.	3.80%	40.8 %

The recovery of the zinc is 91.5% of that in the original ore, in a product assaying 40.8%. The smelting mixture contains 81.2% of the lead and 66.5% of the silver. Apart from the low tenor of the zinc ore the scheme is not quite practicable.

Test D.—Official. Ore reduced to pass a 0.03 in. screen aperture, assaying silver 11.5 ozs.; lead, 9.8%; zinc, 33.2%.

The screen analysis was:

$$\begin{array}{rl} + & 0.01 \text{ in.} = 44.44\% \\ + & 0.006 \text{ in.} = 25.92\% \\ + & 0.006 \text{ in.} = 29.64\% \\ \hline & 100.00\% \end{array}$$

Each size was fed separately to a concentration table and all corresponding products combined for the general result:—

Weight		Silver	Lead	Zinc
Lead concentrate	14.98% assaying...........	49.5 ozs.	50.2 %	11.9%
Zinc concentrate	55.55% assaying...........	4.2 ozs.	1.7%	44.1%
Tailings	17.01% assaying...........	4.5 ozs.	2.17%	20.2%
Loss	12.46%....................	....	....	....

These results figured in percentage of the whole give the following distribution of metals:—

	Silver	Lead	Zinc
14.98 tons saving	64.5%	76.7%	5.3%
55.55 tons saving	20.3%	9.6%	73.8%
17.01 tons saving	6.6%	3.7%	10.3%
12.46 tons loss	8.6%	10.0%	10.0%
100.00 tons	100.0%	100.0%	100.0%

A portion of the zinc product, 55.55% of the ore, assaying 44.1% zinc, was next passed over the Wetherill separator to remove the siderite; current 120 volts and 1.90 amperes. Result:

Weight	Silver	Lead	Zinc
Magnetic 9.40% assaying	1.60 ozs.	0.90%	6.70%
Non-magnetic 90.60% assaying	3.20 ozs.	1.20%	48.50%

A second portion of the zinc concentrate, 55.55%, assaying 44.1% zinc, was roasted. The roasted ore, assaying 47.5% zinc, was next passed through the Wetherill separator, at 120 volts and amperes as indicated, with the following results:

Weight	Silver	Lead	Zinc
At 0.30 amperes 18.0% assaying	4.6 ozs.	2.3%	14.9%
At 0.70 amperes 7.6% assaying	3.0 ozs.	2.8%	42.7%
At 3.00 amperes 72.4% assaying	3.5 ozs.	0.9%	55.2%
Non-magnetic 2.0% assaying	23.0 ozs.	24.2%	16.1%
100.0%	....	....	....

By mixing the 0.70 and the 3.0 ampere product a zinc product is obtained, amounting to 80% of the roasted ore, assaying 54% zinc. Comparing concentration test A. with test D. the result of the fine grinding is seen to reduce the zinc in the lead concentrate from 15.6% in A. to 11.9% in D., thus producing a better smelting ore. The tenor of the zinc concentrate is raised from 42.3% in A. to 44.1% in D., the zinc value in the tailings remaining about the same in each test. The metal loss, however, is greater in B. than in A., as a result of the finer grinding, this notwithstanding that all the tailings were settled and filtered; the presumption is that some of the water escaped without being properly filtered in D. However the demonstration was made that by finer grinding the galena could be separated from both the zinc and the iron minerals, and when we consider that in practical operations, it is only the middling products that require fine grinding, the test furnishes all the required data, and a repetition of it is unnecessary.

Referring now to the magnetic treatment it will be seen to confirm clearly the general conclusions already reached, viz.: That in the treatment of the blende-siderite ores a 48% to 49% zinc product can be obtained by treating the concentrates in the raw state and 55.0% zinc by roasting the concentrates previous to the magnetic treatment.

"The Lucky Jim" concentrating ore is easily treated. It is in fact what I would designate as one of the average Slocan ores, which require ordinary milling methods; preferably, however, on the more modern principles outlined in the chapter on milling. The zinc concentrates should be treated magnetically as indicated, and the lead concentrates sold to the smelters, mixed with the magnetically separated iron products when they are of sufficient value.

LOT NO. 16—HEWITT MINE.

This is a sample of the concentrating ore from the Hewitt mine, said to be a sample of the feed from the trial run of this ore in the Wakefield mill, the zinc concentrates from which were treated as Lot No. 13. The ore is mostly quartz with a sprinkling of zinc blende and galena in fine grains, together with some siderite and pyrites. It was crushed to pass 0.03 in. screen aperture and assayed, silver 18 oz.; lead 5.4%; zinc, 5.8%.

Screen analysis	+ 0.008 in............	63%
	— 0.008 in............	37%

Test A.—Preliminary. The blende and galena were in such small quantity, it was deemed best to save them as one mixed concentrate, for subsequent separation by the Blake electrostatic machine. Result from wet concentration on tables was as follows:—

Weight		Silver	Lead	Zinc
Mixed concentrates	9.0% assaying.........	94.5 ozs.	34.4%	21.8%
Tailings	82.8% assaying.........	8.8 ozs.	0.7%	4.6%
Loss	8.2%	....	....	....

The concentrates were next passed over the Wetherill separator to remove the siderite and zinc. Current 120 volts. Result:

Weight		Silver	Lead	Zinc
At 2.20 amperes	1.7% assaying.........	31.2 ozs.	9.8%	8.6%
At 4.00 amperes	2.25% assaying.........	148.5 ozs.	6.1%	45.4%
Blake product A	4.40% assaying.........	92 9 ozs.	58.6%	12.3%
Blake product B	0.65% assaying.........	44.8 ozs.	19.4%	36.3%
	9.00%	....	....	....

Combining the first and third products, we have 6.1% of the weight of concentrates, assaying silver 75.7 ozs., lead 43.10%, zinc 11.2%, showing a silver recovery of 24.4% and lead recovery of 50.9%.

Test B.—Official.—The ore was ground to pass 0.03 in. screen aperture and sized.

+ 0.008 in.	61.78%
— 0.008 in.	38.22%

Each size was fed separately to a wet concentration table and like products combined for the general result. The crude ore used in this test assayed silver, 16.4 ozs.; lead, 4.2%; zinc, 5.9%. Result of wet concentration:

	Weight	Silver	Lead	Zinc
Lead concentrate	4.34% assaying	131.0 ozs.	68.4%	6.87%
Zinc concentrate	3.70% assaying	67.5 ozs.	3.9%	39.4%
Tailings	77.29% assaying	7.2 ozs.	0.3%	4.5%
Loss	14.47%			

The distribution of the metals figured in percentages of the whole being:

	Weight	Silver	Lead	Zinc
Lead concentrate	4.34%	34.6%	68.3%	5.3%
Zinc concentrate	3.70%	15.2%	3.4%	24.7%
Tailings	77.79%	33.3%	5.5%	59.0%
Loss	14.47%	16.9%	22.8%	11.0%

The metal loss appears high, partly accounted for by the very low grade of the raw ore; the tailings for example are quite low in lead and fairly low in silver, if we refer to the assays; figured in percentage, however, 59% of the zinc, 5.5% of the lead and one-third of the total silver in the ore are found in the tailings.

The zinc concentrates were next sized, giving:—

	Before roasting	After roasting
+ 0.008 in.	16.3%	10.0%
+ 0.005 in.	23.7%	39.4%
— 0.005 in.	60.0%	50.6%
	100.0%	100.0%

They were then roasted at a dull red heat for 30 minutes: Loss in weight 6%. The roasted ore was then passed over a Wetherill separator at a current of 120 volts, giving the following result:—

	Weight	Silver	Lead	Zinc
At 0.15 amperes	43.7% assaying	118.8 ozs.	6.9%	25.9%
Non-magnetic	56.3% assaying	29.2 ozs.	1.7%	54.8%

This showed a recovery of the zinc in the roasted ore amounting to 73.17%, or 18% of the zinc in the original ore.

The magnetic product assaying 25.9% zinc can be improved as a smelting ore by grinding finer and retreating it on a magnetic separator.

A portion of the above roasted ore was next treated on the Dings separator, at 120 volts, with the following result:—

	Weight	Silver	Lead	Zinc
At 0.30 amperes	53.0%..................	111.0 ozs.	5.8%	32.3%
Non-magnetic	47.0%..................	33.0 ozs.	1.9%	54.1%

The 53% product assaying 32.3% zinc was next ground to pass a screen of 0.01 in. aperture and again passed through the Dings separator. Result:—

	Weight	Silver	Lead	Zinc
At 0.30 amperes	40.96% assaying.........	....	....	24.60%
Non-magnetic	12.04% assaying.........	....	....	54.20%

Adding this zinc to that obtained in the first pass we have

	47.00 tons.................	@	54.1% zinc
	12.04 tons.................	@	54.2% zinc
Total	59.04 tons.................	@	54.1% znic

To obtain a zinc blende assaying 54% metal from an ore assaying only 5.8% zinc is a satisfactory achievement, largely due no doubt to the docile nature of the ore, the proper treatment of which is clearly indicated in the official tests.

LOT NO. 17—AURORA MINE.

This ore appeared to be a heavy zinc-lead sulphide, practically free from gangue. It was crushed to pass an 0.03 in. screen aperture and assayed: Gold, 0.02 ozs.; silver, 7.3 ozs.; lead, 21.5%; zinc, 33%.

Test A.—Preliminary. Table concentration to remove the galena.

	Weight	Silver	Lead	Zinc
Lead concentrate	12.90% assaying.........	23.0 ozs.	71.6%	8.0%
Zinc concentrate	80.40% assaying........	2.4 ozs.	7.8%	39.7%
Loss	6.70%.................	....	....	....
	100.00%	....	....	....

The zinc product was treated on a Wetherill separator at 120 volts and 4 amperes and given two passes through the machine. Result:—

	Weight	Silver	Lead	Zinc
Magnetic	55.00% assaying........	1.20 ozs.	2.80%	48.80%
Non-magnetic	25.40% assaying.........	6.10 ozs.	18.80%	16.80%
	80.40%................	....	....	....

The magnetic portion assaying 48.8% contains 81.3% of the zinc present in the original ore.

Test B.—The crude ore was roasted at 0.03 in. size, giving a loss in weight of 18% and then was treated on the Wetherill separator with 120 volts current. Result:—

	Weight	Silver	Lead	Zinc
At 1.0 ampere	8.64% assaying.........	4.50%	12.4%	40.00%
At 3.0 ampere	56.79% assaying.........	2.50%	6.4%	49.40%
Non-magnetic	34.57% assaying.........	13.40%	47.8%	8.60%
	100.00%	....	....	....

The 1.0 ampere current was evidently too strong and removed considerable zinc. The product magnetic at 3.0 amperes assaying 49.4% zinc contains 81.3% of the zinc in the roasted ore. The product non-magnetic at 3 amperes, if mixed with the 1 ampere product would give a smelting ore assaying 12.3% silver, 24.8% lead and 14.8% zinc, a saving of 78.8% of the silver and 56.6% of the lead.

Test C.—In this test the raw ore was passed over the Wetherill separator and the ferruginous blende picked out at 4 amperes. The non-magnetic product at the higher amperage was next passed over the Blake electrostatic machine to separate the lead. Result:—

	Weight	Silver	Lead	Zinc
At 2.20 amperes	1.60% assaying.........	....	....	....
At 4.00 amperes	52.90% assaying.........	1.00%	3.0%	51.4%

Tue non-magnetic product, 45.50%, was next passed over the Blake electrostatic machine with the following result:—

	Weight	Silver	Lead	Zinc
Product A	27.3% assaying..........	16.00 ozs.	45.80%	14.60%
Product B	18.2% assaying..........	11.20 ozs.	36.40%	8.40%

The product assaying 51.4% zinc shows a recovery of 82.4% of the zinc in the original ore. The Blake treatment does not help the non-magnetic portion of the product. This in fact simplifies the process, which works out as follows:—

A. Dry the ore and crush to pass an 0.03 in. screen aperture.

B. Pass through high intensity machine at 4 amperes, which gives an 82% zinc recovery in a blende assaying 51% zinc and a non-magnetic tailing, containing 82% of the silver and 96% of the lead in the crude ore, which is an excellent smelting ore.

Test D.—Official. The ore reduced to pass 0.03 in. screen aperture, assayed: silver 7 ozs.; lead, 20.5%; zinc, 33.7%. It sized as follows:—

+ 0.008 in.	62.06%
+ 0.005 in.	8.17%
— 0.005 in.	29.77%
	100.00%

The method of treatment to include wet concentration to separate a lead concentrate, a zinc concentrate and a silicious, valueless tailing. The zinc concentrate to be roasted and treated on high intensity machines.

The above sizes were fed separately to the concentrating table, and like products combined for the general result.

	Weight	Silver	Lead	Zinc
Lead concentrate	25.36% assaying	20.6 ozs.	64.6%	11.9%
Zinc concentrate	56.08% assaying	1.5 ozs.	4.7%	48.2%
Tailings	14.63% assaying	3.0 ozs.	7.0%	19.2%
Loss	3.93%	....	....	....
	100.00%	....	....	....

The galena concentrates very nicely and can easily be kept below the smelter limit of 10% zinc. The lead saving is practically 80%, carrying 74.63% of the silver in the crude ore. The zinc saving, however, is only 80%, due to a heavy tailing loss that in working on a large scale can be avoided by cutting out a zinc middling for re-treatment.

The 56.08% of zinc concentrates were then sized and roasted.

	Before roasting	After roasting
+ 0.008 in.	59%	60.00%
+ 0.005 in.	15%	14.00%
— 0.005 in.	26%	26.00%

The ore was next charged into a red hot muffle, the heat sustained and the charge drawn at 35 minutes. The cooled ore was then passed over the Wetherill separator. The loss in roasting amounted to only 1.50%.

Weight		Silver	Lead	Zinc
At 0.30 amperes	7.50% assaying	5.20 ozs.	9.00%	36.1%
At 3.00 amperes	84.80% assaying	1.60 ozs.	3.20%	51.3%
Non-magnetic	7.70% assaying	1.60 ozs.	3.20%	30.7%

The 84.8% zinc product removes 89.57% of the zinc in the roasted ore or 71.27% of the zinc in the original ore. The much simpler and cheaper process in Test C., gives a zinc recovery of 82.4%, assaying 51.4% zinc, while Test D. gives only 71.27%, assaying 51.3% zinc. The former is therefore the correct method of treatment.

The ore represented by Lot No. 17 is about the simplest ore to treat of any so far examined in this series.

LOT NO. 18—BLUE BELL MINE.

This ore is from the Blue Bell mine, near Ainsworth, and consists of massive pyrrhotite, zinc blende, galena, pyrites and quartz in a limestone gangue. For the purpose of preliminary tests the ore was crushed to pass a 0.03 in. screen aperture. It assayed: Gold 0.02 ozs.; silver, 3.80 ozs.; lead, 12.80%; and zinc, 14.60%.

Test A.—Preliminary. Wet concentration on tables, giving the following products.

Weight	Silver	Lead	Zinc
Lead concentrate 12.40% assaying..........	13.6 ozs.	70.6%	1.90%
Zinc concentrate 72.60% assaying..........	2.5 ozs.	4.5%	18.3%
Tailings 11.20% assaying..........	2.0 ozs.	5.4%	5.2%
Loss 3.80% assaying..........	....	....	....
100.00%	....	....	....

This shows a recovery by wet concentration of 68.4% of the lead and 44.3% of the silver in a lead concentrate and 91% of the zinc combined with pyrrhotite and pyrites, in a middling product called zinc concentrate. The zinc saving appears high on account of the ratio of concentration being so low, the zinc tenor of the concentrates being only 3.7% higher than that of the crude ore. The lead concentration is good, but could be improved by finer crushing.

The zinc concentrate was next roasted and passed over a Wetherill separator, with a current of 120 volts. Results were as follows:

Weight	Silver	Lead	Zinc
At 0.30 amperes 39.47% assaying..........	3.00 ozs.	5.00%	5.8%
At 0.60 amperes 9.20% assaying..........	4.60 ozs.	12.4 %	26.9%
At 2.20 amperes 31.60% assaying..........	3.50 ozs.	13.4 %	31.4%
Non-magnetic 19.73% assaying..........	8.20 ozs.	38.8 %	2.1%
100.00%	....	....	....

This test cannot be considered satisfactory, inasmuch as the highest zinc product obtained carries less than 32% metal.

Test B.—A portion of the raw ore, passing 0.03 in. screen aperture was next passed over the Wetherill machine at 120 volts, giving the following results:

Weight	Silver	Lead	Zinc
At 0.45 amperes 15.15% assaying..........	2.0 ozs.	0.20%	1.9%
At 2.10 amperes 15.15% assaying..........	1.0 ozs.	0.40%	8.2%
At 4.00 amperes 30.30% assaying..........	1.2 ozs.	0.20%	37.5%
Non-magnetic 39.40% assaying..........	6.2 ozs.	30.60%	4.0%
100.00%	....	....	....

This is quite a remarkable separation of pyrrhotite and galena from zinc blende, without either preliminary concentration or roasting. The raw ore used in this test assayed: silver, 3.8 ozs., lead 12.8%, zinc 14.6%. The products produced up to 2.10 amperes amounting to 30.30% of the weight of the crude ore can be rejected as worthless. The product magnetic at 4.0 amperes amounting to 30.3% of the total weight contains 77.8% of the zinc present in the ore, while the non-magnetic product at 4.0 amperes carries 94% of the lead and 64.2% of the silver.

Comparing these results with wet concentration we have for the lead,

	Tons	Silver	Lead	Saving Silver	Saving Lead
Wet, lead concentrate	12.4	13.6 ozs.	70.6%	44.3%	68.4%
Dry, lead concentrate	39.4	6.2 ozs.	30.6%	64.2%	94.0%

And for the zinc concentration,

	Tons	Silver	Lead	Zinc	Saving Zinc
Wet, zinc concentrate	72.6	2.5 ozs.	4.5%	18.3%	91.0%
Dry, zinc concentrate	30.3	1.2 ozs.	0.2%	37.5%	77 8%

The simple process of crushing dry and magnetic separation on high intensity machines appears best for the zinc, yet the tenor of the product is perhaps too low (37.5%). Consequently this zinc product would require roasting and further treatment on low intensity magnetic separators, but as it will be observed it only means roasting 30% of the original or crude ore, which is not a very serious matter.

The ferruginous blende in the Blue Bell ore carries a large percentage of iron; pure crystals of the mineral assayed 12.8% metallic iron.

Test C.—Official. The ore was ground to pass a screen of 0.02 in. apertures and gave the following analysis; Gold, 0.02 ozs.; silver, 4.40 ozs.; lead, 12.9%; zinc 14.4%.

It was next sized, giving the following result:—

+ 0.008 in.	51.35%
+ 0.005 in.	16.22%
— 0.005 in.	32.43%
	100.00%

Each size was treated separately on a concentration table and like products combined for the general result given below:

Weight	Silver	Lead	Zinc
Lead concentrate 20.00% assaying	11.0 ozs.	55.2%	5.1%
Zinc concentrate 53.15% assaying	1.7 ozs.	2.0%	21.0%
Tailings 25.23% assaying	1.1 ozs.	2.0%	7.8%
Loss 1.62%	....	....	......
100.00%	....	....	....

Giving the following distribution of metals in percentage of the whole:

Tons	Silver	Lead	Zinc
20.00, lead concentrate	50.0%	85.6 %	7.08%
53.15, zinc concentrate	20.5%	8.24%	77.51%
25.23, tails	6.3%	3.90%	13.66%
1.62, loss	23.2%	2.26%	1.75%
100.00	100.0%	100.00%	100.00%

The fine grinding shows an increase in both the lead and silver saving in the lead concentrates when compared with test A., as well as a lower zinc saving, but the concentrate is of higher zinc tenor. The results all around are therefore an improvement on test A. On account of the pyrrhotite and pyrites, however, it is impossible to make even a fair grade of zinc concentrate by the wet process alone.

The following sizing tests were then made on the zinc concentrates:

	Before roasting	After roasting
+ 0.008 in.	46.0%	54.0%
+ 0.005 in.	17.0%	23.0%
— 0.005 in.	37.0%	23.0%
	100.0%	100.0%

The zinc concentrates were roasted for 40 minutes at a dull red heat, the loss in weight being 5.5%.

The roasted ore was then treated on a Wetherill separator, with a current of 120 volts. The results were as follows:

Weight	Silver	Lead	Zinc
At 0.15 amperes 55.60% assaying..........	3.80 ozs.	2.10%	12.80%
At 0.30 amperes 9.40% assaying..........	2.20 ozs.	1.40%	34.8 %
At 0.45 amperes 8.00% assaying..........	1.60 ozs.	0.90%	39.1 %
At 2.10 amperes 23.40% assaying..........	1.20 ozs.	0.30%	41.3 %
Non-magnetic 3.60% assaying..........	1.40 ozs.	0.50%	14.6 %
100.00%	....	....	....

This test resembles a mixing of the various minerals rather than a separation, due no doubt, to the amount of highly magnetic material associated with the zinc blende, itself so highly magnetic that 34.8% zinc came off with the iron at 0.30 amperes; showing clearly that the pyrrhotite, and other highly magnetic ores, should first be removed from the concentrates, and the resulting blende if not then of sufficiently high zinc tenor, could be roasted and passed over low intensity machines. To make sure that the roasting was sufficient, a portion of the product magnetic at 2.10 amperes and assaying 41.3% zinc was ground to pass an 0.008 in. screen aperture, subjected to a further roast, and then passed over a magnetic separator. Result:

At 0.030 amperes 10.00% of weight assaying, zinc 27.50%
Non-magnetic at 3. amperes 90.00% of weight assaying, zinc 44.50%

This raises the value only about 3%, proving the first roast was practically correct. The products assaying 34.8% zinc and 39.1% zinc were mixed, and ground to pass a screen of 0.008 in. apertures and given another magnetic treatment to see if the zinc could be separated from the iron. Result:

At 0.15 amperes 33.60% of weight assaying, zinc 26.60%
Non-magnetic At 0.15 amperes 66.50% of weight assaying, zinc 38.40%

This shows that the zinc blende is so highly magnetic that it cannot be effectively separated from the iron after roasting.

Test D.—Official. A portion of the zinc concentrates from Test C., assaying 21% zinc and forming 53.15% of the ore concentrates, was, after drying, passed through a Wetherill separator, at 120 volts, giving the following results:

At 0.15 amperes 9.16% of weight assaying, zinc 0.60%
At 0.30 amperes 9.54% of weight assaying, zinc 1.30%
At 1.05 amperes 16.03% of weight assaying, zinc 2.40%

34.73%

Thus 34.73% of the weight of the concentrate is removed as worthless material before roasting, eliminating the highly magnetic minerals and reducing the amount of mineral to be roasted by 34.73%. The concentrate thus freed from highly magnetic mineral was given a strong roast which caused a loss of weight of 6.36% of the whole. When cooled the roasted ore was passed over a Wetherill separator, with 120 volts current, and gave the following results:

At 0.15 amperes	15.65%	of weight assaying, zinc	8.0%
At 0.30 amperes	0.61%	of weight assaying, zinc	24.3%
At 3.00 amperes	45.65%	of weight assaying, zinc	44.0%
Non-magnetic	3.36%	of weight assaying, zinc	3.6%
	65.27%		

By this method of procedure the zinc value of the product magnetic at 0.15 amperes is reduced from 12.8% zinc in test C. to 8% zinc. That product magnetic at 0.30 amperes from 34.8% zinc in test C. to 24.3% the non-magnetic product from 14.6% zinc to 3.6%, while the tenor of the zinc product is raised from 41.3% to 44.0%, which represents the highest zinc product that can be commercially obtained from Blue Bell ore by any of the methods of treatment herein outlined,

For greater clearness I tabulate the steps in this treatment and the results obtained.

Taking of the original ore		100 tons.
First. We obtain as lead concentrate	20.00 tons	
Second. Zinc concentrate	53.15 tons	
Third. Tailings and loss	26.85 tons	
		100 tons.

The lead concentrate is ready for sale to the lead smelters, while the zinc concentrate must be brought up to a higher tenor.

Taking the zinc concentrate amounting to 53.15% of the original ore or		53.15 tons
Fourth. Removed by magnetic treatment of the raw concentrate	18.46 tons	
Fifth. Weight lost in roasting	2.21 tons	
Sixth. Removed by magnetic treatment of roasted ore	9.77 tons	
Seventh. Final zinc product	22.71 tons	
		53.15 tons

100.00 tons of ore @ 14.4% zinc=28,800 lbs. zinc.
22.71 tons of ore @ 44.0% zinc=19,984 lbs. zinc.

A saving of 69.4% of the zinc in the crude ore, which I consider very satisfactory, and consequently the method of treatment tabulated above, appears to me as the best.

In my report on the Ainsworth mines, written before any of these tests were made, it was necessary for the purpose of estimating the possible zinc production of the district, to forecast the treatment of the Blue Bell ore, and the losses incident thereto. I assumed a saving of zinc in concentrates by wet treatment of 77% against 77.51% actually recovered in test C. I estimated the ultimate zinc recovery as a 50% zinc ore, at 65% of the zinc contained in the crude ore. The saving shown in test C. is 69.4% of 44% ore, which appears to be the highest commercial product obtainable from the samples sent to Denver for treatment.

LOT NO. 19—ST. EUGENE MINE.

This is a low grade zinc concentrate, produced from the Saint Eugene ore. It consists of zinc blende, garnets, pyrites, galena and silica, and assayed silver, 5.70 ozs., lead, 9.70%, zinc, 20.20%.

Test A.—Preliminary. The concentrates were passed over a Wetherill separator to remove the garnets and the other highly magnetic products and lastly to remove the zinc blende at a high amperage. The current used was 120 volts. The results were as follows:—

Weight	Silver	Lead	Zinc
At 0.30 amperes 11.30% assaying..........	4.50 ozs.	6.80%	5.20%
At 1.95 amperes 31.40% assaying..........	4.20 ozs.	6.40%	6.60%
At 4.00 amperes 43.60% assaying..........	5.00 ozs.	7.20%	34.90%
Non-magnetic 13.70% assaying..........	14.00 ozs.	31.60%	11.80%

The zinc product magnetic at 4 amperes contains 75% of the zinc in the ore, but the grade is too low and it also contains too much lead for good marketable zinc ore.

Test B.—The ore was roasted before submitting it to treatment on the Wetherill separator; current, 120 volts. The results were as follows:—

Weight	Silver	Lead	Zinc
At 0.30 amperes 37.44% assaying..........	7.80 ozs.	12.8%	10.7%
At 0.45 amperes 7.02% assaying..........	8.50 ozs.	14.3%	30.3%
At 3.50 amperes 50.72% assaying..........	4.50 ozs.	8.1%	29.1%
Non-magnetic 4.82% assaying..........	16.30 ozs.	37.2%	7.7%

This test shows that roasting in the presence of so much lead and highly magnetic minerals, simply mixes the zinc ore with the strongly magnetic ores. It is therefore evident that the galena should be removed, preferably by concentration, as a condition precedent to magnetic work.

Test C.—Official. In an attempt to liberate the minerals the mixed ore was reduced to pass a screen of 0.02 in. apertures and sized:

+ 0.008 in.	29.00%
+ 0.0055 in.	12.50%
— 0.0055 in.	58.50%
	100.00%

Each size was passed separately over a concentrating table and similar products combined for a general result:—

Weight	Silver	Lead	Zinc
Lead concentrate 10.52% assaying..........	19.8 ozs.	41.4%	8.8%
Zinc concentrate 66.46% assaying..........	4.4 ozs.	6.2%	23.0%
Tailings 19.37% assaying..........	3.2 ozs.	4.2%	13.2%
Loss 3.65% assaying..........	....	....	....
100.00%	....	....	....

This concentration test is not satisfactory, inasmuch as the zinc concentrate still contains 6.2% lead, while the tailings carry 13.2% zinc. This ore should evidently have been crushed finer prior to concentration.

The lead concentrate shows a saving of 36.50% of the silver and 44.9% of the lead in the original ore.

The zinc concentrate accounts for 75.6% of the zinc in the original ore (concentrate).

The zinc concentrate was then divided into two parts, one of which was passed over the Wetherill separator in the raw state, just as it came from the concentrator. The other was given a preliminary roast, which reduced the weight of the concentrates 9%. Wetherill results on roasted ore at 120 volts, were:—

Weight	Silver	Lead	Zinc
At 0.15 amperes 32.09% assaying..........	5.6 ozs.	9.2%	11.1%
At 1.50 amperes 44.30% assaying..........	4.2 ozs.	7.6%	23.1%
At 4.00 amperes 21.40% assaying..........	2.6 ozs.	3.0%	47.8%
Non-magnetic tails 1.40% assaying..........	6.8 ozs.	11.2%	8.3%
100.00%	....	....	....

These results are practically the same as those in Test B., the unfavourable result being due to the lead present with the zinc. It is true that a zinc ore is obtained assaying 47.8% zinc and 3% lead, but this product only accounts for 26% of the zinc in the original ore.

The second portion of the zinc concentrate was passed over the Wetherill separator, without roasting. Current 120 volts. The results were as follows:

Weight	Silver	Lead	Zinc
At 0.15 amperes 8.8% assaying............	3.80ozs.	5.9%	3.20%
At 2.10 amperes 36.2% assaying............	2.8 ozs.	5.0%	7.10%
At 4.00 amperes 47.2% assaying............	3.4 ozs.	5.6%	40.80%
Non-magnetic 7.8% assaying............	9.2 ozs.	18.0%	12.80%

This test is fairly satisfactory, inasmuch as 56% of the zinc in the original ore is recovered in a product assaying 40.8% zinc, which product can be raised to 50% zinc by roasting and treating it on a low intensity magnetic separator. It will be noticed that the non-magnetic tailings assay 9.2 oz. silver and 18% lead. This product can be added to the lead concentrate produced from the tables, the mixture forming a very desirable lead smelting ore.

The Saint Eugene concentrate is by no means difficult to treat. The proper treatment for this ore, as indicated by the foregoing tests is:—

A. Crush to about 40 mesh, size and concentrate;

B. Dry the zinc concentrates and treat them raw on high intensity separator for 40% zinc blende, and 55% saving.

C. Roast the 40% zinc blende and treat on low intensity magnetic separator for 50% zinc blende and 45 to 50% saving of the zinc in the original ore.

LOT NO. 20.

This ore was shipped from Marysville in response to a request for an average sample of the mixed lead-zinc ores of the Sullivan mine. The ore received, however, was a mixture of zinc blende, siderite and quartz, the minerals very finely blended, and bore no resemblance to the ores observed by the Commission in the Sullivan mine, which are of peculiar and unmistakeable character. Consequently, the tests made on the ore received are not reported.

REPORT

ON THE

METHODS OF ASSAYING

BY

HENRY HARRIS

AND

HENRY E. WOOD

1906.

METHODS OF ASSAYING.

The assaying and analysis of the samples of ore taken in the field was done by Mr. Henry Harris, assistant superintendent of the Hall smelter, Nelson, B.C. The ana'ytical work in connection with the samples obtained in the ore tests at Denver was done by Mr. Henry E. Wood. Some special analytical work, which is referred to in the reports, was performed by Mr. W. George Waring, of Webb City, Mo.

In all cases, the determinations of lead in the ores were made by a volumetric method, the common fire assay being rejected because of its well recognized inaccuracy. The determinations for zinc by Mr. Harris and Mr. Wood were made by the Low method, which is a scientific, commercial method, and is sufficiently accurate for general technical purposes when intelligently executed. The assays made by Mr. Waring were performed by his own excellent method.

The methods employed by Messrs. Harris and Wood are described in the following reports:

REPORT OF HENRY HARRIS.

W. R. INGALLS, Esq.,
New York.

Dear Sir:—

The methods followed in assaying samples delivered to me by Mr. Philip Argall, in connection with the work of the Zinc Commission, are described below:

Silver Assay.—This of necessity varied much with the nature of the sample. In every case the assay was made in triplicate. For the majority of the samples 3 assay tons of the flux specified below was used with 0.2 assay tons of the ore, nitre or flour being added to obtain a lead button of about 25 grammes. The fusion was made at a moderate heat and poured *very hot*, occupying in all probably about 40 minutes.

Flux—Litharge ....9 parts
Sod. Bicarb.........3 parts
Borax glass ....1 part
Salt cover

Lead Assay.—One half to one gram of the sample was decomposed with concentrated nitric acid and heated until action ceased. Add 8 c.c. sulphuric acid and boil until heavy SO_3 fumes are evolved, and continue heating until drop under cover glass is of an oily consistency. Cool. Add water and boil. Cool and allow to settle for 45 minutes. Filter. Wash residue well with cold distilled water. Wash contents of filter paper into original beaker, add 5 grams ammonium chloride and a few crystals of sodium acetate. Keep bulk of solution low and digest until all lead sulphate is dissolved. Dilute with hot water to about 250 c.c. and titrate with ammonium molybdate, using as an indicator a freshly prepared solution of tannin.

Zinc Assay.—All determinations were made in duplicate. Some of the first samples were checked by means of a method involving the elimination of manganese through the agency of bromine and the use of sodium sulphate. As the results obtained by this method checked those obtained by the shorter Colorado method when carefully handled, the bulk of the samples were determined by that method as follows:—

Treat one half to one gram in a casserole with 10 c.c. of water, which must be well mixed with the ore, and afterwards add 25 c.c. of a saturated solution of potassium chlorate in nitric acid. Evaporate to dryness at a *low temperature* and dehydrate. Before dehydration is complete the assay should be examined for sulphur globules, and if such are present the assay should be rejected. (If much lead is present the addition of 5 grams of NH_4Cl before evaporation is of advantage in preventing caking and if the evaporation is carried on carefully, there need be no fear of loss of zinc by volatilization as chloride.)

To the solid mass add 5 grams of NH_4Cl, 15 c.c. NH_4HO and 25 c.c. water. Boil for a few minutes. Filter off insoluble oxides, etc. (which should include all the manganese). Wash with hot water. Dilute. Neutralize with HCl and add 10 c.c. in excess. Titrate hot with standard solution of potassium ferrocyanide, using uranium acetate as an indicator. I titrate hot, but this is immaterial as long as the standardization is performed under the same conditions. Of course, the reaction in the cold being very much more slow, a longer time must necessarily be taken over the titration. In using this method I find it very necessary for close work to standardize under exactly the same conditions as exist in the actual assay, even to the quantity of zinc as far as possible. The only interfering impurities I found in these ores were copper and manganese. The elimination of the former was effected by agitation with pure granulated lead, and the latter of course was filtered off together with the iron and insoluble matter.

Insoluble Matter.—Determined in the case of sulphides by decomposing slowly one half gram in a casserole with potassium chlorate and nitric acid. Oxidized ores were treated with aqua regia. After baking for a few minutes, digested with HCl. The filtered residue was well washed with hot water and a solution of ammonium acetate, ignited and weighed as insoluble matter.

Iron.—The solution from the above was heated to boiling and the iron precipitated hot with NH_4HO. The precipitate was redissolved in HCl and the solution whilst hot was reduced with stannous chloride. After cooling, the excess of stannous chloride was taken up by the addition of a considerable quantity of a solution of $HgCl_2$ and the iron estimated by titration with bichromate of potassium.

Yours respectfully,

Nelson, B.C.
Nov. 31, 1906.

HENRY HARRIS.

The following are the results of the assays of samples reported by Mr. Harris:

SAMPLES PER P. ARGALL.

No.	Silver Oz.	Lead %	Zinc %	Iron %	Insoluble %
1	2.0	1.9	32.5	....	....
2	71.0	73.5	....	....	....
3	49.2	45.0	7.0	....	....
4	0.8	....	21.9	....	....
5	....	2.0	14.9	....	....
6	....	....	4.6	21.8	....
7	41.3	?	....	....	....
8	1.0	....	31.6	....	....
9	2.2	nil	48.5	....	....
10	2.2	trace	40.0	....	....
11	2.1	1.0	46.1	....	....
12	14.6	2.6	46.3	....	....
13	13.0	5.5	30.4	....	....
14	20.9	13.1	31.7	....	....
15	2.2	1.0	29.0	....	....
16	1.5	1.2	25.7	....	....
17	2.9	0.8	26.7	....	....
18	1.5	2.2	15.7	....	....
19	1.7	1.0	17.4	....	....
20	1.4	1.3	20.2	....	....
21	1.6	0.8	31.0	13.8	....
22	2.3	1.3	38.5	13.2	....
23	1.1	nil	29.2	13.2	....
24	1.4	....	20.5	....	....
25	1.6	0.7	19.2	....	....
26	28.5	0.7	23.9	5.3	49.0
27	50.8	0.6	19.0	6.8	53.0
28	12.0	4.4	18.9	6.1	50.4
29	11.5	22.9	12.0	7.4	36.1
30	111.9	78.6	3.0	0.8	4.6
31	3.1	....	27.0	....	....
32	1.9	nil	28.2	....	....
33	2.6	nil	24.6	....	....
34	3.0	nil	39.8	....	....
35	15.1	nil	55.6	....	....
36	33.2	18.8	42.6	....	....
37	12.6	5.2	42.2	....	....
38	113.6	67.9	12.1	....	....
39	57.4	26.2	29.6	....	....
40	28.4	6.7	27.8	....	....
41	58.5	10.7	26.6	....	....
42	26.1	5.1	31.0	....	....
43	77.7	39.7	17.9	....	....
44	66.4	10.3	35.4	....	....

No.	Silver Oz.	Lead %	Zinc %	Iron %	Insoluble %
45	44.9	1.0	36.2		
46	62.3	21.2	32.8		
47	53.4	5.1	41.0		
48	18.2	36.7	32.6		
49	10.6	7.5	38.0		
50	19.4	6.1	34.8		
51	4.0	21.8	7.2		
52	1.9	nil	13.9		
53	0.8	2.4	5.9		
54	1.3	4.9	27.5		
55	4.8	23.4	19.5		
56	0.5	0.4	31.9		
57	2.1	11.3	10.9		
58	1.3	2.0	34.5		
59	2.6	15.7	21.1		
60	24.6	3.2	22.7		
61	0.7	nil	23.0		
62	34.1	6.0	18.0		
63	74.4	17.4	19.6		
64	64.	0.5	nil.		
65	2.7	5.5	16.8		
66	36.3	2.6	15.4		
67	190.8	12.5	24.7		
68	18.3	5.0	11.4		
69	4.3	1.0	13.0		
70	16.7	4.9	12.0		
71	102.6	19.4	19.2		
72	23.4	6.1	16.1		
73	37.1	8.0	13.6		
74	7.2	7.7	34.2		
75	13.2	12.4	29.2		
76	10.4	8.7	26.2		
77	10.3	10.0	24.6		
78	32.3	13.4	38.4		
79	14.6	6.8	42.9		
80	122.6	80.8	trace		
81	1.0	nil	54.6		
82	2.5	2.1	46.6		
83	2.1	1.4	46.6		
84	74.7	81.2	5.7		
85	196.5	26.8	38.4		
86	2.3	trace	46.7		
87	6.7	13.7	28.5		
88	3.6	1.7	27.8		
89	15.0	20.7	23.6		
90	16.2	nil	41.0		
K 1	5.2	3.9	49.2		
K 2	3.8	3.4	17.5		
K 3	5.0	4.2	50.8		
K 4	6.0	4.5	28.8		
K 5	4.5	3.3	50.5		
K 6	3.5	3.3	11.2		
BB 1				12.8	

SAMPLES PER DR. BARLOW.

No.	AU Oz.	AG Oz.	PB %	ZN %	FE %	SiO_2 %	CU %	CaO %
1	trace	2.3		29.1	7.6	16.0	2.5	
2	0.01	0.6	nil	35.0	12.0	34.0		nil
3	nil	trace		40.0	5.5	33.0	nil	
4	0.03	15.1	8.8	29.2	3.2	20.0		8.9

SPECIAL SAMPLES OF LIMESTONE, PER P. ARGALL.

No.	SiO_2 %	CaO %	MgO %
1	....	53.0	....
2	0.8	53.4	nil
3	....	50.7	....
4	3.0	51.8	nil

REPORT OF HENRY E. WOOD.

W. R. INGALLS, ESQ.,
New York.

DEAR SIR,—

The methods of analysis employed for the determination of samples obtained in the ore tests of the Zinc Commission were as follows:—

Lead.—The ordinary ammonium molybdate method was used. The ore was dissolved in aqua regia, evaporated almost to dryness, sulphuric acid added, and heated until all nitric and hydrochloric acids were driven off and fumes of sulphuric anhydride were coming off freely. It was then allowed to cool, taken up with cold water, boiled, filtered and washed with dilute sulphuric acid. The paper containing the lead sulphate, etc. was placed in a beaker, a hot dilute solution of ammonium acetate added, heated to boiling and titrated with standard ammonium molybdate, using a dilute solution of tannic acid on a spot plate as an indicator.

Zinc.—A modification of Low's method was used. The ore was dissolved in nitric acid, a little potassium chlorate was added and the solution evaporated to dryness. It was taken up with a strongly ammoniacal solution of ammonium chloride, (and in case much manganese was present bromine water was added); this was boiled a few moments, filtered and washed with a hot dilute solution of ammonium chloride rendered slightly alkaline by ammonia. The filtrate was acidified with hydrochloric acid and boiled with a little granulated lead, to precipitate the small amount of copper present. The boiling was continued until all bromine was removed. It was then titrated hot with standard potassium ferrocyanide, using a saturated solution of uranium acetate on a spot plate as an indicator.

Iron.—The filtrate from the lead sulphate (see lead determination) was boiled with metallic aluminum until all the iron was reduced. It was then diluted, cooled, filtered, washed and titrated with a standard solution of potassium permanganate.

Insoluble Matter.—The ore was digested with aqua regia and evaporated slowly to dryness. It was then taken up with dilute hydrochloric acid and a dilute solution of ammonium chloride added. It was then boiled, filtered, washed, ignited and weighed.

Sulphur.—The ore was fused with a mixture of sodium carbonate and potassium nitrate. The fused mass was dissolved in hot water, filtered and washed. The filtrate was slightly acidified with hydrochloric acid, heated to boiling and an excess of barium chloride added. After settling, the barium sulphate was filtered, washed, ignited and weighed, and the sulphur contents calculated.

Gold and Silver.—Gold assays were made by the crucible method and the silver assays were all made by the ordinary scorification method.

Yours respectfully,

HENRY E. WOOD.

DENVER, COLO.,

March 29, 1906.

APPENDICES.

SYNOPSIS OF MINING LAWS OF BRITISH COLUMBIA.

The Lead Bounty.

SYNOPSIS OF MINING LAWS OF BRITISH COLUMBIA.

The following is from a publication of the Department of Mines of British Columbia.

The mining laws of British Columbia are very liberal in their nature and compare favourably with those of any other part of the world. The terms under which both lode and placer claims are held are such that a prospector is greatly encouraged in his work, and the titles, especially for mineral claims and hydraulic leases, are absolutely perfect. The fees required to be paid are as small as possible, consistent with a proper administration of the mining industry, and are much lower than those of the other Provinces of Canada or the mineral lands under Dominion control. Provision is also made for the formation of mining partnerships practically without expense, and a party of miners is enabled to take advantage of these sections of the Acts and work their claims together, without the trouble or expense of forming a joint stock company.

Considering the success that has characterised alluvial mining on a 'arge scale in British Columbia, the rentals for hydraulic leases are particularly low. It will be found on reference to most of the Australian colonies and Natal, that the rentals are, in most instances, eight times as much as in this Province, while the areas permitted are generally much smaller. The period for which leases are granted is practically the same. On a lode mine of 51 acres the expenditure of $500, which may be spread over 5 years, is required to obtain a Crown grant, and surface rights are obtainable at a small figure, in no case exceeding $5 per acre.

The following synopsis of the mining laws will be found sufficient to enable the miner or intending investor to obtain a general knowledge of their scope and requirements; for particulars, however, the reader is referred to the complete Acts, which may be obtained from the King's Printer, Victoria, B.C.

FREE MINERS' CERTIFICATES.

Any person over the age of 18, and any joint stock company, may obtain a Free Miner's Certificate on payment of the required fee.

The fee to an individual for a Free Miner's Certificate is $5 for one year. To a joint stock company having a capital of $100,000, or less, the fee for a year is $50; if capitalised beyond this, $100.

All these certificates expire at midnight on the 31st of May in each year. Certificates may be obtained for any part of a year, terminating on the 31st May, for a proportionally less fee.

The possession of this certificate entitles the holder to enter on all lands of the Crown, or other lands on which the right to so enter is reserved, and prospect for minerals, locate claims and mine.

A free miner can only hold, by location, one mineral claim on the same vein or lode, but may acquire others by purchase. In the case of placer claims only one can be held by location on each creek, ravine, or hill, and not more than two in the same locality, only one of which shall be a "creek" claim.

In the event of a free miner allowing his certificate to lapse, his mining property (if not Crown-granted) reverts to the Crown, but where other free miners are interested as partners or co-owners the interest of the defaulter becomes vested in the company continuing co-owners or partners, pro rata, according to their interests.

It is not necessary for a shareholder, as such, in an incorporated mining company, to be the holder of a certificate.

MINERAL CLAIMS.

Mineral claims are located and held under the provisions of the "Mineral Act."

A mineral claim is a rectangular piece of ground not exceeding 1,500 feet square. The angles must be all right angles unless the boundaries, or one of them, are the same as those of a previously recorded claim.

No special privileges are allowed for the discovery of new mineral claims or districts.

A mineral claim is located by erecting three legal posts, which are stakes having a height of not less than four feet above ground and squared for four inches at least on each face for not less than a foot from the top. A tree stump so cut and squared also constitutes a legal post.

The "Discovery post" is placed at the point where mineral in place is discovered.

Nos. 1 and 2 posts are placed as near as possible on the line of the ledge or vein, shown by the discovery post, and mark the boundaries of the claim. Upon each of these three posts must be written the name of the claim, the name of the locator and the date of location. On No. 1 post, in addition, the following must be written:—"Initial post. Direction of post No. 2, [*giving approximate compass bearing*];——feet of this claim lie on the right, and——feet on the left of the line from No. 1 to No. 2 posts."

The location line, between Nos. 1 and 2 posts, must be distinctly marked —in a timbered locality by blazing trees and cutting underbrush, and in bare country by monuments of earth or rock not less than two feet in diameter at the base, and at least two feet high—so that the line can be distinctly seen.

Mineral claims must be recorded in the Mining Recorder's Office for the mining division in which they are situated within 15 days from the date of location, one day extra being allowed for each 10 miles of distance from the recording office after the first 10 miles. If a claim is not recorded in time it is deemed abandoned and open for re-location, but if the original locator wishes to re-locate he can only do so by permission of the Gold Commissioner of the District and upon the payment of a fee of $10. This applies also to a claim abandoned for any reason whatever.

Mineral claims are, until the Crown grant is issued, held practically on a yearly lease, the condition of which is that assessment work be performed on the same during such year to the value of at least $100, or payment of such sum be made to the Mining Recorder Such assessments must be recorded before the expiration of the year or the claim is deemed abandoned. If, however, the required assessment work has been performed within the year, but not recorded within that time, a free miner may within 30 days thereafter record such assessment work upon payment of an additional fee of $10. The actual cost of the survey of a mineral claim, to an amount not exceeding $100, may also be recorded as assessment work. If, during any year, work is done to a greater extent than the required $100, any further sum of $100—but not less—may be recorded and counted as further assessments. As soon as assessment work to the extent of $500 is recorded, the owner of a mineral claim is entitled to a Crown grant on payment of a fee of $25, and giving the necessary notices required by the Act. Liberal provisions are also made in the Act for obtaining mill-sites and other facilities in the way of tunnels and drains for the better working of claims.

PLACER CLAIMS.

Placer mining is governed by the "Placer Mining Act," and by the interpretation clause its scope is defined as "the mining of any natural stratum or bed of earth, gravel or cement mined for gold or other precious minerals or stones." Placer claims are of four classes, as follows:—

" 'Creek diggings': any mine in the bed of any stream or ravine:

" 'Bar diggings': any mine between high and low water marks on a river, lake or other large body of water:

" 'Dry diggings': any mine over which water never extends:

" 'Precious stone diggings': any deposit of precious stones, whether in veins, beds or gravel deposits."

The following provisions as to extent of the various classes of claims are made by the Act:—

"In 'creek diggings' a claim shall be 250 feet square: Provided always, that the side lines of each claim shall be measured in the general direction of the water-course or stream:

"In 'bar diggings' a claim shall be:—

"(*a*) A piece of land not exceeding 250 feet square on any bar which is covered at high water, or

"(*b*) A strip of land 250 feet long at high water mark and in width extending from high water mark to extreme low water mark:

"In 'dry diggings' a claim shall be 250 feet square."

The following provision is made for new discoveries of placer mining ground:—

"If any free miner, or party of free miners, discover a new locality for the prosecution of placer mining and such discovery be established to the satisfaction of the Gold Commissioner, placer claims of the following sizes shall be allowed to such discoverers, viz.:—

"To one discoverer, one claim	600 feet in length;
"To a party of two discoverers, two claims, amounting together to	1,000 "

"And to each member of a party beyond two in number, a claim of the ordinary size only.

"The width of such claims shall be the same as ordinary placer claims of the same class: Provided that where a discovery claim has been established in any locality no further discovery shall be allowed within five miles therefrom, measured along the water-courses."

Every placer claim shall be as nearly as possible rectangular in form, and marked by four legal posts at the corners thereof, firmly fixed in the ground. On each of such posts shall be written the name of the locator, the number and date of issue of his free miner's certificate, the date of the location and the name given to the claim. In timbered localities all boundary lines of a placer claim shall be blazed so that the posts can be distinctly seen, underbrush cut, and the locator shall also erect legal posts not more than 125 feet apart on all boundary lines. In localities where there is no timber or underbrush, monuments of earth or rock, not less than two feet high and two feet in diameter at base, may be erected in lieu of the last-mentioned legal posts, but not in the case of the four legal posts marking the corners of the claim.

A placer claim must be recorded in the office of the Mining Recorder for the Mining Division within which the same is situate, within fifteen days after the location thereof, if located within ten miles of the office of the Mining Recorder by the most direct means of travel. One additional day shall be allowed for every ten miles additional or fraction thereof. The number of days shall be counted inclusive of the day upon which such location was made, but exclusive of the day of application for record. The application for such record shall be under oath and in the form set out in the Schedule to the Act. A claim which shall not have been recorded within the prescribed period shall be deemed to have been abandoned.

To hold a placer claim for more than one year it must be re-recorded before the expiration of the record or re-record.

A placer claim must be worked by the owner, or someone on his behalf, continuously, as far as practicable, during working hours. If work is discontinued for a period of 72 hours, except during the close season, lay-over, leave of absence, sickness, or for some other reason to the satisfaction of the Gold Commissioner, the claim is deemed abandoned.

Lay-overs are declared by the Gold Commissioner upon proof being given to him that the supply of water is insufficient to work the claim. Under similar circumstances he has also the power to declare a close season, by a notice in writing and published in the Gazette, for all or any claims in his district. Tunnel and drain licenses are also granted by him on the person applying giving security for any damage that may arise. Grants of right of way for the construction of tunnels or drains across other claims are also granted on payment of a fee of $25, the owner of the claim crossed having the right for tolls, etc., on the tunnel or drain which may be constructed. These tolls, however, are, so far as the amount goes, under the discretion of the Gold Commissioner.

CO-OWNERS AND PARTNERSHIPS.

In both the "Mineral" and "Placer Mining" Acts provision is made for the formation of mining partnerships, both of a general and limited liability character. These are extensively taken advantage of and have proved very satisfactory in their working. Should a co-owner fail or refuse to contribute his proportion of the expenditure required as assessment work on a claim he may be "advertised out" and his interest in the claim shall become vested in his co-owners who have made the required expenditure, pro-rata according to their former interests. It should not be forgotten that if any co-owner permits his free miner's certificate to lapse, the title of his associates is not prejudiced, but his interest reverts to the remaining co-owners.

HYDRAULIC AND DREDGING LEASES.

Leases of unoccupied Crown lands may be granted by the Lieutenant-Governor in Council, upon recommendation of the Gold Commissioner of the District, after location by placing a legal post at each corner of the ground applied for. On the post nearest the placer ground then being worked the locator must post a notice stating the name of the applicant, the location of the ground to be acquired, the quantity of ground and the term for which the lease is to be applied for. Within thirty days application must be made in writing to the Gold Commissioner, in duplicate, with the plan of the ground on the back, and the application must contain the name of each applicant, the number of each applicant's free miner's certificate, the locality of the ground, the quantity of ground, the terms of the lease desired and the rent proposed to be paid. A sum of $20 must accompany the application, which is returned if the application is not granted The term of leases must not exceed 20 years. The extent of ground covered by leases are not in excess of the following:—Creek—half a mile; hydraulic diggings—80 acres; for dredging leases—5 miles; precious stone diggings—10 acres. Under Order in Council the minimum rental for a creek lease is $75 per annum, and for a hydraulic lease, $50 per annum, with a condition that at least $1,000 per annum shall be spent in development. For dredging leases the usual rental is $50 per mile per annum; development work worth $1,000 per mile per annum must be done.

TAXATION OF MINES.

Mineral or placer claims, when Crown-granted, are subject to a yearly tax of 25 cents per acre, but if $200 is spent in work in a year this tax is not levied. A tax of 2% is levied quarterly on all ores and other mineral substances mined in the Province, based upon the net value of such ore at the mouth of the shaft or tunnel, but where ore-producing mines produce under $5,000 in a year half the tax is re-funded, while placer or dredging mines that do not produce a gross value of $2,000 in a year are entitled to a refund of the whole tax. These taxes are in substitution for all taxes on the land and for the personal property tax in respect of sums so produced, so long as the land is only used for mining purposes. By the "Land Act," a royalty of 50 cents per M, board measure, is levied on timber suitable for mining props, a cord of props being considered as 1,000 feet board measure.

COAL AND PETROLEUM PROSPECTING LICENSES.

Any person desiring to prospect for coal or petroleum upon any lands held by the Crown may acquire a license to do so over a rectangular block of land not exceeding 640 acres, of which the boundaries shall run due north and south and east and west, and no side shall exceed 80 chains (1· mile) in length. Before entering into possession of the said land he shall place at the corner of such block a legal stake, or initial post, and shall inscribe thereon his name and the angle represented by such post, thus:—"A. B.'s N.E. corner," or as the case may be, and shall keep posted for 30 days in a conspicuous place upon the said land, and also on the Government Office of the District, as well as publishing it in the B. C. Gazette and in a local newspaper for a like period, a notice of his intention to apply for such prospecting licence.

The application for said licence shall be in writing, in duplicate, and shall contain the best written description possible, with a diagram of the land sought to be acquired, and shall be accompanied with a fee of $100. The application shall be made to the Assistant Commissioner of Lands and Works for the District, and by him forwarded to the Chief Commissioner of Lands and Works, who shall grant such licence—provided no valid protest is substantiated—for a period not to exceed one year, and at the expiration of the first year an extension of such licence may be granted for a second or third year.

Should the licensee discover coal or petroleum upon such land during the period of his licence, and produce satisfactory evidence under oath of the fact, he may obtain from the Lieutenant-Governor in Council, after having had the land properly surveyed, a lease of the said block for a term of five years at an annual rental of 15 cents an acre, and if during the term of such lease, or within three months thereafter, he can show conclusively that he has continuously and vigourously prosecuted the work of coal or petroleum mining, and has fully carried out the terms of such lease, he shall be entitled to purchase the said lands, including the coal or petroleum thereunder, at the rate of $10 per acre, or in the event of the surface rights having been alienated from the Government, he can purchase the coal and petroleum underlying such lands at the rate of $5 an acre: Provided also, that in addition to the rental or purchase price there shall be paid to the Government as a royalty 5c. per ton (2,240 lbs.) of merchantable coal, or 2½ cents per barrel (35 Imp. gallons) of crude petroleum raised or gotten from such land. (See chap. 137 Revised Statutes, 1897, and chap. 37, 1903–4.)

MINING RECORDERS IN OUTLYING DISTRICTS.

Where mineral is discovered in a part of the Province remote from Mining Recorders' offices, so that the provisions of the Act cannot be justly enforced, the miners themselves may, by a two-thirds vote at a meeting for that purpose, appoint a Mining Recorder from among themselves. Such Recorder' can issue free miners' certificates, records of mining property, etc., and such entries will be valid notwithstanding any informality. Under the Act such Mining Recorder shall, as soon as possible, forward a list of the free miners' certificates issued by him, and of records made, to the nearest Gold Commissioner or Mining Recorder, together with the fees required by law therefor.

TABLE OF FEES.

Individual Free Miner's Certificate	$ 5.00
Company Free Miner's Certificate (capital $100,000 or less)	50.00
Company Free Miner's Certificate (capital over $100,000)	100.00
Recording Mineral or Placer Claim	2.50

Recording Certificate of Work, Mineral Claim	2.50
Re-record of Placer Claim	2.50
Recording Lay-over	2.50
Recording Abandonment Mineral Claim	10.00
Recording Abandonment, Placer Claim	2.50
Recording an Affidavit under three folios	2.50
Per folio over three, in addition	30
Records in "Record of Conveyances," same as Affidavits	
Filing Documents	1 00
For Crown Grant of Mineral Rights under Mineral Act	25 00
For Crown Grant of Surface Rights of Mineral Claim	10.00
For every lease under "Placer Mining Act"	5 00

THE LEAD BOUNTY.

Silver-lead mining in British Columbia, except for those ores carrying high silver value, had been at a very low ebb during the two years immediately previous to 1903; in fact, most of the lead mines carrying low silver value suspended operations, the owners claiming they could not be worked at a profit under existing circumstances.

Strong representations were made to the Dominion Government of these facts, and application made for a bounty on lead mined in Canada. This application was granted at the last session of the Dominion Parliament in 1903. The following is the bounty awarded:—

AN ACT TO PROVIDE FOR THE PAYMENT OF BOUNTIES ON LEAD CONTAINED IN LEAD-BEARING ORES MINED IN CANADA.

[*24th October*, 1903.]

1. The Governor in Council may authorize the payment of a bounty of 75 cents per 100 lbs. of lead contained in lead-bearing ores mined in Canada, such bounty to be paid to the producer or vendor of such ores: Provided that the sum to be paid as such bounty shall not exceed $500,000 in any fiscal year: Provided, also, that when it appears, to the satisfaction of the Minister charged with the administration of this Act, that the standard price of pig lead in London, Eng., exceeds £12 10*s*. sterling per ton of 2,240 ℔s., such bounty shall be reduced by the amount of such excess.

2. (1.) Payment of the said bounty may be made from time to time to the extent of 60% upon smelter returns showing that the ore has been delivered for smelting at a smelter in Canada. The remaining 40% may be paid at the close of the fiscal year, upon evidence that all such ore has been smelted in Canada.

(2.) If at the close of any year it appears that during the year the quantity of lead produced, on which the bounty is authorised, exceeds 33,333 tons of 2,000 ℔s., the rate of bounty shall be reduced to such sum as will bring the payments for the year within the limit mentioned in section 1.

3. If at any time it appears, to the satisfaction of the Governor in Council, that the charges for transportation and treatment of lead ores in Canada are excessive, or that there is any discrimination which prevents the smelting of such ores in Canada on fair and reasonable terms, the Governor in Council may authorise the payment of bounty, at such reduced rate as he deems just, on the lead contained in such ores mined in Canada and exported for treatment abroad.

4. If at any time it appears, to the satisfaction of the Governor in Council, that products of lead are manufactured in Canada direct from lead ores mined in Canada without the intervention of the smelting process, the Governor in Council may make such provisions as he deems equitable to extend the benefits of this Act to the producers of such ores.

5. The said bounties shall cease and determine on the 30th June, 1908.

6. The Governor in Council may make regulations for carrying out the intention of this Act.

7. Chapter 8 of the Statutes of 1901, intitled "An Act to provide for the payment of bounties on lead refined in Canada," is repealed.

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