THE DEVELOPMENT OF THE SULLIVAN MINE AND PROCESSES FOR THE TREATMENT OF ITS ORES

By the Staff of The Consolidated Mining and Smelting Company of Canada, Limited (1)

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This paper contains an outline of the history of the Sullivan mine, now owned and operated by the Consolidated Mining & Smelting Company of Canada, Ltd.; some account of the various methods by which the ore has been treated in efforts to solve the metallurgical problems which its complex character presents, and, finally, to give, as concisely as possible, an outline of the present mining and metallurgical practice along with a description of the mining and milling plant at Kimberley and the smelter and refineries at Trail, in British Columbia.

EARLY HISTORY

In May, 1892, Pat Sullivan, John Cleaver and Mike Holland left the Coeur d'Alene country, in the State of Idaho, to seek their fortunes in the Kootenay lake district in British Columbia. They were aided in their venture by James Cronin, afterwards locator and owner of the St. Eugene mine at Moyie.

Two months later the party broke up, Sullivan and Cleaver crossing the mountains from Crawford bay on Kootenay lake to the head waters of the St. Mary's river, the course of which they followed down to Fort Steele. Here they found a good deal of excitement about the North Star mine, and, joined by "Ed." Smith, who was familiar with this part of the country, they determined to return and prospect in its vicinity.

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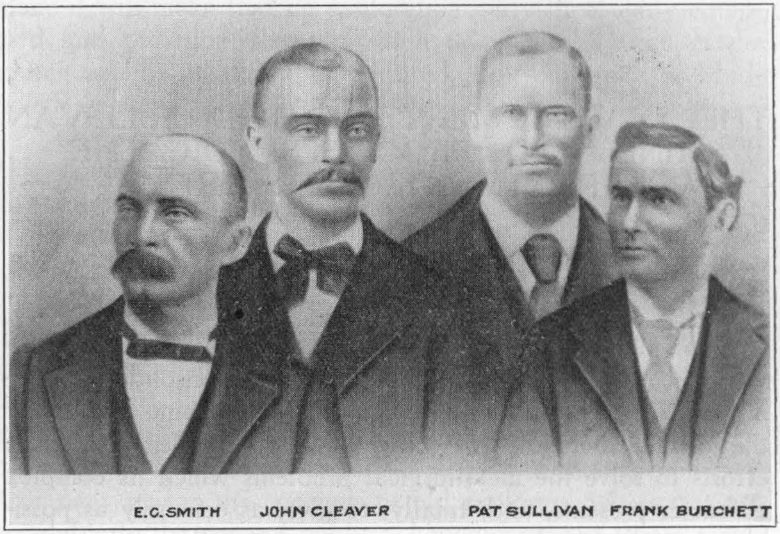

Plate 1

The Discoverers of the Sullivan Mine.

On reaching the North Star they found all of the hill located, but, impressed with the exposure of galena there, they decided to continue their search, in the course of which they crossed Mark creek and prospected the other slope. The second day out Pat Sullivan found the outcrop of the Sullivan vein almost directly opposite the North Star outcrop.

Four claims were located, one for each of them and one for James Cronin, who next year came up to see these claims and also the North Star. At Fort Steele he heard of the St. Eugene ore and returned to Moyie, where he located the St. Eugene and Peter claims. Going back he looked over the claim staked for him on Mark creek and decided to abandon it and devote his attention to the St. Eugene.

The Sullivan group was bonded by A. Hanson, of Leadville, in 1896, but the bond was not taken up. It was again bonded to Col. Redpath, Judge Turner and associates, who formed the Sullivan Group Mining Company.

From 1896 to 1899 some surface stripping was done and several small shafts sunk. In '98 and '99 the Kimberley branch of the Canadian Pacific railway was built to connect with the main line of the Crow's Nest branch at Cranbrook, a distance of 19 miles. This afforded an outlet for the ore by rail, Kimberley being about 2 miles from the Sullivan outcrop.

In 1900 systematic development was started. A shaft house and compressor plant were erected and a steam diamond-drill put into operation. The first shipments were made in this year. During the three following years four or five thousand tons of 35 to 40 per cent. lead ore, carrying about 15 oz. of silver, were hauled to Kimberley and shipped to the Hall mines smelter at Nelson and to the Canadian Smelting Works at Trail.

By this time it was considered that the tonnage of ore developed, some 300,000 tons, justified the construction of a smelter, and in 1903 construction was commenced on a smelter and power plant at Marysville, five miles below Kimberley, on Mark creek. Many serious metallurgical difficulties were met with and many of these successfully overcome, some 75,000 tons of ore being smelted. However, the mine development was not carried sufficiently far enough ahead, nor was care taken in sorting the ore, either under-ground or on the surface. In consequence, the lead content could not be treated profitably at the smelter, and the mine and smelter were closed down late in 1907.

The company had numerous creditors and could not raise sufficient money to meet its debts. The Crow's Nest Pass Coal Company started an action and got a judgment under which they made a seizure, subject to the interest of the bond-holders.

In 1909 the bond-holders and the creditors, including the Crow's Nest Pass Coal Co., re-organized the company under the name of the Fort Steele Mining & Smelting Co., the control of this company being vested in the Federal Mining & Smelting Company.

In December, 1909, The Consolidated Mining & Smelting Company of Canada, Ltd., took a lease and bond on the Federal Mining & Smelting Company's holdings in the Fort Steele Mining & Smelting Company. They immediately took steps to improve the grade of ore mined and installed additional sorting facilities on the surface. This had such a mark-ed effect on the grade of the ore shipped that further improvements in the sorting plant were installed.

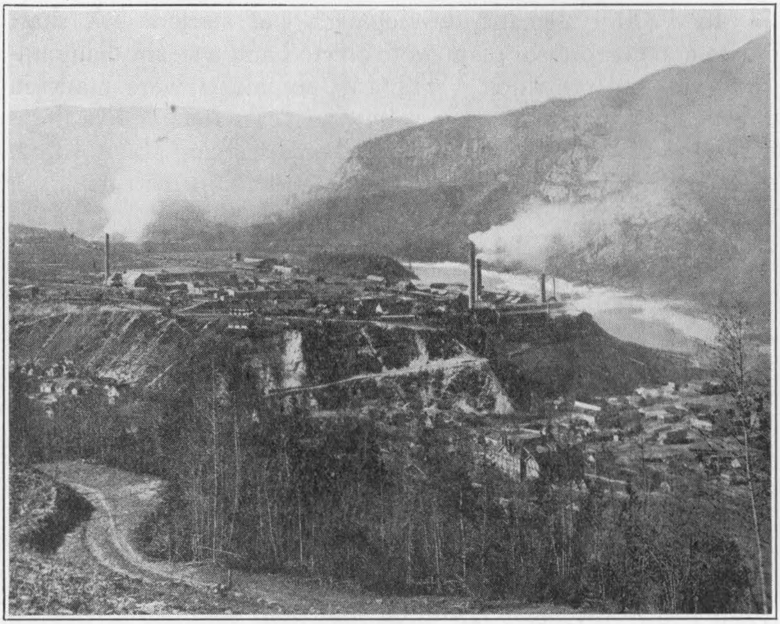

Plate 2
Smelters and Refineries of the Consolidated Mining & Smelting Company of Canada, Ltd., Tadanac, B.C.

The underground development and diamond drilling convinced the officials of the C. M. & S. Co. of C., Ltd., that the mine contained a large tonnage of complex ore which would become very valuable when a satisfactory process of extraction had been developed, also that there were high-grade ore zones which could be worked during the interval and smelted for lead in a suitable smelter with a proper ore mixture. Subsequently, towards the close of 1910, the option on the stock of the Federal Company and on that of some of the other shareholders was exercised, and the control passed into the hands of the C. M. & S. Co. of C., Ltd. Prior to this, however, it had been decided that before the option was exercised it would be necessary to acquire the Hamlet, Shylock and Hope mineral claims in the vicinity of the original group. All of this desired territory passed into the hands of the Company about the same time, and early in 1911 the metallurgical problem was attacked in earnest.

For the next few years, the mine development was directed with a view to discovering ore sufficiently high in lead and silver and low enough in zinc to be smelted with the equipment available at Trail. A considerable measure of success was attained, and the following shipments made:

1910	23,000 tons	1913	31,000 tons
1911	24,700 tons	1914	36,000 tons
1912	28,650 tons	1915	44,650 tons

During this period many improvements were made at the mine. In 1911 a surface crushing and ore-sorting plant was installed at the mouth of the 1,000 level, hoisting was abandoned, and all the ore hauled through this tunnel. In 1912 an hydraulic installation was made on Mark creek and a power house built at the lower terminal of the tramway contained air compressors and electric generators driven by Pelton wheels. Air and power lines were constructed to the mine, a distance of 8,000 feet. In addition, an auxiliary steam-plant was built to provide for periods of low water.

In 1914 the Sullivan mine became the largest lead producer in the Dominion of Canada. It has retained that distinction ever since.

GEOLOGY AND DESCRIPTION OF ORE DEPOSITS

The ore deposits are found in a series of beds of argillaceous quartzite and argillites known as the Aldridge formation, which covers an area of not less than 2,500 square miles. This formation has been intruded by several large sills of gabbro, none of which, however, are in the vicinity of the orebodies. Near the mine the beds strike approximately north and south and dip to the east at angles averaging about 23°. While possessing many of the features of a regular vein, the ores are essentially replacement deposits in the argillaceous quartzites. Irregularities of foot and hanging-walls are prevalent, but in general the deposits are conformable to the beds.

The ore, which in places has attained a thickness at right angles to the dip of approximately 240 feet, is usually a very fine-grained mixture of galena, zinc blend, pyrite and pyrrhotite. The blend is of the variety known as marmatite, represented, in this case, by the formula FeS.5ZnS. Microscopic examination shows that the iron sulphides crystallized first and the galena last.

The ore as developed occurs in two zones which, for convenience, have been called the south and north ore-zones. In the upper levels, between these ore-zones, the replacement is composed entirely of pyrite, while in the lower workings it consists of a fine-grained massive pyrrhotite. The maximum length of the pyrite zone is 800 feet.

Subsequent to the replacement period a moderate folding took place, which resulted in some fissuring, and, in places, a readjustment of the outlines of the orebodies as well as a rearrangement of the minerals composing them. Some faulting occurred at this later period, the lines of fracture striking about north and south. The greatest displacement shown is about 150 feet, but it has had no appreciable effect on the orebody.

While the two ore-zones do not vary greatly in the ratio of lead to zinc, both have produced ores of fairly high-grade in each metal. In the same working-face it is not unusual to find clean galena, zinc blend and pyrrhotite as well as the usual intimate mixture.

MINING PRACTICE AND MINE DEVELOPMENT

Early in the year 1915 it was decided that the ore reserves as indicated by diamond-drilling justified a more ambitious scheme of development to open up the property at a greater depth and make available the ore then known to exist. To this end, in June, 1915, a tunnel was started from Mark creek to develop the orebodies at a depth of 600 feet below the upper workings. It was estimated that this tunnel would come under the south orebody about 7,000 feet from the portal.

The size of the heading as started and timbered was 8 by 10 feet. This size was maintained through 600 feet of gravel, sand and gumbo, at times difficult to handle. Then, owing to still further favorable developments, together with the commercial possibilities of the zinc ores, the size of the tunnel was increased to 9 by 12 feet, which has been held ever since.

No unusual difficulties were encountered in this work other than those which might be expected in cutting pervious beds with extensive outcrops in a country of fairly heavy snowfalls. While no driving records were endangered, progress was satisfactory when considered as part of the ordinary mine development.

Meanwhile research work for improvements in the metallurgy of the ore, especially for turning to commercial account the contained zinc, was carried on with such good results that regular shipments of zinc ore were made to Trail in February, 1916.

In August of the same year there was also shipped iron pyrites for the manufacture of sulphuric acid in the Company's plant. We have, perhaps, an unusual condition in the production of lead ore, zinc ore and iron pyrites from one level of a mine opening the same vein or orebody.

In March, 1919, a raise from the lower tunnel to the vein at a distance of 7,100 feet from the portal, showed ore, and shipments from this continuation of the south-zone orebody have followed with regularity ever since.

As soon as ore was located from the lower tunnel, a crosscut was driven west in the footwall, a distance of 1,075 feet, and a raise made to the upper workings for ventilation and to facilitate the extraction of ore in the intermediate section of this zone. Before raising, a diamond-drill hole 2 11/32 inches in diameter was put down at an angle of 47° in the axis of the raise. This hole served as an alignment, for delivery of water to machine-drills, and afforded some assistance in ventilation. At the same time the lower tunnel was carried forward for the development of the north orebody. Improvements in metallurgical treatment permitted favorable changes in mining practices, so that the average dip, 23° of the orebody, which at one time was considered unfavorable for low mining costs, has not produced the disadvantages anticipated.

The following brief outline of the development of the present mining method may be of interest.

The first shipments were made from ore won from a tunnel which was know as the 20-foot level, and these shipments were hauled by teams to the railway. The mine was then opened with a vertical shaft from which levels were run at short intervals, presumably on account of the flatness of the orebody. Some glory-holing was done, which yielded both carbonates and sulphides, but this did not take place until after the mine was owned by the Consolidated Company.

Stoping operations and timbering followed the usual procedure in square-set stopes. As the workings became deeper, the good hanging-wall of quartzite and the close selection of ore in the sulphide-zone, to provide a shipping product low enough in zinc, initiated a kind of pillar and room system with occasional timber stulls.

After the control of the mine passed to the Company, the 100-foot level was connected with the surface. The pillar and room system was further expanded, and as selective mining for lead ore exclusively was still being carried on, the pillars generally represented portions of the vein too low in lead for shipment, and were frequently of unusual proportions.

The discovery by diamond-drilling of a large section of the vein carrying high-grade lead ore determined that the usual procedure of leaving pillars would tie up too much valuable ore, and it was decided to carry large cribs in the stopes, varying in size from 200 by 18 feet to 20 by 10 feet. These were constructed of heavy timber tied in with wire cable and filled with waste, principally pyrrhotite. When the back was nearly reached, sections of the intervening space were filled solid with timber. More than a year after all the ore had been extracted from this section, which covered an area of 50,000 square feet, wedges between the solid timber and the hanging-wall could be removed by hand, a further evidence of the strength and toughness of the hanging-wall quartzites. After the shipping of zinc ore had commenced, the size of the pillars was reduced and an endeavor was made to leave them with more regularity, sections of the vein below shipping grade being left wherever possible.

With the advent of concentration, especially after satisfactory savings of both metals had been made and shipments of lead ore were not so imperative, further sections of the vein became valuable and the mineable portion in unexploited areas began to embrace almost the whole orebody, except where well-defined and proportionately large bands of pyrrhotite occurred. This condition was reached shortly after the opening of the orebody in the south-zone from the 3,900 [1] tunnel level, where development and haulage ways were principally in the footwall.

The ore was rendered accessible by raises which were driven through the vein to the hanging-wall, at which points stoping operations were commenced. After the broken muck had ceased to run into the raises by gravity, a drag-line scraper operated by a double or single-drum hoist with under and over-winding ropes, was installed for the purpose of drawing the broken ore to the raises. The efficient operating radius of this apparatus in a large stope is approximately 250 feet, and some very satisfactory costs are anticipated when large tonnages are required.

PLAN FOR FUTURE DEVELOPMENT

A survey of stoped areas all over the mine showed that 15 per cent. of the vein had been left in pillars, although in some sections the percentage left was less than half of that amount.

In consideration of this fact, together with the knowledge gained through past stoping operations, and more recent experiences with the drag-line scrapers, it was considered feasible to formulate a definite plan for the exploitation of ore in the north-zone, which the lower tunnel had now reached, and where diamond-drilling had shown a width of vein of over 200 feet. It was decided to continue the footwall development programme as started in the south ore-zone, and from the footwall workings to raise to and through the orebody to the hanging-wall at intervals, governed by the operating radii of the drag-line scrapers. To this end two crosscuts approximately 500 feet apart are now being driven west from the main tunnel in the footwall of the orebody. From these crosscuts drifts will be run north and south, and from these laterals the raises for the extraction of ore are to be driven

(1) Mouth of tunnel is 3,900 feet above sea level.

to and through the vein. At the start it is proposed to leave pillars, to support the hanging-wall, considerably larger than former operations had shown necessary, and it is expected that a large proportion of each, and possibly all the ore in some of them will ultimately be mined. These pillars are to be 100 feet square; the distance between centres will be 300 feet, and the distance between raises approximately 300 feet. One of the crosscuts will carry the foot of the raise to connect with the upper workings of the north ore-zone; this will be driven at approximately the same angle as the one already completed in the south ore-zone, but in addition to providing ore-passes, will be equipped with an electric-hoist for handling men and supplies.

In order to avoid any possible chance of the fall of a large block of hanging-wall and the attendant damage through change in air pressure, pillars will not be drawn to the margin of safety until substantial stopes have been opened in both ore-zones through to the upper workings.

While no mention has been made of sill-workings on the lower tunnel level, some fairly large stopes from which large tonnages have been mined have been opened by crosscuts to the hanging-wall. This work was out of line with the contemplated procedure and became necessary through demands for suitable ore before the footwall development had been sufficiently advanced.

In addition to the footwall development for the extraction of ore between the lower tunnel and the upper workings, plans for the future provide for the sinking of winzes in the footwall of the orebodies in both the north and south zones; lateral development from these will be in the footwall, from which raises will in turn be made to the vein for the extraction of ore.

As is usual in the mining of sulphide orebodies, there has been considerable speculation regarding the danger of fire in large tonnages of broken ore. In order to keep in touch with this possibility, temperature records have been kept of air and of muck piles for a considerable period. These show a remarkable constancy as far as the air and muck piles in the same stope are concerned, the variation being only about 1° Fahrenheit. The temperature of the air and muck in different stopes has also varied less than 15° F., due, no doubt, to differences in ventilation.

While there have been periods during the development of the property when sufficient power was not available for the work in hand, an endeavor has been made to keep pace with the requirements of the mine in the matter of equipment, and the use of new labor-saving devices has always been encouraged.

The power plant has recently been increased by the addition of two 3,000 cu. ft. Nordberg compressors, driven by synchronous motors, making available a total of about 9,500 cu. ft. of air per minute. Mine trackage at the lower tunnel has been changed from 18-inch to 36-inch gauge, over which 6-ton Jeffrey locomotives haul approximately 100 ton trains. Granby self-dumping cars of 80 cu. ft. capacity are in use, and storage-battery locomotives are used for assembling loads.

Bunk-house and mess-house accommodation has kept pace with the steadily increasing crews. Additional houses for married employes have been built and hospital accommodation with a resident doctor and nurses has been provided.

A Meyers Whaley mucking machine, electrically-driven, was installed when the lower tunnel was started. This was followed later by the air-driven Armstrong shovel-loader, and more recently by the Hoar shovel, also air-driven. While these machines have periodically handled large tonnages and were of great benefit during times of labor shortage, the more recent types of air-driven shovels have given better service with lower upkeep costs. One-man water-Leyner type machines of various manufacture are used for drilling, with jack-hammers for blockholing. For blasting, 35 per cent. powder is used in the stopes and 50 per cent. in development faces.

In conclusion, it may be interesting to note that the lower tunnel is now 10,015 feet from the portal and the 600 feet of narrow work at the entrance has been widened to the standard size, 10 x 12 feet, the timber being replaced by concrete posts with steel girders for caps.

The total mining development slightly exceeds 10 miles, and the total shipment of all classes of ore to date is approximately two million tons.

The original Sullivan group, which consisted of three claims of 143 acres, has been enlarged until it now comprises 8,147 acres of crown granted mineral claims, 1,284 acres of locations not crown granted, and 2,386 acres of surface holdings, part of which, however, cover mineral locations.

EARLY RESEARCH WORK

Early in 1910 some research work was started. The old slime-mill of the St. Eugene mine at Moyie was remodelled for experimental work, and exhaustive tests were carried out, using the various methods of water concentration, also air jigs, and film flotation. The results were not encouraging, a recovery of about 50 per cent. of the lead was made in a fairly high-grade concentrate, but nothing was accomplished in the recovery of the zinc.

Samples of ore were sent to different places in many parts of the world, but no encouraging results were received.

Direct fusion of the ore was tried and the galena was found to separate out readily when the mass was cooled. The infusibility of the zinc-iron crusts and the volatilization losses, however, soon indicated that further work along these lines was not justified.

Experiments were next made in efforts to volatilize the silver, lead and zinc in the ore by mixing the ground ore with coal or coke dust and heating in a revolving kiln. Some success was indicated, but early in 1912 leaching processes promised better results. The fuming experiments were tried again later as a preliminary to the leaching operation.

The sulphite process, and one similar to the Ashcroft process, were tried out with some success. Indicated costs, however, and the failure to duplicate results consistently, were not encouraging. Developments in electric smelting were also watched closely.

At this time the French zinc process was being widely discussed. One of the French patents covered the use of sodium sulphate or bisulphate during the roasting or as addition agents to the leaching liquors, and one covered the electrical separation of zinc and manganese from a sulphate solution. An option was taken on this process, but investigation showed there was no advantage to be gained by using it, so it was dropped and experiments were continued, using nothing but sulphuric acid as a leaching agent.

The discoveries of the ease with which zinc could be plated on and stripped from aluminum cathodes, and that in order to secure good deposition the zinc sulphate solution required to be much freer from other elements than had previously been considered necessary, were decisive factors in placing the electrolytic-zinc process on the road to commercial success. The problems relating to the best and most economical means of carrying out the various steps still remained. Perhaps the greatest accomplishments were the successful meeting of the difficulties as they arose and at the same time reducing costs fast enough to keep ahead of the rapidly falling price of zinc.

A few car-loads of high-grade zinc were made in 1915 in the experimental plant.

The present zinc plant, a description of which will be given later, was built in 1915 and started operation early in 1916. Throughout all the later work there was a constant interchange of ideas and results with the staff of the zinc department of the Anaconda Copper Mining Company, and there is no doubt that the speed with which the process developed was due in great measure to this friendly exchange.

EXPERIMENTS IN CONCENTRATION

While research for the extraction of zinc was being continued on the crude ores, of which there was a considerable tonnage, carrying 25 to 35 per cent., some suitable method of preliminary concentration was being sought. Concentration tests were undertaken at the Highland mill at Ainsworth, and also at the old Le Roi mill in Rossland. The St. Eugene concentrator at Moyie was burned down in 1916 after it had been remodelled at considerable expense, and this led the Company to move the testing department to Trail, where it would be under the direct control of the management.

In the spring of 1917, an experimental mill of some 150 tons daily capacity was built at Trail, the intention being to use the Horwood process in the first tests. After a short trial this process was abandoned, as the operating costs were high and the control difficult.

Gravity concentration was tried again, using ore that probably had been more finely ground than in the earlier tests, but the results were no more encouraging.

Wet-magnetic concentration of table middlings and other products gave some promising results which prompted further investigation, leading to the discovery of a new preparatory treatment for this method of concentration. It consisted of heating the ore, crushed to about an inch, to about 750 or 800° F. and allowing it to cool slowly. This rendered the pyrrhotite much more magnetic and made possible a much better separation of this mineral than had been attainable by any other process.

After a thorough laboratory investigation a plant capable of treating 150 tons per day was built to use this method of concentration. The crushed ore, after being heated in a rotary kiln, with practically no sulphur elimination, was cooled by passing through a rotating inclined-cylinder sprayed externally with water. It was then ground in ball-mills and treated on wet-magnetic machines of a modified cross-belt Wetherill or Rowand type, with such satisfactory results that it was decided to increase the capacity to 600 tons per day.

Early in the summer of 1919, during investigation of differential flotation methods, some success was obtained in the laboratory in the concentration of Sullivan ore by this means. Delicate manipulation was essential, however, and the laboratory machine used had at that time no parallel in commercial form, but the excellent results obtained established the fact that flotation would constitute an important part in the ultimate successful treatment of the ore.

Construction of the 600-ton magnetic-plant was rushed to completion because it was essential to keep the zinc-plant supplied with suitable feed, and furthermore with the idea that flotation would find its most economical application in the treatment of magnetic-mill products. It was also realized that extensive laboratory investigation of flotation methods would be necessary before a successful flotation-plant could be designed. Three months' continuous study proved the wisdom of this step, because it was found impossible to duplicate the laboratory results of the small machine in a larger machine of 60-pound capacity.

About this time a combination lead-leaching and flotation process was developed in the laboratory. This step in the flotation work was important, as it was made along somewhat similar lines to the straight flotation treatment. The results were considerably inferior to those obtained in the parallel flotation treatment, as only a mixed float of lead and zinc minerals were made, but the recoveries were superior to those under the magnetic process.

In the first flotation work a lead concentrate and zinc concentrate were made, each containing a high percentage of their respective minerals. The outstanding feature in the new work, however, was that it was done in a laboratory Spitz-machine, a miniature type of a successful commercial machine.

In the meantime the 600-ton magnetic-plant had started operation.

It was then decided to parallel the magnetic treatment with a flotation process in the first experimental plant that had been built. The flotation-plant started on March 7th, 1920, but the first two weeks of operation were discouraging. However, very shortly afterwards, results were obtained that equalled those from the magnetic-plant, and it soon became apparent that magnetic separation for Sullivan ore was obsolete. By June 1st, 1920, a large part of the 600-ton magnetic-plant had been remodelled to enlarge the flotation operation and magnetic separation was permanently discontinued.

During the first five months of operation on flotation there was produced only a mixed lead-zinc concentrate. In these early months considerable progress was made in beneficial reagent changes. In August, 1920, the first lead concentrate by differential flotation was made, and in November lead cleaners were introduced. From that time until August, 1923, the test mill at Trail underwent a long series of changes; for the purpose of increasing capacity, which was finally built up to 1,100 tons per day; for improvements in metallurgical practice; for investigation of efficiencies of machines under different operating conditions; for comparison of machines of different types for the same work; and for establishing the most suitable methods for the installation of satisfactory machines. All changes were made with two definite objects in view:

(1) Carrying on the operations to provide material for plants dependent upon it.

(2) Collecting data and information to insure the proper design and process for the proposed Sullivan concentrator at Kimberley.

The choice and arrangement of equipment and the establishment of the flow-sheet for the Kimberley mill have been entirely controlled by the results and knowledge gained from the operation of the experimental mill at Trail.

THE KIMBERLEY CONCENTRATOR

The site for the Sullivan mill was selected about seven years ago. An extensive reconnaissance in 1921 of all the surrounding country proved the wisdom of this choice. No other location could be found which combined easy access to the existing railway lines, a sufficient head for the water supply, and a large area for the disposal of tailings by gravity.

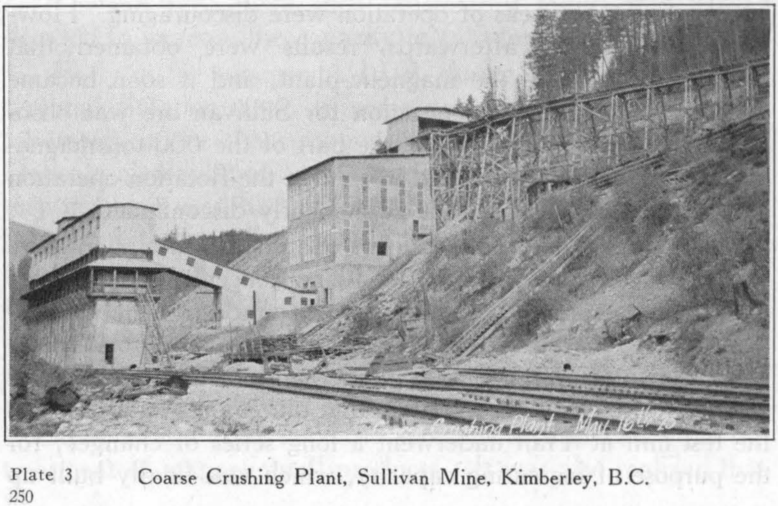

Plate 3 Coarse Crushing Plant, Sullivan Mine, Kimberley, B.C.

Coarse Crushing Plant: The coarse crushing plant is located 500 feet east of the mouth of the 3,900 tunnel. Ore from the mine is delivered to an 800-ton circular concrete receiving bin at the head of the plant. From a double 48-inch air-operated arc-gate in this bin the ore feeds, by gravity, to a 36 by 42-inch Buchanan jaw crusher.

Above the arc-gate is a set of heavy cast-iron fingers which restrict the flow of ore, but swing back automatically to permit the passage of oversize pieces. The jaw crusher, which is set to 8 inches, discharges to a 10-foot inclined fan-grizzly with an average opening of 3 inches. The oversize from this grizzly passes to a 42-inch by 22-foot picking-belt, travelling at 40 feet per minute, the waste being discharged to an 18-inch conveyor. The picking-belt feeds to either or both of two No. 8 Gates gyratory crushers set to 2½ inches.

A 30-inch by 76-foot inclined belt-conveyor delivers the gyratory crusher product, together with the undersize from the grizzly under the jaw crusher, to a 30-inch by 62-foot shuttle conveyor distributing to the 2,500-ton capacity railroad bins. Ore is drawn from the bins to standard gauge railway cars, through rack and pinion gates 2 feet wide by 3 feet 6 inches high on both sides of the bin. Four cars at a time can be filled on each side.

A 15-ton electric travelling crane of 45-foot span and 60-foot lift serves all the machinery in the plant proper. This crane also operates an inclined skipway which brings material from the track level to the plant, a distance of 115 feet. Space is provided at one side of the jaw crusher for making repairs to machinery.

The capacity of this plant, in crushing from 24 to 2½ inches, is 2,500 tons in eight hours or 60 cubic feet per minute, with a total requirement of 355 horsepower.

The site is on a clay hillside with a 33° slope. Test pits showed that this clay extended to a depth of about 40 feet and also that below a depth of 12 feet the clay was quite wet. Consequently the foundations were designed as a unit and were so spread that the dead weight of the building should not exceed 3,000 lbs. per square foot. The coarse crushing building is 50 by 54 feet and its height from the floor at the base

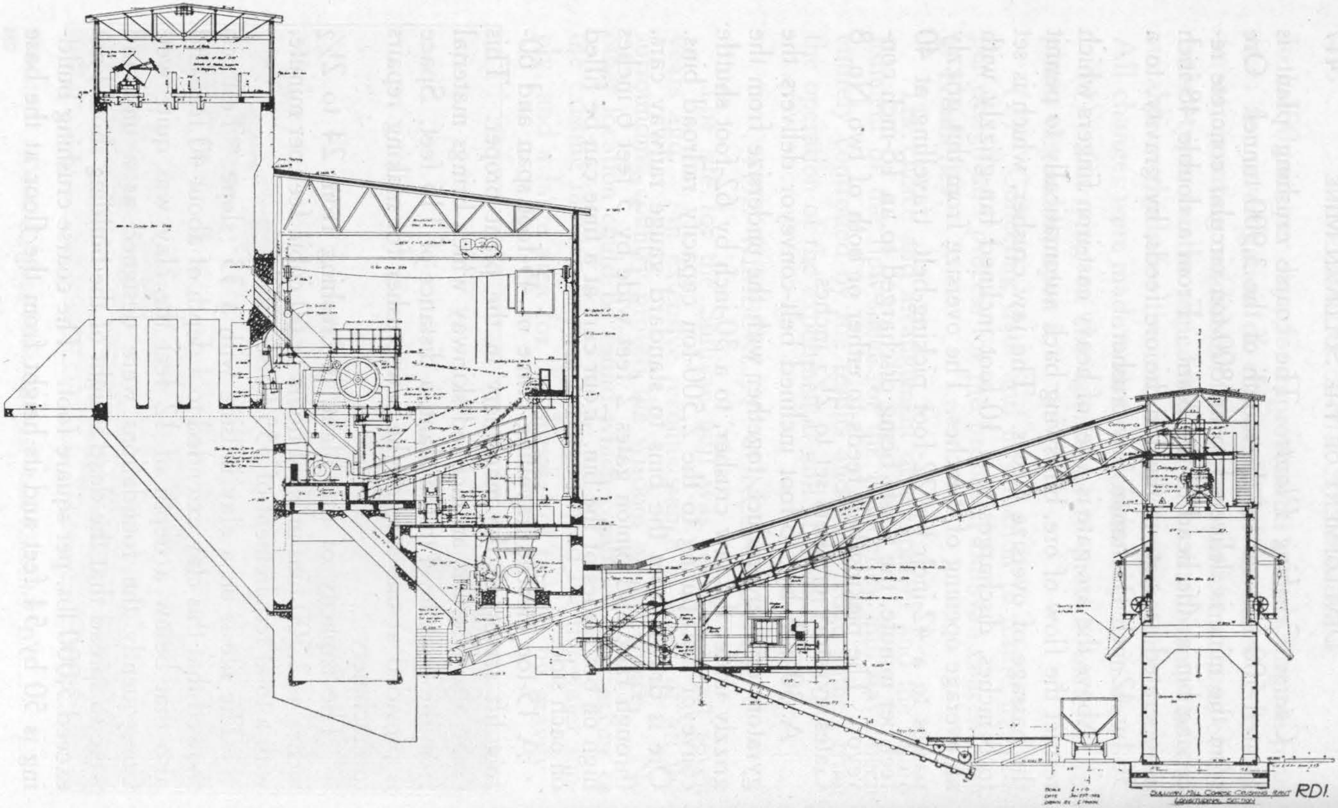

Fig. 1 Coarse Crushing Plant, Sullivan Mine, Kimberley, B.C.

of the gyratory crushers to the bottom chord of the roof trusses is 64 feet. The difference in elevation between the top of the receiving bin and the base of the rail at the shipping tracks is 108 feet. The railroad bins have a concrete foundation 22 feet high, 18 feet wide and 135 feet long, on which rests the steel bins lined with timber. Taken altogether the coarse crushing plant and bins contain 188 tons of steel and 3,000 cubic yards of concrete.

Fine Crushing Plant: From the railroad bins ore is hauled in standard gauge bottom-dump cars 3½ miles to the roll plant. Here it is weighed on 150-ton track scales and dumped into a 1,000-ton capacity concrete bin. The 2½-inch ore is drawn from the bin by 36 by 36-inch roll-feeders and fed over 30-inch belts to two sets of 72 by 20-inch Garfield rolls, in series, the first crushing to 1½ and the second to ¾ inches. The chutes are so arranged that either of the machines can be by-passed.

Each set of rolls is driven by a 150 h.p. squirrel-cage motor. The first set of rolls are run at 50 and the second set at 100 revolutions per minute. A 30-ton electrically-operated crane serves the rolls and as the roll-plant floor is at the same elevation as a nearby track to the machine shop, this facilitates handling of repair parts.

The capacity of the fine crushing plant is 2,500 tons in eight hours from 2½ to ⅝ inches. A total of 380 horsepower is installed for all purposes, including the feeders and conveyors to and from the plant. The building is 35 by 80 feet, of the same type of construction as the coarse crushing plant, and required 85 tons of steel and 1,000 cubic yards of concrete.

A 30-inch by 192-foot inclined conveyor, tandem-driven, carries the product from the roll plant to the head of the sampling mill, where a 100 to 1 cut is taken by a bucket-sampler, which will be described in detail later. This sample is mixed on a revolving table and another 100 to 1 cut taken by a similar sampler. The reject from the second sampler drops direct to the mill bins, and the sample to the bucking room. Reject from the first sampler passes to a 30-inch conveyor equipped with an automatic tripper, which beds the ore

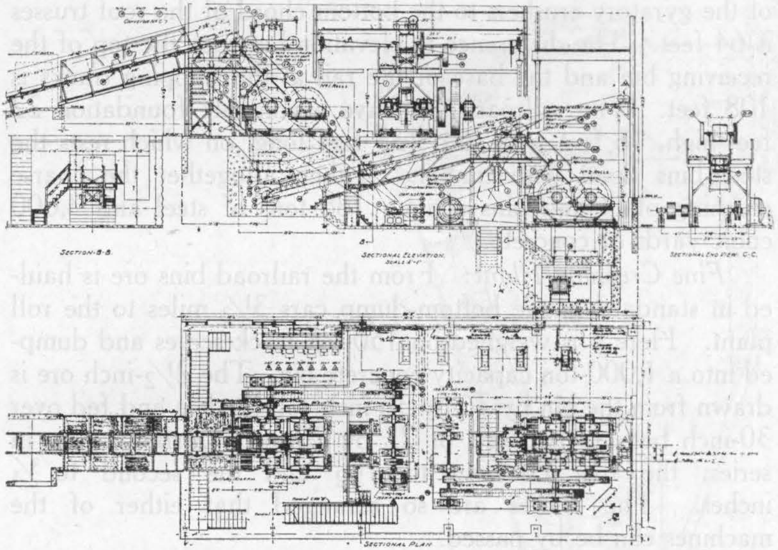

Fig. 2 Fine Crushing Plant ,Sullivan Concentrator, Kimberley, B.C.

in a set of 2,500-ton bins. These bins are 20 feet wide, 18 feet deep and 112 feet long. They are of the same type of construction as those at the coarse crushing plant. Fine ore from these bins is fed by 28 roll-feeders to four 18-inch horizonal conveyors discharging to belts which feed the two primary 8-foot by 48-inch Hardinge mills.

The Concentrator: The total length of the mill building from the north wall of the bins to the south wall of the thickener bay is 606 feet. The width is 126 feet for a length of 181 feet; 78 feet for a length of 152 feet; 126 feet for 145 feet; 48 feet for 16 feet, and 144 feet for 112 feet. The difference in elevation betwen the primary ball-mill floor and the compressor floor of the thickener bay is 73 feet. The average height of the building is 40 feet.

In the classifier and ball-mill bays, basements are provided below the operating floors. When a mill or classifier becomes choked it can be discharged to the basement and immediately put into operation again, thus reducing the lost time. This arrangement has already proven its value. Practically all floors and platforms as well as stairs are of con-

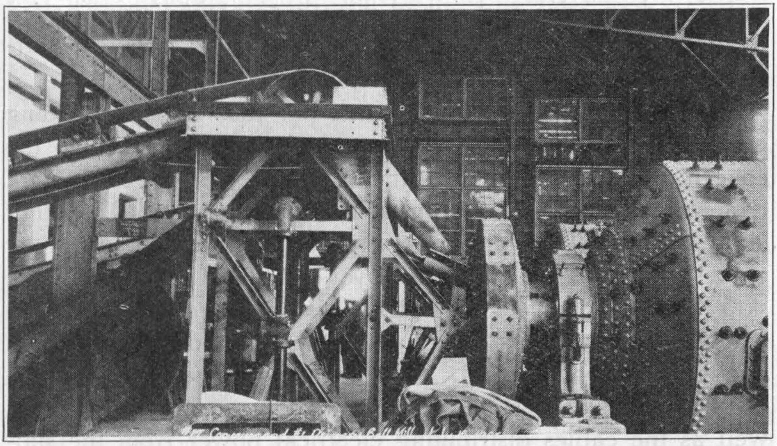

Plate 4

Primary Ball Mill Feeders, Sullivan Concentrator, Kimberley, B.C.

crete or of concrete and steel. In all cases there is provided easy access from the operating floors of any bay to those of the adjacent bays. The zinc retreatment, table, lead-cleaner and filter operating floors, a length of 273 feet, are continuous and on the same level. All main stairways are 4 feet 6 inches wide. A main pump-pit in the centre of the lower half of the mill contains 25 of the 33 sand-pumps in use. All pumps have direct connected motors and independent pipe lines.

The total horizontal area of the mill proper, including reagent mixing shed, sampling plant and heating plant, is 71,532 square feet. The concrete foundations and floors required for this area have a calculated volume of 6,000 cubic yards. The total cubic yards of concrete for all plants, including coarse crushing, is 12,200. The total weight of structural steel for all plants is 1,412 tons. The structural steel for the mill area of 71,532 sq. ft. is 870 tons, or 24.2 lbs. per square foot, including all intermediate floors, stairs, crane and crawl beams.

All outside walls are of 1½-inch gunite reinforced with 12-gauge electric welded 2 by 2-inch mesh. The total area of gunite walls in the mill proper is 43,000 square feet. The window area, 19,400 square feet, is 14.4 per cent. of the total outside walls and roof. The roof is of laminated 2 by 4-

inch wood on edge and spiked directly to nailing strips bolted to the roof trusses. On the 2 by 4-inch wood is laid a Barrett-specification tar and gravel roof.

An inclined skipway, $1\frac{1}{2}$-inch per foot slope, extending along the west side from the machine shop at the top to the concentrate loading tracks at the bottom, serves the entire mill. Material can be loaded from standard gauge tracks at either end. Practically all machinery in the mill is served either by cranes or crawls which extend over this skipway. Under the skipway is a concrete tunnel, 6 feet wide, 6 feet 6 inches high and 960 feet long, extending from the water distribution tank above to the boiler plant. Through this tunnel is carried all main steam, water, flotation-oil, and reagent-solution lines, as well as the cables distributing electrical power from the substation to the starting-boxes in the mill.

The water supply is brought from the tail race of the mine compressor building through a 16-inch woodstave pipeline 16,000 feet long, and enters the distribution house under a head of 131 feet. From this building a separate line is taken off for fire purposes. In the distribution building is a 30-foot diameter by 8-foot tank, placed so as to give a 30-foot head of water at the primary ball-mill floor, and all water for milling purposes is drawn from this tank. In the same building there is another tank which distributes the reclaimed soda solution. Soda solution and water supply are carried in pipe lines to the mill through the tunnel previously mentioned and distributed to all points where needed.

The shop building is an extension of the primary ball-mill bay. It is 41 feet wide, 175 feet long and is served by a $7\frac{1}{2}$-ton electric travelling crane, which also serves the primary ball-mills. Incoming material from a standard gauge car can be lifted and put on the skip for delivery to any point in the mill. The carpenter, machine, boiler, blacksmith, plumber and electric shops are all under one roof.

Fifty-four feet south and on the same level with the machine shop is the warehouse building, which also contains the office, assay office and change room. This building, like all others, is fireproof, and is 40 feet wide by 120 feet long, with one story and basement. A concrete loading platform 10 feet wide extends the full length, and below this platform is a storage space for balls and liners. An air-operated elevator serves the lower floor or basement. In the east end of the warehouse there are bins for the storage of reagents. A lubricating-oil storage house is located diagonally across the tracks from the warehouse building.

Between the warehouse and the mill is a reagent mixing shed with its floor level at the same elevation as the warehouse basement. A set of 2-ton scales is located between the buildings. Reagents, balls and supplies are carried by push-cars over the scales to the mill skipway or to the mixing shed. The reagents are automatically mixed, dissolved, and delivered through pipe lines by gravity to their destinations.

Power from the East Kootenay Power Company's plant at Bull river is transmitted at 60,000 volts, then transformed to 550 volts and delivered to a sub-station adjacent to the mill. From eight 800-ampere panels in this sub-station 500,000 circular-mill cables feed through the distributing tunnel to the individual starting panels.

Nearly all machines are driven by individual motors, most of which are direct connected through flexible couplings. A large number of spur-geared speed reduction units are in use. There are 160 motors in use throughout the plant, varying from 1 to 200 h.p. The power used is 550 volts, 60 cycles, 3 phase, and is distributed as follows:

No. 2. Primary ball-mills, classifiers, conveyors and feeders, distributors.
Installed horsepower 480
Operating horsepower 385

No. 3. Reagent feeders, surge-tank, bowl-classifiers, distributors and No. 1 to No. 11 sand-pumps.
Installed horsepower 537
Operating horsepower 352

No. 4. Secondary ball-mills.
Installed horsepower 800
Operating horsepower 660

No. 5. Rougher flotation-machines.
Installed horsepower 700
Operating horsepower 630

No. 6. Retreatment ball-mill and classifiers, zinc-cleaner, tables, vacuum-pumps and sand-pumps, No. 12 to No. 34 inclusive.

Installed horsepower	783	
Operating horsepower		528

No. 7. Sand-pumps No. 35 to No. 40, lead M.S. cleaners, vaccum-pumps, blowers, compressors, filters, conveyors, soda-return pump.

Installed horsepower	739	
Operating horsepower		509
Total installed horsepower...	4039	
Total operating horsepower (calculated)		3064

NOTE—The actual operating horsepower is 30% less than these calculated figures.

Concentrates are conveyed from the filters to two circular steel loading-bins, one for zinc and one for lead, which discharge into railway cars. Outgoing concentrates are weighed on track scales, and provision is also made for stocking concentrates.

At the foot of the mill are situated the boiler and oil-storage plants. The boiler-plant is 42 by 82 feet, and contains five 100 h.p. boilers, which are used for heating the solutions at all times and also the buildings in winter. The receivers for the vacuum-filters and Genter thickners are drained by barometric legs to the boiler-house, where an 8-inch two-stage pump returns reclaimed soda-solution to the distribution tank at the head of the mill.

The oil-storage building is 32 by 100 feet and contains four 13,500 gallon steel-tanks, which can be filled by gravity from tank cars. An upper floor in this building provides storage for any oil that may be shipped in drums. The storage-tanks are drained by gravity to two mixing-tanks mounted on Fairbanks scales. The oil from these mixing-tanks is pumped to distributing-tanks on the upper floor of the surge-tank bay. From here it is fed by gravity to the oil feeders, which are installed on fireproof floors over the M.S. machines.

As soon as the feeder-tanks are filled, the distributing-tanks are drained back to the storage-plant.

Mill tailings are laundered to an area where a 40-acre settling-pond allows the waste water to clarify.

Nearly all the machinery for the plant was manufactured in the Company shops at Trail. These shops include a modern foundry; a machine shop containing 9 lathes, the largest of which will swing 10 feet and take work 17 feet 6 inches between centres with a 4-foot gap for 20-foot work; also planers, shapers, milling machines, gear cutters, etc. The blacksmith shop contains a 1,500-lb. steam-hammer in addition to the necessary forges. Well equipped boiler and welding shops supplied with Oxweld generators and I.O.C. oxygen and hydrogen cells, makes it possible to undertake a great variety of work. The following list gives an idea of the type of work accomplished:

7—8-foot by 48-inch Hardinge ball-mills, complete with liners.

5—6-foot dia. 6 disc. American filters.

3—Genter thickeners. (Diamond Stiles type.)

158—Cells of M.S. machines, complete.

All conveyor pulleys and idlers.

Balls for Hardinge mills.

All transmission machinery, including bearings, pulleys, flexible couplings, and many cut gears.

Agitator mechanism for stock tanks.

32—Wilfley pumps, 6-inch and 4-inch, with baseplates for motors.

5—Mechanical distributors.

Oil and reagent feeders.

Cast-iron liners for chutes.

Windows and doors.

Circular wooden-tanks.

And many other machines and parts.

All plans and designs for most of the machinery were made at Trail by members of the staff, and the construction was supervised and the mill erected by Company employees trained for the most part at the Trail reduction works. Construction was completed and the concentrator put in operation on August 24th, 1923.

Concentration Practice: The present concentration treatment is represented in the accompanying flow-sheet and may be classified as:

(1) Coarse and intermediate crushing.
(2) Fine grinding.
(3) Concentration.
(4) Dewatering.

The process and equipment in the coarse and intermediate crushing plants have already been described.

It has been found necessary to grind Sullivan ore very much finer than is usual in milling ores for other methods of concentration, so that the constituent minerals may be broken apart and also for the satisfactory operation of the differential flotation process.

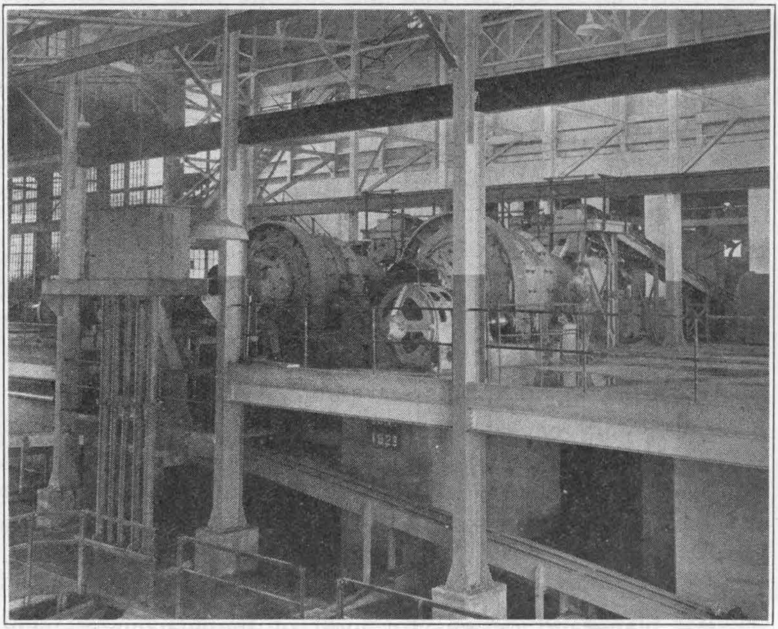

Plate 5 Primary Ball Mills, Sullivan Concentrator, Kimberley, B.C.
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At present the product after fine grinding is approximately 0.5 per cent. plus 100 mesh and 85 per cent. minus 200 mesh. As soon as the bowl-classifiers are put into service it is expected that the ground products will be 90 per cent. minus 200 mesh. To effect this grinding a two-stage reduction is practised in 8-foot by 48-inch Hardinge ball-mills. The primary mills operate at 20.7 r.p.m. with a charge of 3 and 4-inch steel balls. The secondary mills, which operate in a closed circuit with Dorr classifiers, operate at 18.2 r.p.m. with a charge of 1¾ and 2¼-inch chilled cast-iron balls. The Hardinge mill pinion shafts are fitted with S. K. F. ball-bearings, and the mills with special feeders. The feeders were designed at Trail, and consist of an inclined spout through the feed-trunnion with a circular drum fitted with peripheral flights for returning spill from the feed-trunnion to the feed-spout. Ten 6 by 25-foot Dorr rake-classifiers are installed at a slope of 2½ inches per foot with a rake speed of 17 strokes per minute. The operating conditions are unusual since the overflow contains approximately 45 percent. solids, which is found advantageous in subsequent treatment. This overflow is reclassified in two Dorr 10-foot diameter bowl-classifiers. As a result of the high density of the circuit in the primary classifiers some overflowing sands may be classified more satisfactorily in a bowl-type classifier, which has a higher efficiency than the straight rake-type. The pulp density in the bowl-type is also much lower than in the primary classifiers, making possible more efficient classification.

To the grinding circuit are added the reagents essential for the subsequent galena separation, consisting of sodium cyanide, a 40:60 mixture of coal-tar creosote and water-gas tar, and soda-ash.

The 30-foot diameter surge-tank feeding the flotation machines is constructed of reinforced concrete. The agitating mechanism, hung from a chain block, consists of a vertical-revolving shaft to the bottom of which are attached horizontal radial-arms carrying scrapers which convey the pulp to a needle-valve discharge on the outer edge of the tank bottom and insures a uniform flow of pulp to the M. S. machines.

The M. S. machines are of a local design with cast-iron spitz-bottom, variable sand-spigot control and weir-overflow water-level control; features which make the machines more efficient and simple in operation.

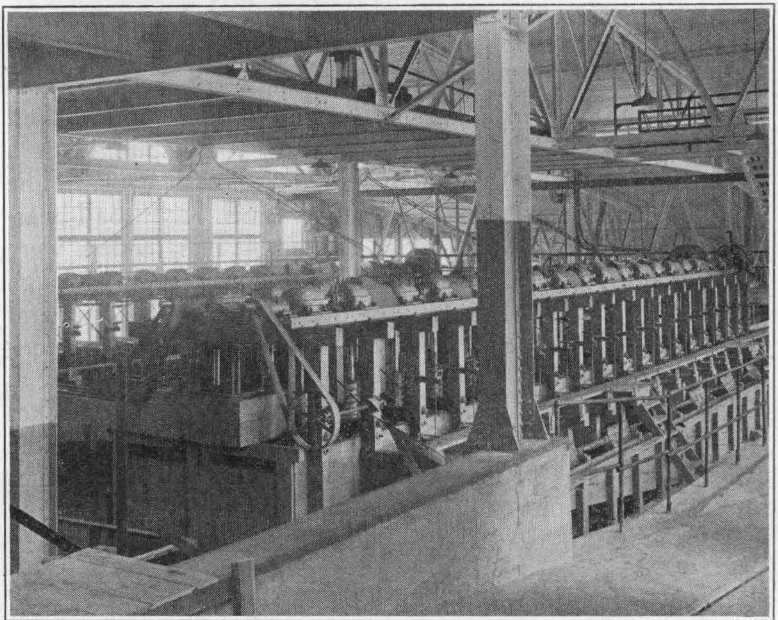

Plate 6
Mineral Separation Flotation Machines, Sullivan Concentrator, Kimberley, B.C.

Reagent mixing and feeding are very important and the facilities provided for this work are excellent. Soda-ash is fed in the dry state by a pan-feeder, which also serves as the soda-bin bottom, and is driven by a ratchet mechanism which allows a large opening in the front of the bin. The soda-ash from the feeder is put into solution by spraying, and is conveyed through pipe-lines to the mill. Bluestone is dissolved in batches to give a 15 per cent. solution of $CuSo_4$ in lead-lined tanks, each of which holds a supply sufficient for 32 hours. These tanks are connnected by float-valves to disk-feeder tanks which allow a very fine control of the quantity of solution fed. Sodium cyanide is handled in practically the same manner as bluestone, excepting that the cyanide requires asphalt-lined tanks and iron-feeders.

Flotation-oil is fed from small oil-feeder units consisting of a storage-tank equipped with small slow-speed submerged piston-pumps with easily controlled variable displacement. Each pump is equipped with an individual motor-drive using power from the lighting circuit, and also with 1, 2, 3 or 4-way distributors. The quantity of oil fed by these feeders is not altered by changes in viscosity, temperature, or other physical changes in the oil, nor by climatic changes which generally affect other types of oil-feeders. Indicators on the pumps show the quantity of oil being fed at any time.

The pulp is first treated to produce a lead concentrate of approximately 50 per cent. lead and 10 per cent. zinc, with a high recovery of lead. To the tailing from the lead-machines there is added the copper sulphate solution preparatory to the zinc-flotation. The pulp is then heated to approximately 25° C., water-gas tar is added and zinc-flotation follows in M. S. machines. The concentrate from the first half of the cells contains approximately 45 per cent. zinc and 3 per cent. lead. From the centre to the end of the machine the zinc in the product falls gradually to about 20 per cent and the products from the last few cells are considered as middling.

A very small amount of galena occasionally escapes through the lead-roughers, but this immediately floats in the first zinc-cell. For guidance in the lead-reagent control the froth from the first spitz of one of the zinc-roughers is pumped to a pilot-table on the same elevation as the lead-rougher floor.

It is found that the coarser sizes in the tailing from the zinc-roughers contain most of the remaining lead and zinc. This tailing is therefore sent to a bowl-classifier making a sand or concentrate product to which is added the zinc-middling from the M. S. machines. This mixture, together with a zinc-iron-middling product from the tables, is reground in an 8-foot by 48-inch Hardinge mill in closed circuit with a 6 by 25-foot Dorr rake-classifier and retreated in a 8 cell M. S. machine, which grades it up and returns a tailing product to cell No. 13 in the zinc-roughers. The solution carrying the tailings from the zinc-roughers contains about one gram of soda per litre and is, therefore, sent to a Genter thickener for dewatering. The filtered solution is returned to the circuit by pumps and a distributing tank.

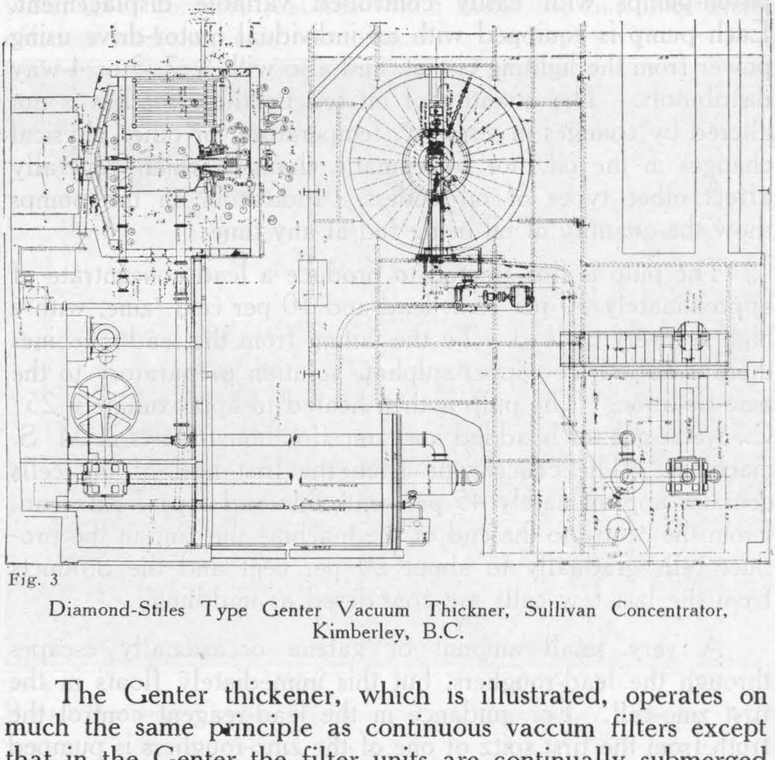

Fig. 3

Diamond-Stiles Type Genter Vacuum Thickner, Sullivan Concentrator, Kimberley, B.C.

The Genter thickener, which is illustrated, operates on much the same principle as continuous vaccum filters except that in the Genter the filter units are continually submerged and the cake formed on them is discharged during submergence. Alternate suction and blow in the filter-frames is effected by a plug valve. The filtrate passes from the frames through the valve to a receiver placed in the centre of the thickener-tank and supporting the valve. This receiver is discharged through a barometric leg. The cake formed on the filter-units is discharged with each blow-back and falls through the pulp to the tank bottom, where it is raked to the centre and discharged in the form of a thick mud. The time of suction is approximately four minutes, and time of blow four seconds. For the blow-back either water or air under about 15 lbs. pressure is used. The filter rate is about 0.2 imperial gallons per sq. ft. of canvas per minute. This machine is of particular value for thickener-feeds of high densities. Apart from the Genter principle, which consists of the filtering and discharge of the cake while submerged, all the structural details, the most important of which are the valve and frame assembly, were developed in the test-mill at Trail.

After re-grinding and flotation of the zinc-machine middling, table-middling and tailing-sands, further liberation of comparatively pure mineral grains has been secured. The concentrate product is therefore tabled, giving a low-grade lead-iron concentrate, which is returned into the primary grinding-circuit. The table zinc-iron middling is disposed of as explained before. The zinc concentrate combines with the primary M.S. machine zinc concentrate for thickening.

In the lead-cleaners the lead concentrate from the rougher is graded up to approximately 65 per cent. lead and 5 per cent. zinc, the tailing being returned to the grinding-circuit.

The American filters are of excellent design with cast-iron shaft, redwood frame, valves at each end of the shaft and individual direct-connected variable-speed motor-drive.

The use of 20-foot stock tanks between the thickeners and the filters for both lead concentrate and zinc concentrate should be noted. These tanks permit a wide flexibility in thickener and filter operation. The vacuum piping for the filters is so installed that separate vacuum control can be maintained both in the submerged portion and in the drying frames. A vacuum of about 24 inches is maintained on the machines. The moisture in the lead concentrate is approximately 8.25 per cent., and in the zinc concentrate is approximately 9 per cent.

Floor drainage is so arranged that all spills from the top of the mill to the end of the lead-rougher are returned by pump into the grinding-circuit. All spill in the zinc-rougher and re-treatment-bays is returned into the zinc-middling M. S. circuit and the areas in the mill are so arranged that lead-spills drain to a large lead-well at the lowest point of the mill, and zinc-spills drain to a similar well. Pumps return these accumulations from time to time to their respective thickeners.

All zinc-thickener feed first passes through a shovel-wheel installation for trapping and by-passing sands and then through a screen-box to eliminate chips and other foreign material.

For metallurgical accounting purposes the 150-ton track-scales on the upper level record the weight of ore delivered to the mill, and the sample-mill installed at the head of the fine-ore bin takes the samples of crude ore. All concentrate as loaded is weighed over 150-ton track-scales on the lower tracks, each day's production being recorded in lots, and individual cars are sampled with ship augers. The accuracy of this method has been established. On the tail-race from the mill are placed two independent automatic Flood samplers actuated by solenoids, one taking a shift sample, the other a 24-hour sample. With the data from these sampling arrangements the total plant efficiency can be calculated. For intermediate mill-samples, automatic Flood samplers, identical with those on the tail-race, are used. In all sixteen of these are installed. Other samples as may be desired are taken by hand.

The concentrate is loaded into C.P.R. 60-ton centre-dump steel ore-cars, specially sealed so as to reduce caulking to a minimum.

ELECTROLYTIC-ZINC PLANT

The steps which led to the construction of the zinc plant in 1915 have already been outlined. This process has now shown that it is able to produce zinc more cheaply than the retort method. It is also able to handle more complex material of a lower grade. On account of this success a great deal has been written about it in the technical press and no attempt will be made to cover ground that has been thoroughly gone over before. A brief description of the plant will be given, and some mention made of the points on which the Trail practice differs from others.

Roasting Division: The zinc concentrate received in bottom-dump railroad cars and unloaded into storage-bins is conveyed by belts to hoppers over the roasters. Two 25-ton hoppers placed 180° apart over each roaster, are provided with slow-moving 30-inch feed-belts which deliver the concentrate to the dryer-hearths of the 25-foot diam, Wedge roasters.

There are thirteen roasters and their function is to oxidize the zinc sulphide so as to render it soluble in dilute H_2SO_4. Some of the chemical problems connected with this operation will be mentioned later.

The roasters have 7 hearths in addition to the dryer-hearth. The working diameter of these hearths is, however, only 21 feet, as the roasters were built with 2-foot walls. Portions of the walls have since been cut out for operating purposes, and changes are being made to increase the effective hearth area. The rabble-arms, two for each hearth, are air-cooled and some of the air is returned to the fifth hearth. The central shaft makes one revolution in 3.5 minutes and the ore remains in the furnace about 8 hours. Each roaster has two coal-burning fire-boxes which deliver heat to the seventh hearth.

Originally the roasters were of the idler type, but it has been found expedient to add a step-bearing, an interesting feature of which is that it can be raised or lowered by an oil-pressure pump. In this way the central shaft can be moved and some operating difficulties overcome.

The situation of the roasters relative to the leaching plant made it difficult to tram material from one plant to the other, so that the installation of an efficient calcine-conveyor was necessary. The task was difficult, however, because of the properties of the material to be handled. The first installation consisted of covered revolving-tables which received the calcine and where water was sprayed for cooling as well as to lay the dust. Rubber belts then conveyed the material to the leaching plant. This method proved unsatisfactory, as did the ordinary type of screw-conveyor, which was next tried out. A screw-conveyor spiral was then fitted tightly inside a wrought-iron pipe and by revolving these as a unit hot material could be conveyed with a minimum amount of wear on the moving parts and without any dusting. From this basic idea the conveyor now in use has been developed, improvements being made as defects appeared.

Conveyor sections, as illustrated, are made of cast-iron with inside diameters of 9 or 12 inches. Ties are mounted at 10-foot centres on the pipe and rotate on idlers which carry the conveyor. One flanged tire is mounted on each conveyor, thus allowing expansion to take place in both directions. The conveyors are driven at about 40 r.m.p. through a spur-gear mounted on the pipe. The capacity of the nine-inch conveyor

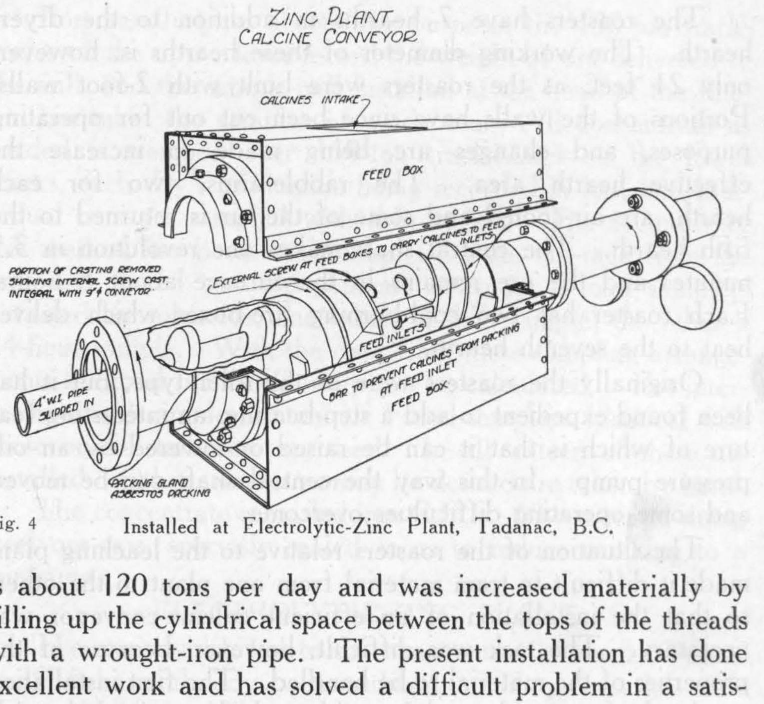

Fig. 4 Installed at Electrolytic-Zinc Plant, Tadanac, B.C.

is about 120 tons per day and was increased materially by filling up the cylindrical space between the tops of the threads with a wrought-iron pipe. The present installation has done excellent work and has solved a difficult problem in a satisfactory manner.

One roaster is used for flue-dust and one for ore and concentrate other than Sullivan mine product. The product from the latter, mostly customs ore, is treated in a separate leaching unit; that from the former joins the calcine from the other furnaces.

The roasters are housed in two parallel buildings with a common flue between. This leads to a 16-section Cottrell plant constructed on the plan of those connected to the Dwight and Lloyd plants at the lead-smelter, which will be described later. This Cottrell treater is operated on the down-draft principle, however. Between the treater and the 200-foot stack is placed a No. 16 Sturtevant fan, belt-driven. The draft pressure at the inlet to the treater is 0.12 of an in. of water. Two 25 k.v.a. rectifier-sets supply power for this plant.

The theory and practice of roasting at Trail: The most important feature of the Sullivan ore in its relation to the zinc-process is the fact that the zinc occurs not as sphalerite but as marmatite, the isomorphous iron of which, on roasting, so readily combines to form zinc ferrate that no satisfactory method of overcoming this difficulty has yet been developed. Zinc ferrate can be broken up, of course, by sulphatising the zinc, but the problem of disposing of the surplus sulphuric acid would have to be overcome and a new roasting technique developed.

Careful work has shown that each unit of isomorphous iron under normal roasting conditions renders approximately 0.58 of a unit of zinc insoluble as zinc ferrate. In other words, practically all the iron in the marmatite is found existing as zinc ferrate after a normal roast.

The iron in the ore, which is present as pyrite or pyrrhotite, does not combine to the same extent. The most important factors determining the quantity of ferrate formed by pyritic iron are: (1) the temperature of the ore bed, (2) the length of time the particles are in contact, (3) fineness of the particles, (4) the conditions under which the roast is finished, viz., oxidising, sulphatising, or reducing.

Zinc combined as ferrate will react wtih the SO_3, formed in the ore bed, and oxidise to sulphate. It is not, however, economical to attempt making all the zinc soluble by this means. In the present roasting practice at Trail about 30 per cent. of the pyritic iron combines to form ferrate. Low temperatures on the upper hearths are not necessary for the formation of the maximum amount of sulphate. The amount formed depends on conditions at the end of the roast, a large quantity being made when the temperature is low with a large excess of oxygen in contact with the roasting surfaces.

An interesting experiment has been made which shows the effect of the second factor above mentioned, and also indicates a means of decreasing the amount of ferrate formed in roasting. Some concentrate roasted so that, as far as possible, no particle was in contact with any other, showed that practically no pyritic iron had combined with zinc. This indicates that more zinc would be rendered soluble in drop-hole roasting than in hearth roasting. The best type of roaster for the electrolytic process would be one that decreases the hearth-factor and increases the drop-hole effect, or in other words, a heated shaft.

An examination of the magnetic properties of the calcine taken from different hearths shows that the susceptibility increases rapidly up to the third hearth, stays about the same across the fourth, and then decreases until, on the sixth hearth, it is about the same as on the first.

The temperature of the ore on the hearths is about as follows, an effort being made to keep the sixth hearth at 1350° F.:

2nd Hearth	1430° F.	3rd Hearth	1420° F.
4th Hearth	1360° F.	5th Hearth	1315° F.
6th Hearth	1350° F.	7th Hearth	1230° F.

The calcine now being made contains about 2.5 per cent. of sulphur as sulphate and about 0.7 per cent. sulphur as sulphide. If the percentage of sulphide is lowered too much, the subsequent extraction is poor.

Leaching-plant: The Trail method of leaching, which requires a double-circuit continuous counter-current leach, has proved to be an important factor in the efficiency and cost of this part of the process.

In this method there are two distinct leaching-circuits, the "acid" and the "neutral." In the "neutral" circuit all the calcine is added to the solution which has already had most of its acid neutralized in the "acid" circuit. As there is not enough acid left to combine with all the soluble zinc present, the solution in this circuit is alkaline and so ferric iron is precipitated along with antimony, arsenic and some other impurities. The solids from this "neutral" circuit then go to the acid side, where they are treated with returned acid electrolyte. The flow of electrolyte is controlled in such a manner that the solution in this circuit is kept acid with from ½ to 1 per cent. of H_2SO_4 at all times. Thus a complete dissolving of all the soluble zinc is assured.

The pulp from the acid circuit is settled, thickened, filtered, and then washed and filtered twice again before it is discharged to be smelted for its lead and silver.

The effluent solution from the neutral circuit is ready for the final purification treatment by which the remainder of the interfering elements are removed with zinc-dust. After this treatment it is ready for electrolysis.

The Trail plant also uses a safety method for catching particles of zinc sulphide which may have passed unaltered through the roasters. This consists of grinding the sands from the acid-pachucas in a ball-mill and then, after treating them by flotation, returning the concentrates to the roasters.

Electrolytic division: The purified solution from the leaching-plant is treated by electrolysis, the zinc being deposited on aluminum cathodes.

The electrolytic department is housed in two large buildings, one of which contains 12 and the other 14 electrolytic units. Each unit consists of 18 double-cell tanks. These are constructed of reinforced concrete and are now lined with a mixture of sulphur and sand $3/4$ to $1\frac{1}{2}$ inches thick. This lining has given excellent service, its di-electric properties making it better than lead, and its wearing qualities better than asphalt.

The inside dimensions of the cells are 27 inches wide, 80 inches long and 42 inches deep, and they each hold 17 lead anodes and 16 aluminum cathodes. The cathodes are 24 by 36 inches and are spaced 4 inches centre to centre. The anode dimensions are one half-inch less each way.

The arrangement of lugs and bus-bars is unique and has given very good satisfaction. It uses a minimum amount of copper and the current leakage is small. The anodes are welded to the lead-covered copper bus-bars and the copper cathode-lugs are split so that they act as a spring when forced down into a slot in the copper bus. One advantage of this system is that it allows the anodes from each cell to be lifted as a unit for cleaning purposes. As the Trail type of bus-bars has frequently been described in detail, it will not be given here. (A good description appears in *Chem. and Met. Eng.* for Aug. 11, 1920.)

Each cell has a separate feeding arrangement controlled by wooden spigots, and also receives the flow from the cell above it in the cascade. The electrolyte maintains about 8

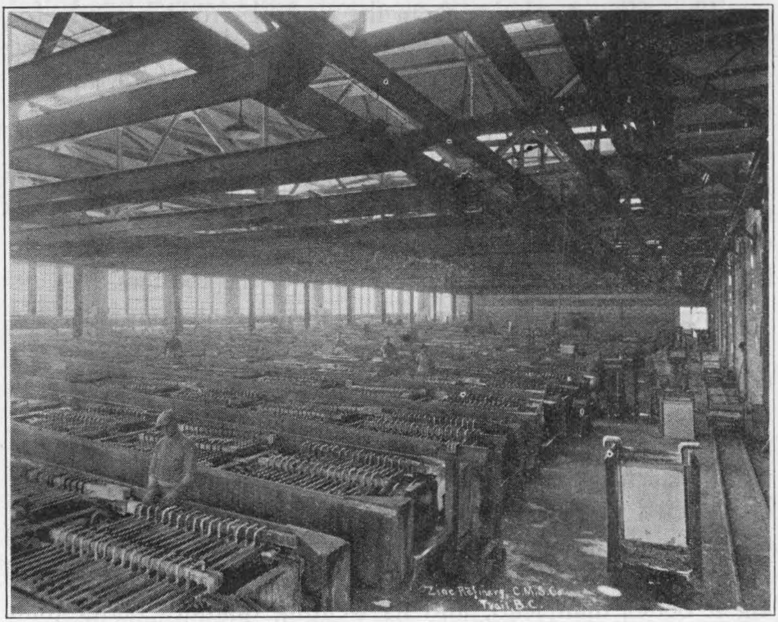

Plate 7

Electrolytic-Zinc Room, Zinc Refinery, Tadanac, B.C.

per cent H_2SO_4 throughout the cascade, thus keeping the electrical conditions constant. One man strips the zinc from the cathodes from each unit, about 8,000 lbs. per day. The stripping work is done on a bonus system and it is not difficult to hold a good class of labor. Not more than four cathodes are removed from one cell at one time, the practice being to pull the cathode, strip it ,and return it to the same place in the tank.

The cathode sheets are stripped every 48 hours, the deposit on each cathode in that time being from 28 to 30 lbs., The average ampere-hour efficiency is 90 to 95 per cent. The cells are kept between 32° and 36° C. by cooling water which flows through lead pipe-coils at the head end of each tank. The current density is from 26 to 27 amperes per square foot and the voltage drop across each cell is about 3.5 volts. Eight to ten ounces of glue per ton of cathode-zinc produced is added to the electrolyte.

The power is supplied by motor-generator sets. Each set consists of one 1,150 k.v.a. synchronous motor and two 500 k.w. generators. A generator is placed on each side of the motor, and each generator supplies power for one unit of 36 cells.

Each building is served by three 3-ton electric cranes, and the I beams on which these cranes travel are hung over the

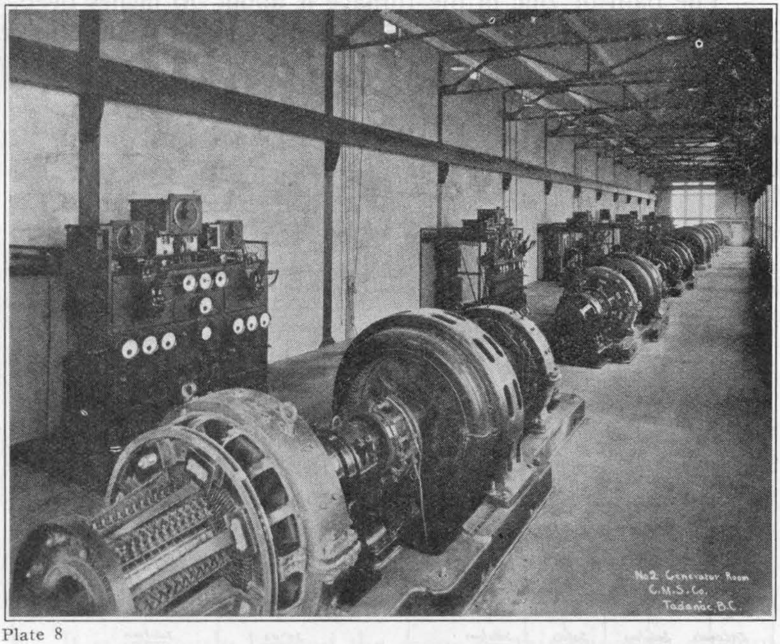

Plate 8

Motor-Generator Room, Electrolytic-Zinc Plant, Tadanac, B.C.

centre of each row of cells. A transfer system running the length of the building allows the cranes to be used where they are required.

Melting and casting: The zinc sheets stripped from the cathodes are stacked on small cars, weighed, and pushed by hand to the melting room. About 7 tons at a time is charged through the roof of the reverberatory furnace. This furnace, 15 by 20 feet, is coal-fired and has a capacity of 100 tons of bar-zinc per day. A reducing atmosphere is maintained in the furnace and the temperature is controlled by a pyrometer placed in the outlet-flue which is kept at 1,250° F.

About 3 or 4 per cent. of the charged zinc has to be removed as dross. Some sal-ammoniac is added during the drossing period to help the segregation of globules of zinc. The dross is leached with returned acid-electrolyte, but as the chlorine would be harmful if returned to the system the dross is given heat treatment in another furnace and the chlorine eliminated before leaching.

The bath of zinc in the furnace is about 30 inches deep. At one corner of the furnace an inverted siphon well is built

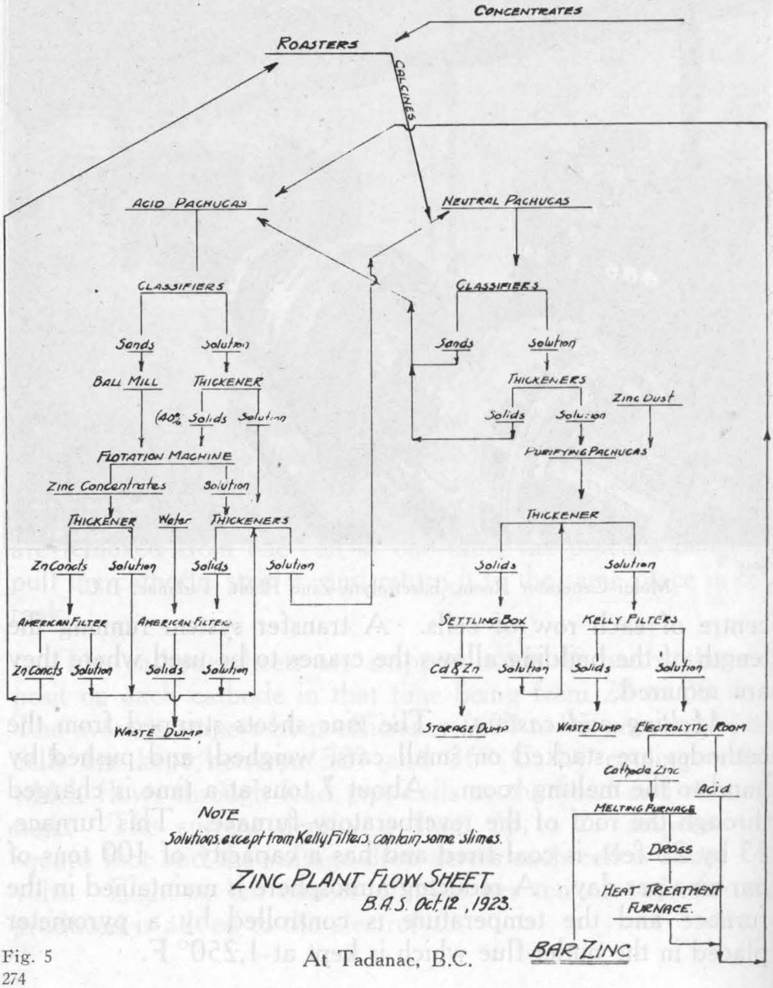

Fig. 5
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At Tadanac, B.C.

which fills with molten zinc, free of dross. The metal from this well is dipped by a ladle holding 200 lbs. and carried by a crawl over cast-iron moulds into which it is poured. The zinc is skimmed before it solidifies in the moulds, which on cooling are turned over by hand, the slabs loaded on small cars, weighed, and either stored or shipped.

The analyses of some of the zinc-plant products are given in the table at the end of this paper.

LEAD-SMELTING PLANT AND PRACTICE

In the lead-smelter the plant and practice have changed as the character of the ores to be treated has varied. The materials received, which on the whole used to be coarse and clean, has tended to become finer and more refractory as methods of concentration have improved. At first it may seem strange to speak of concentration making it more difficult for the metallurgist on occasion, but it is the more desirable slag-forming elements which are removed most easily. A desirable increase in the lead content of the concentrate usually brings with it an undesirable increase in the zinc-iron ratio. This has made the primary treatment of the charge more difficult and of more importance. Where emphasis used to be placed on the blast-furnace work, in recent years the practice has been to concentrate on preparing the charge in the proper manner. If this is done the blast-furnace troubles are few.

It is obviously beyond the scope of this paper to attempt a description of the many steps which have resulted in establishing the present practice.

Sampling division: There are two sampling-mills at the smelter, and as one of these is generally used for copper-bearing ores, the one used for lead ores will be described. It is the custom, however, to check one mill against the other periodically.

Any ores or fluxes that require crushing are delivered by railway-cars to the mill-bunkers, which are fitted with automatic shaking-feeders. Depending on the initial size and the fineness desired in the crushed product, the material can be passed through any or all of the machines shown in the accompanying flow-sheet.

The conveyor between the bunkers and the 18 by 30-inch Blake crusher has a 30-inch rubber belt and "Tadanac" troughing idlers with return idlers of the squirrel-cage type. These idlers have given better service than others that have been tried, and they are being installed elsewhere in the plant as required, The belts carrying the crushed material from the mill are 18 inches wide and deliver to three bedding-systems, two of which are served as a Stephens-Adamson reclaiming device, which works well on dry material.

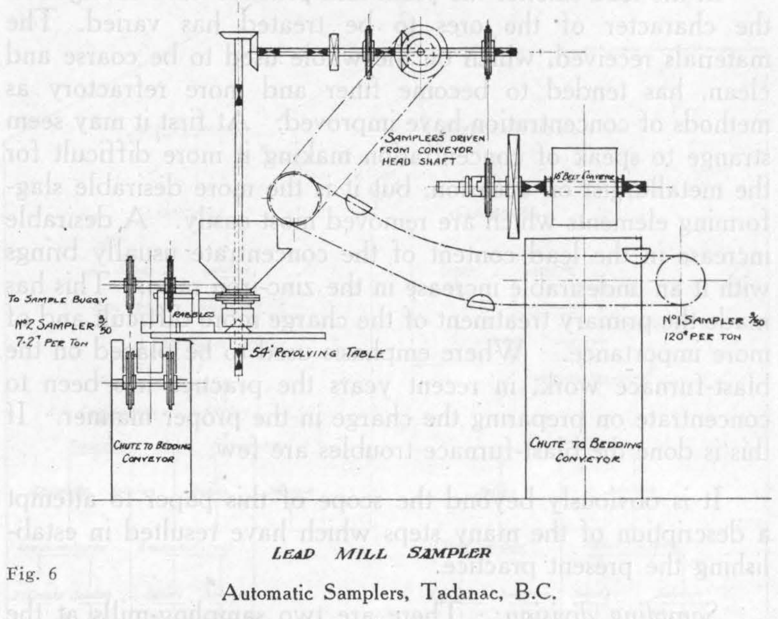


Fig. 6

Automatic Samplers, Tadanac, B.C.

The sampler in this mill is unusual and has given splendid results, being both accurate and flexible. All the crushed material emerges from the mill on one belt. After it is discharged it falls between two parallel horizontal endless-chains running at right angles to the centre-line of the conveyor. Spaced at equal intervals between the chains are three buckets the width of each being one-fiftieth of the length of each chain. The material cut out of the steream by these buckets discharges to a revolving table. The functions of the table are to mix the first cut and also to deliver a steady flow to the second cutter, which is similar to the first but with smaller buckets. Each of these small buckets cuts out one-fiftieth, so that by changing the number of buckets any desired quantity of sample may be taken. When the various lots of ore received at a smelter vary in weight from a few pounds to several thousand tons, it is important that the sampling equipment be as flexible as possible in this respect.

The largest particle in the sample from these two cuts is about 3/8 of an inch. After further cutting on a Jones riffle to 200 lbs. the sample is crushed in a small grinder to 16 mesh and a 40-lb. sample is then taken to the assay office for control work.

Concentrates finer than 1/4-inch are sampled by the fifth-shovel method, except where, as in the case of the Sullivan shipments, they are received in bottom-dump cars when the sample is taken by drilling auger-holes through the concentrates at regular intervals.

The Sintering division: After sampling, the ores and fluxes are either stored or delivered directly to the bins of the primary Dwight & Lloyd sintering machines. There are ten of these bins, and a good deal of thought was expended on their design to make them suitable for wet, sticky material. The top of the bins is 20 feet above the feed-belts, and the bin fronts are *louvred*. While the low height of these bins cuts down their capacity, it is compensated for by the comparative ease with which material near the bottom can be removed, an important feature in the winter months. There are two bins equipped with one feed-belt, five with two belts, and two with three belts. The flue-dust bin, instead of belts, has two pug-mills fitted into the bin bottom. This was necessary to make the bins as dust-tight as possible, and also because the dust is liable to burn under certain conditions. The pug-mills, however, only work well when the dust is fairly dry.

The bins discharge by means of 30-inch feed-belts to 24-inch collecting-conveyors, which in turn deliver to a mixing-table where any water which may be necessary is added. This feature is not a common one in most Dwight & Lloyd plants, the mixing usually being done at each machine. After trying different types of mixers and even attempting to sinter without any mixer at all, it has been found that best results are obtained by the system described.

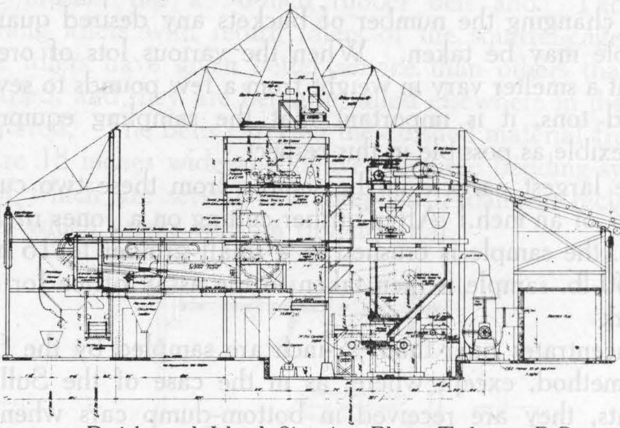


Fig. 7 Dwight and Lloyd Sintering Plant, Tadanac, B.C.

After the charge is mixed it is delivered to a Peck carrier or to a series of 18-inch belts, if the Peck carrier is under repair. The hoppers over the machines receive the mixed feed from these conveyors and deliver it by 30-inch belts to an oscillating chute, the function of which is to distribute the coarse and fine particles of the charge evenly across the machine pallets. It is the best device of the many that have been tried for the same purpose.

In the primary sintering-plant there are at present five 42 by 264-inch Dwight & Lloyd machines. The pallets of these are fitted with a style of grates devised at Trail, and which are now coming into general use in other plants. There is a shear-pin arrangement in the driving mechanism. At the front-end the pallets discharge the sinter into hoppers which deliver either to a 36-inch pan-conveyor or to V-bottom cars. This makes it possible to use these machines for the making of second-sinter if desired.

Some of the features of this installation are worth noting. The speed-cones made at Trail are of extra heavy construction, as an analysis of the lost-time record showed that a considerable amount was caused by the cones. All the spill through the pallets at the feed-end drops through the opening between the pallets as they are lifted by the drive-sprockets and falls into a chute which delivers to the Peck carrier and is returned to the hoppers, thus providing an automatic removal of this troublesome material. To take full advantage of this method of spill-disposal and also to save time in wind-box cleaning, the latest machines are built with doors in the wind-box bottoms. These doors permit the spill material to drop on the pallets as they are making their return trip, and it is disposed of as described above. The sinter is sprayed with water after it passes the dead-plate at the end of each machine. This is done not only to keep the dust down but to make use of the slaking action of the lime in the charge.

A series of pan-conveyors delivers the wetted primary-sinter to a crusher which was developed at Trail. Toothed rolls had been used previously without much success. If the material was wet the rolls plugged up, if it was dry the dust was very bad. The sinter crushed by the rolls was conveyed by a small screw-conveyor and it was noticed that the product delivered by the conveyor seemed to be as well crushed without using the rolls as with them. This led to the design of the present crusher, which is simply a heavy sectional cast-iron screw-conveyor working in a trough, the bottom of which is made up of a series of grates. Any material which reaches the end of the screw is forced through a vertical steel-grating at the end of the trough. The grates are spaced about 1¼ inch apart, and satisfactory second-sinter can be made from material crushed to this size. The crushed product is conveyed by 18-inch rubber belts to the bins at the secondary-sintering plant. The life of the belts on this class of work is about nine months.

The bins at the resintering plant are made of reinforced concrete with hopper-bottoms. They have given satisfaction on products which run readily, but are not to be recommended for very wet fine material. They discharge by 30-inch belts to a collecting-conveyor which feeds a belt-and-bucket elevator, discharging in turn to the mixing-table, which in this plant is at the top of the building. The mixture, wetted if necessary, is then conveyed to the machine-hoppers and fed to the pallets as previously described. In this plant the wind-boxes on each machine have shutes which discharge into V-bottom cars. The finished product is weighed and transported to the lead-furnace storage-bins in 18-inch gauge V-bottom cars hauled by an electric locomotive.

Fuel-oil is burned by a special type of burner, designed for low-pressure air (4 to 8 ozs.), which has been very successful and has reduced the oil consumption by about 60 per cent.

The thickness of the charge on the pallets is 4 inches on the primary machines and 8 inches on the secondary. The thick bed has proved advantageous in many ways and it is possible that even thicker beds may be used.

Each machine has its own fan driven by a 75 h.p. 900 r.p.m. motor. About 20,000 cu. ft. of gas per min. is exhausted from each machine, the wind-box draft varying, of course, with the porosity of the charge.

On the machines in the primary plant considerable trouble is experienced with the fan-casings and runners being eaten out by the sulphuric acid in the gases. Copper-steel has been tried in place of ordinary plate and has about fifty per cent. greater life under similar conditions.

Blast-furnace division: The charge to the blast-furnaces is made up of sinter, settler-shells, furnace-cleanings, dross and sometimes oxidized ore, mostly from the Paradise mine in East Kootenay. This material is contained in reinforced concrete-bins of approximately 2,500 cu. ft. capacity each. There are 3 shutes to each bin which deliver to pan or belt conveyors operated by d.c. motors. These discharge the material into weighing-hoppers, which in turn dump into V-bottom charge-cars. The coke is weighed in similar hoppers, but is hand-loaded. A charge-train is made up of 6 V-bottom cars hauled by an electric locomotive, three of the cars containing coke and three containing the charge.

There is a charge-train for each furnace and the charges, about 2,000 lbs. each, are dumped at alternate sides on cast-iron feed-plates set at an angle of 22½° from the horizontal; experiments having proved that this was the most suitable slope to use. Since this method of feeding has replaced the use of centre-dump cars with a stationary spreader in the furnace, there has been less operating trouble, the coke consumption has been less, and a smaller amount of lead has been fumed per ton of charge and per unit of time.

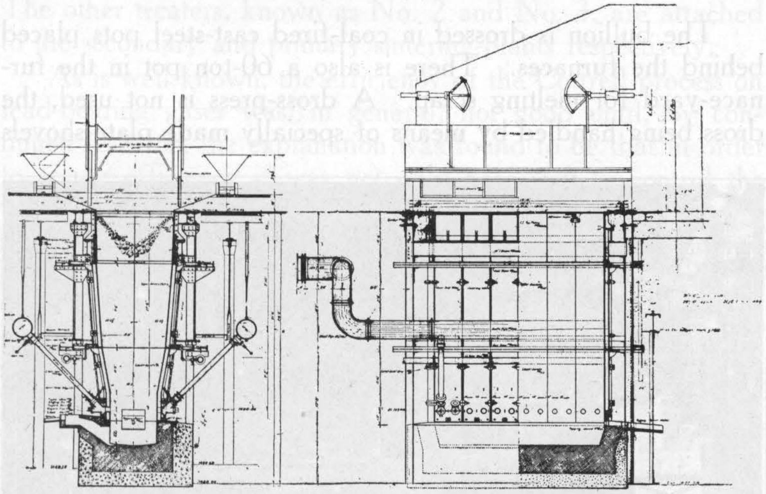


Fig. 8 Lead Blast-Furnace, Tadanac, B.C.

There are four lead-furnaces, the main dimensions being 180 by 50 inches at the tuyeres and 19 feet from tuyere-line to feed-floor level. They have six jackets to a side, each with openings for two four-inch tuyeres. Two of the furnaces have two tiers of water-jackets and two have but one, the shafts being of reinforced-concrete lined with fire-brick. The latter type is more satisfactory for the work at present, as there is less tendency to form crusts and consequently the operation is better and the campaign longer. The furnace crucibles are fitted with the "Northport" type of lead-well, which is easier to work with than the older style. A pot lined with fire-brick is used as a cooler under the lead-well and the bullion is run through launders to the 40 or 50-ton skimming-pots. Each furnace has two slag-settlers operated in series, one of which is usually changed every 24 hours. The slag overflows into a launder in which it is granulated and carried to the slag-dump. That portion of the slag which is mixed with the crushed sinter is elevated by a 4-inch Krough pump from a sump in the launder to a storage-bin, where the water is drained off.

The furnace-room is well supplied with fans and ventilators and every effort is made to keep the building clear of smoke.

The bullion is drossed in coal-fired cast-steel pots placed behind the furnaces. There is also a 60-ton pot in the furnace-yard for melting scrap. A dross-press is not used, the dross being handled by means of specially made plate-shovels

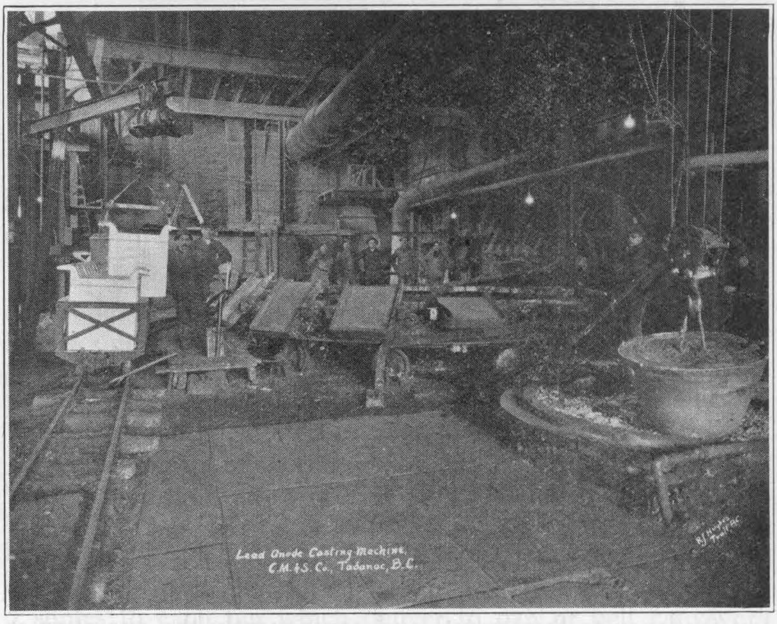


Plate 9 Anode Casting Wheel, Lead Plant, Tadanac, B.C.

with perforated blades. The bullion is pumped from one pot to the anode casting-wheel, which has 16 moulds. These moulds break at the lugs, and as the wheel revolves the pan is tilted in such a way as to make it an easy matter for the crane-man to life the anodes out and place them in rack-cars ready for transportation to the refinery. All dross is sent back to the blast-furnaces for resmelting.

Smoke treating division: There are three separate Cottrell plants in use for handling lead-bearing gases. The oldest of these, known as No. 1 treater, was the first commercial installation of its kind in connection with the lead-smelting industry, and is used for cleaning the gases from the lead blast-furnaces. The other treaters, known as No. 2 and No. 3, are attached to the secondary and primary sintering-plants respectively.

As is well known, the efficiency of the Cottrell process on lead-bearing gases was, in general, not good until, by continued research, the explanation was found to be that in order to secure efficiency it was not only necessary to control the velocity of the particles through the pipes, but it was also necessary to have a conductive deposit. The general method of obtaining this has been to humidify the gas by means of water sprayed under pressure.

This necessary humidifying has brought in its train a number of new problems, some of which have yet to be solved in a satisfactory manner. The use of water reduces the gas temperature, thereby reducing the stack-draft, making it necessary (in order to maintain the proper draft) to install fans or add heat to the gas, both expensive methods. Then, as there is generally a trace of SO_3 in Dwight & Lloyd machine gas, the problem of securing material that will stand the corrosion is a serious one. In the humidifying flues themselves, even in the absence of acid, the continuous action of the water-sprays deteriorates the flue. In addition there are the problems of handling and settling the water from these flues, the cleaning them out, and the handling and treatment of the resultant wet slimy material.

A No. 18 Sirocco fan is connected to the flue from the lead blast-furnaces and delivers the gases into a gunite-lined brick-flue built on 5-foot concrete walls and having a concrete floor. This flue is 80 feet long and 16 feet high, and is divided into two sections 12 feet wide. It is fitted with dampers so arranged that the sections may be used either together or separately. This provision avoids a shut-down of the furnaces when cleaning out the flues, once every six weeks. The gunite lining of this flue has just been removed after 4½ years' service.

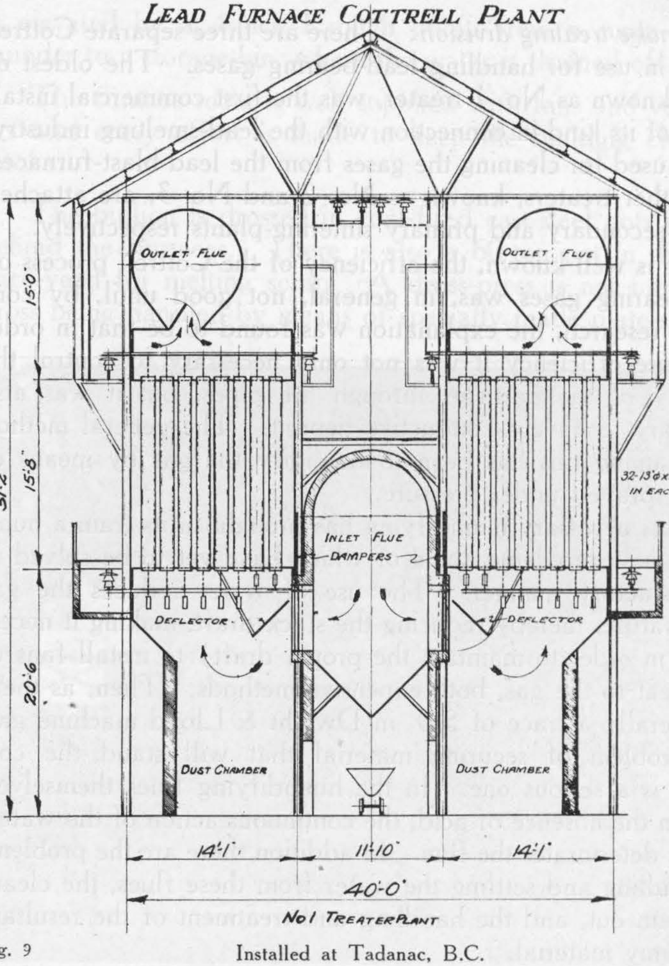


Fig. 9 Installed at Tadanac, B.C.

Considerable difficulty has been experienced in procuring material for this humidifying-flue of the Dwight & Lloyd machine treaters which would stand up against the corrosive action of the gases, but this problem has been solved by a wooden-flue gunited on the outside. This was built and put into service several months ago, and, on examination recently, proved to be in excellent condition. The iron-work round these treaters and the treater-pipes themselves corrode with fair rapidity, because any acid present is precipitated in the treaters. At the present time a test is being made with some pipes plated on the inside with a coating of metallic lead. As this test, however, has only been in progress a few months, it is too early to say whether the pipes so treated will be a success. Water for humidifying is delivered at 200 lbs. pressure through triplex pumps, Meyers sprays fitted with brass or aluminum discs, with 1/16-inch aperture being used. All treaters are up-drafted, but the Dwight & Lloyd machine treaters differ in some important respects from the one connected to the lead blast-furnaces. The latter is entirely of iron construction, while the former are constructed chiefly of brick and concrete, iron being used as little as possible.

The inlet-flue in all of the treaters is in the centre, with the sections containing the pipes on both sides. In No. 1 treater the gases pass up through the pipes to an outlet-flue immediately above the top-header of the sections. Treaters No. 2 and No. 3 differ from No. 1 by having the outlet-flue directly above the inlet-flue, the top of the inlet-flue forming the bottom of the outlet, and the top of the outlet-flue being the operating floor of the treater. The gases from the upper-headers in this case are led through down-takes to the outlet-flue. This latter type gives a better distribution of the gases to the different sections, although it must be noted that the headers of No. 1 treater are not of sufficient height to give the best results.

This question of distribution is one of the important points in treater design which is best attained in the down-draft type, although the opinion at Trail is that the other points in favour of the up-draft type more than offset this advantage.

In No. 1 treater the lower grids are supported from two insulators on the outside of the sections, the inner-ends being supported by the discharge electrode-chains through the corner pipes. In the other type the lower grids are supported from 4 insulators, one at each corner of the section. A treater section consists of 32 pipes, each 13 inches in diameter and 16 feet long, with a discharge electrode of dog-chain passing down the centre of each pipe. The dog-chain electrode was first used in this plant. No. 1 treater has 18 sections, No. 2 has 9 sections, and No. 3 has 8 sections, each section being fitted

"DWIGHT & LLOYD" COTTRELL PLANT

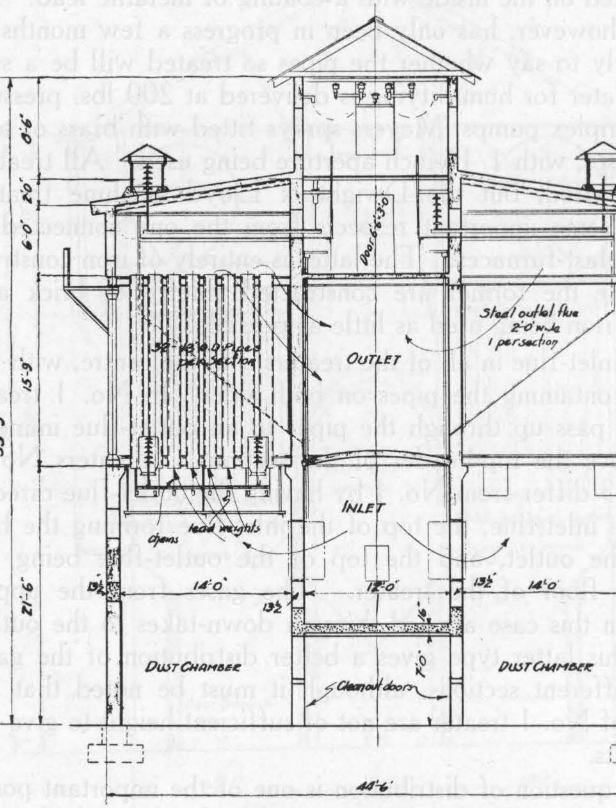


Fig. 10 Installed at Tadanac, B.C.

with separate dampers and switches. The problem of removing the fume from the treaters is sometimes a troublesome one, and after experimenting with various systems without success, brick stalls were built under all the sections and the material shovelled into V-bottomed cars by contract. Whereas in the past great difficulty was encountered in getting the work done at all, there is now no trouble. This is an excellent example of the fact that it is sometimes good practice to discard machinery and turn to man-power, even if high wages have to be paid.

The rectifier-room in connection with the treaters is equipped with six 38.5 h.p. motor-generator sets; 550 volt, 1,800 r.p.m. motors and 25 k.v.a.-110 to 250 volt generators. The transformers step-up from 220 to 60,000 volts. With humidified gas the generators run at about 260 volts. A reinforced concrete-stack 18 feet in diameter by 256 feet high serves Nos. 1 and 2 treaters, while a brick-stack 12 feet square by 180 feet high furnishes draft for No. 3 treater.

METALLURGICAL PRACTICE

The rapid succession of products which the treatment of Sullivan ore has brought about has kept the lead metallurgist on the alert. No sooner had the problem presented by one particular grade of product been solved and the plant rearranged in accordance, than some change, either in the orebody or in methods of concentration, resulted in an entirely new set of conditions. Within the past five years the charge to the furnaces has varied from 8 to over 40 per cent. lead, with corresponding changes in the other elements. At times sulphur had to be added to the charge in the form of raw pyrite, and at other times there was too much of this element present for the available equipment. The problem of smelting the zinc-plant residue has also been a difficult one, not only on account of its chemical composition, but also on account of its physical characteristics. However, with the present concentration programme as outlined, it is hoped that in future there will be a uniform supply of concentrates available.

In preparing the charge for the D. & L.'s the three main points to watch are: (1) size of the sulphide particles and the proper mixing of them; (2) the amount of moisture in the feed; (3) the amount of sulphur in the feed.

When the charge is made up principally of flotation concentrates, as it is now, it is of major importance to get the moisture content of the feed to the point where the material will tend to form little balls as it is rolled around on the mixing-table, and as it rolls down the oscillating-chute to the D. & L. pallets. The object of this is to make the charge to the machines as open as possible, so that the maximum amount of sulphur may be eliminated. Enough sulphur, however, must be left to develop the heat necessary to make a firm second-sinter. From experience here it is found that the limits for a high-zinc charge are from 11.5 to 14.5 per cent. total sulphur in the feed to the primary-machines. For the second sinter, primary-sinter carrying from 8 to 9 per cent. sulphur gives the best results. To the second-sinter is added from 10 to 15 per cent. granulated lead-furnace slag. The functions of this slag are mainly to assist in the formation of a hard sinter by reason of the fusibility of the slag, which melts readily and binds the cake together; and also to act as a factor of safety at times when, for some reason, the primary-sinter runs high in sulphur. By using slag as a diluent, operations can be carried on until the condition is rectified.

The desired objective is the production of a final sinter as low in sulphur as possible, the lower the better. It usually approximates 1 per cent. and seldom exceeds 2 per cent. Higher sulphur gives trouble at the blast-furnaces, forming crusts, and means loss of lead in the slags.

As much as possible of the charge is sintered with the idea of securing pre-digestion, as it were, before it arrives at the blast-furnace, thereby improving the efficiency of the furnace. Results have proved that this is the best practice to follow, under the conditions at Trail.

In regard to blast-furnace practice, so many slag types have been run, of such varied analyses, that it is doubtful whether any other one smelter has ever had quite the same "ups and downs" with the slag-forming elements as can be seen in the Trail slag-graphs. These show a variation in SiO_2 of from 16 to 39 per cent.; in Fe. of from 17 to 35 per cent.; in CaO of from 6 to 19 per cent., and in Zn of from 2.5 to 22 per cent.

The zinc-iron ratio in the charge is of great, in fact greatest importance in running a slag high in zinc. Given a sufficient amount of iron, the whole question of smelting a charge high in zinc turns on the elimination of sulphur. If the charge is oxidized furnace trouble is practically eliminated.

The amount of copper in the charge is negligible and consequently no matte is made, the copper being slagged off. Before being slagged, however, it contributes to the large amount of dross made, which is sometimes as high as 30 per cent. of the bullion production. The dross is returned to the furnace-bins and forms part of the charge. The recovery of the copper as practised in other plants is not profitable at Trail.

In operating the blast-furnaces the level of the top of the charge is kept about 2 ft. below the feed-floor level, or approximately 17 ft. above the centre-line of the tuyeres. The blast pressure is kept as close to 48 oz. as possible, this being the maximum pressure the Root-type blowers will deliver. The tonnage per furnace-day varies, but is usually about 350 tons of charge (excluding dross but including cleanings). It is the practice to run the furnaces down about 12 feet every 10 days and top-bar the side and end crusts. With only two furnaces in operation no particular effort is made to make campaign records, and it is found much cheaper to start a new furnace and dig out the old one. A campaign of 3 months is considered quite satisfactory, though a furnace has just been shut down after a successful run of 7 months. After a longer campaign than this the furnace crucible usually fills up with an artificial sulphide mineral which, it is interesting to note, is practically the same both chemically and microscopically as the original Sullivan ore.

The high-iron high-zinc slag ordinarily made is very fluid and gives no trouble either in tapping or settling. A considerble tonnage of this slag has been stored with a view to recovering the zinc.

The furnaces are served in front by a 20-ton travelling-crane and 5 and 3-ton travelling-cranes serve the bullion casting operation, with a small 1-ton jib-crane for loading anodes.

The bullion is cast at a temperature not exceeding 700° F. in order that as little copper as possible shall be sent to the refinery. The anodes, which weigh about 360 lbs. each, are transported in rack-cars holding 24 anodes.

REFINERY PRACTICE

The refinery operations comprise: (1) the electrolytic separation of pure lead from lead-bullion by the Betts process; (2) the electrolytic separation of copper from copper-bullion; (3) the furnace treatments whereby the gold and silver contained in the anode-sludges from both electrolytic processes are converted into Doré metal; and (4) the parting plant, where the gold and silver are separated from one another.

The electrolytic lead-tank room: The Trail electrolytic-lead refinery was the first of such to operate the Bett's process, and this method was developed at Trail from a laboratory stage to a commercial success.

This plant has two units, one of 75 tons and the other of 85 tons rated capacity of pig-lead per day. Each unit is divided into three cascade-sections, which in the modern unit consists of six rows of double-tanks, separated longitudinally by 23-inch working passageways. There are seven tanks to each cascade and the circulation is maintained by four copper plunger-pumps, 10-inch diameter by 12-inch stroke, which return the overflows to a feed-tank. From this tank 3 individual feeds, composed of $1\frac{1}{4}$-inch hard-rubber piping, deliver the solution to the head-tanks of the cascades. The circulation is maintained with a flow of from 4 to 5 gallons per minute to each tank. The 492 tanks are of concrete and are lined with a high melting-point asphalt mixed with some sulphur for hardening purposes. The tanks in cascades are 5 inches apart with a 3-inch drop between each tank. The solution is fed to the tank 1 inch below the surface level at one corner of the tank and withdrawn from a depth of 30 inches from the surface-level at the opposite corner, thence to the next tank in cascade.

The anodes, weighing about 360 lbs., have a life of 8 days, during which time two crops of cathodes are withdrawn. The voltage drop between electrodes varies from 0.23 to 0.5 volts, according to the length of time in the tanks and to the depth of the adhering slimes. The cathodes, drawn every 4 days, are washed free from electrolyte with water, and dumped into the melting-kettles. The new cathode-sheets are set by hand. The melted lead in the kettle is heated to 500° C. and

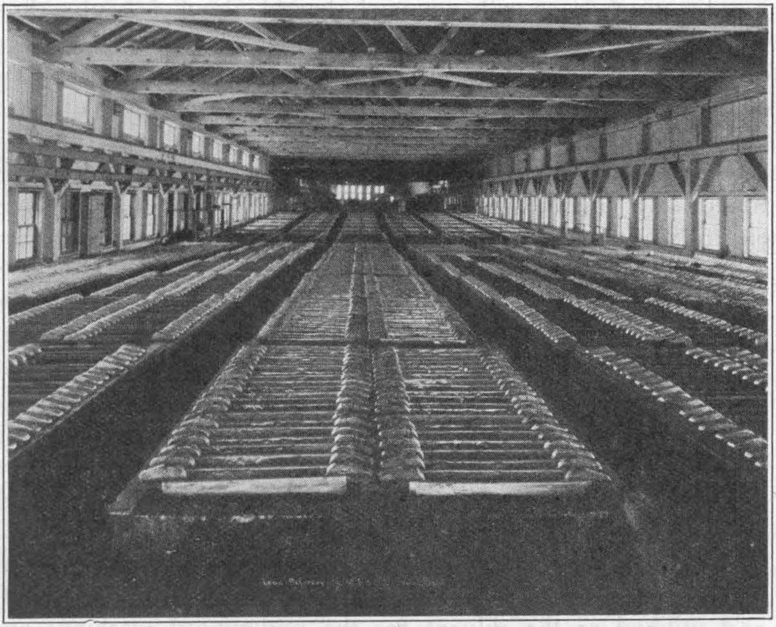


Plate 10

Electrolytic-Lead Room, Lead Refinery, Tadanac, B.C.

treated with compressed air. This poling has the effect of removing a little antimony and tin which enters the cathodes at the operating current densities. After cooling, the lead is pumped by a centrifugal-pump to a movable launder which traverses the circle of moulds. Each pig is skimmed for the removal of dross, and after solidifying is weighed and transferred to railroad-cars for shipment.

The finished product is well over 99.99 per cent. lead, the chief impurities being 0.0025 per cent. antimony and 0.15 ounces of silver per ton. The bismuth and arsenic in the market product are less than 0.0001 per cent.

The electroylte used contains 11 to 13 per cent. total H_2SiF_6 and from 6 to 9 per cent. lead. No purification of the electrolyte is required. The lead content of the electrolyte tends to increase under ordinary operating conditions. The acid loss, averaging 4 to 5 lbs. per ton of pig-lead, is replaced by 33 per cent. H_2SiF_6, which is manufactured in the plant.

To insure a smooth compact deposit about 2/3 of a lb. of glue is required per ton of pig-lead produced. The nature of the tank-linings renders it essential to keep the temperature of the electrolyte below 38° C, otherwise creeping starts. During electrolysis there is an accumulation of neutral lead fluosilicate within the slimes adhering to the anodes. This concentrated salt, together with the adhering electrolyte, and the electrolyte used for scrubbing the slimes from the scrap anodes, is recovered by washing thoroughly in a series of stationary vacuum-filters. Every month a weight of H_2SiF_6 equal to 80 per cent. of the total contained in the tank-room, passes through this washing system. Therefore, it is a first essential that the washing process be adequate and complete in order to avoid losses of acid from this cause. The first washes recovered from the filters contain a greater proportion of H_2SiF_6 than the electrolyte itself. Therefore, the mixed solutions of electrolytic strength are returned direct to the tank-room electrolyte. The intermediates are sent to the evaporators for concentration to electrolytic strength, while the weak wash-waters are held and utilized as the diluent for the preliminary washings on the succeeding day. There is practically no free H_2SiF_6 in the solution sent to the evaporators, so that the evaporation process requires no care. Frequently in the winter time, when natural evaporation is low, the volume of the electrolyte becomes excessive and it has to be evaporated. This is effected at as low a temperature as possible to avoid decomposition losses. The evaporators consist of wooden tanks containing lead-coils through which live steam passes. The lead-coils have a life of about 3 weeks and the wooden tanks about one year.

No innovations have been made in the lead-tank room practice lately. Recent work has centered more on standardization of the actual tank-room manipulations.

Efficient work in the tank-room depends upon the following conditions, given in their order of importance. Efficient tank testing for bad contacts and shorts is a first essential and with this is coupled good circulation of electrolyte, and the presence of the correct amount of glue to give a smooth coherent deposit. Our experience is that after the current density reaches 15.5 amps. per square foot (cathode area) the amount of glue required to obtain the requisite cathode deposit increases rapidly. Thorough recovery of the H_2SiF_6 contained in the slimes, in the filtering and washing process, is, as has already been pointed out, a very important economic factor. Speed and accuracy in removing and replacing anodes and cathodes is also important. Lastly, thoroughness in maintaining the asphalt linings of all tanks, return-launders, aprons and basement floors in perfect order is very important to avoid loss of acid.

The current efficiency is based on pig-lead production and averages about 80 per cent. of the theoretical. As 6 per cent. of the bullion is returned in the form of lead and bullion-drosses to the smelter, about 3 per cent. recovered as slimes which pass for treatment to the silver refinery, and at least 2 per cent. of the total running time is unavoidably lost in cut-out tanks during changing operations, it is apparent that the current efficiency on a strictly electrolytic basis is very good. The cycle of tank-changing operations is never altered except for exceptional causes. Heavy scrap is not reset but is re-smelted. A shortage of bullion receipts appears as a shortage of cathods production 4 and 8 days afterwards.

The chief operating factor utilized to determine the conduct of tank-room at any time is the hourly voltage readings on the generators. With good average work the daily voltage curve should reach a maximum at 7 a.m., then drop irregularly until 1 p.m., during tank changing operations, when tanks are cut-out, etc., and from 1 p.m. to 7 p.m. next morning voltage increase should be gradual but regular every hour.

The major operations in the tank-room are done under contract, payment being for each series of operations and at its own specific rate. The tank-testers, cathode-bar cleaners, tank-cleaners, kettle-firemen and slimes'-washers are all on day's pay. Each new man is given from two to three weeks to learn his particular job and to qualify for piece-rates. The labor turnover after the qualifying period is passed is very small.

Slimes plant: The production of Doré metal from the lead and copper tank-room slimes is effected in small magnesite-lined reverbratory furnaces having a capacity of from 3 to 4 tons of metal. These furnaces are three in number. Two are coal-fired, the third is operated on a semi-gas-producer basis, coke being used as fuel. The sole object of this procedure in the last cast is to insure a permanent oxidizing flame in the reverbratory, which is essential for the successful oxidation of the base metals, particularly copper and bismuth.

The wet slimes are dried and roasted in chambers, which are heated from the waste gases of the reverbratory furnaces. The temperatures of these roasting chambers vary from 300 to 500° F. A period of from 24 to 48 hours is sufficient to oxidize the base metals so that on the melt a 25 per cent. metal-fall is insured and a slag produced which contains the minimum amount of silver. The preliminary melt is effected without fluxes and 90 per cent. of the silver is concentrated in the resultant metal. The slag, composed chiefly of antimonite of lead, carries from 200 to 300 ounces of silver per ton together with small quantities of arsenic, tin, copper and bismuth.

The metal from the initial melt is subjected to an oxidizing heat treatment in the second furnace where the antimony is recovered. The latter is collected in settling-chambers in the form of oxide.. The termination of this stage of the operations is indicated by the formation of litharge on the surface of the bath.

The metal, which now contains about 50 per cent. silver, is transferred to the finishing furnace. Here the oxidizing treatment is continued. In this furnace a jet of air plays on the surface of the molten bath. This provides the oxygen necessary for the oxidation of the remaining base metals and ensures a temperature high enough to keep the bath molten. In this furnace the lead is the first to oxidize and is removed as litharge. As the elimination of the lead proceeds the litharges gradually become coppery. The final copper slags may carry as high as 33 per cent. copper in the form of cuprous oxide. The latter slags contain the major portion of the bismuth in the bullion. Bismuth in the Betts process passes entirely into the slimes and builds up in the system. When the amount becomes excessive special means are taken to effect its removal. The litharge slags are returned to the blast-furnace.

The copper slags are ordinarily returned to the copper-converters. The final copper oxide slags carry a high proportion of silver; in consequence much care is taken at the smelter to eliminate copper from the lead-bullion in the drossing operation and by casting the final bullion at as low a temperature as possible.

The finished Doré metal is from 950 to 970 fine and contains from 1.5 to 2.5 per cent. copper. The slags from the original melt in the first furnace are crushed to ½-inch size and melted with 1 per cent. coal dust and a coke cover. This insures a 10 per cent, metal-fall which carries 90 per cent. of the silver. This metal follows the ordinary routine already described. The slags from this treatment, now containing from 20 to 30 ounces of silver per ton, are again treated as above in a separate furnace. The residual slags from this final treatment, carrying from 3 to 5 ounces of silver per ton, is stored for the production of antimonial lead. The metal from this second treatment carries insufficient silver and too great a proportion of copper to effect the separations in the ordinary manner. With this metal the antimony is burned off and collected in the usual manner, after which the metal is cooled down in the furnace. This effects a segregation of the copper from the lead. The coppery portion is rabbled off and charged to the finishing furnace during the copper elimination stage and the lead metal is returned to the bullion-pot.

The parting is effected by the ordinary sulphuric acid process. The metal is dissolved in iron kettles in 60° Be. acid. The gold sludges are removed to the gold clean-up-kettle. The clear silver solution is syphoned into precipitation tanks, where, after dilution, the silver is precipitated on copper plates. This cement-silver is washed, dried and melted into market bars. The gold-sludges are treated with hot sulphuric acid till free from silver, washed sweet with water, dried and melted for shipment. The copper solutions from the precipitation tanks are evaporated to crystallization point, the bluestone and the acid being recovered as by-products.

GENERAL

The reduction works of the Consolidated Mining and Smelting Company of Canada, Ltd., are situated on a bench overlooking the city of Trail, about 280 feet above the Columbia river. The municipality and railroad stations are called Tadanac, which is the registered trade mark under which all the products of the Company are sold.

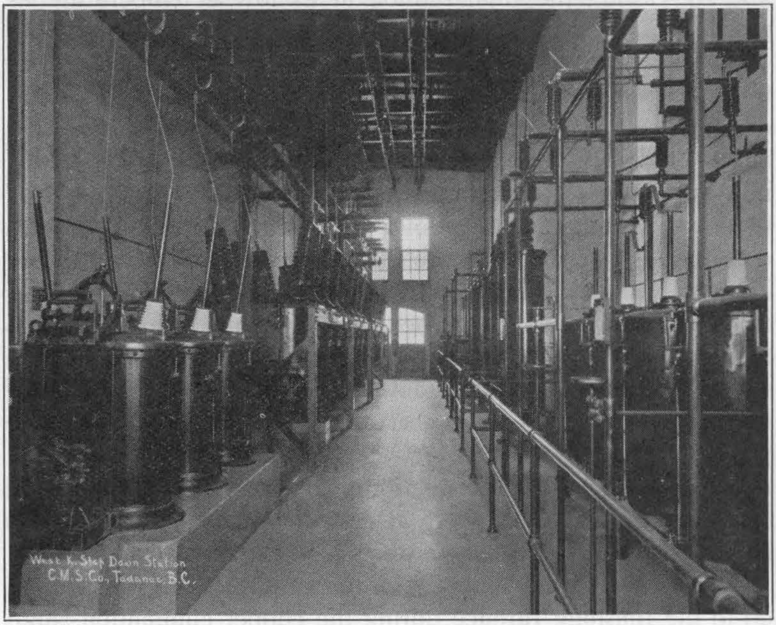


Plate 11

High Tension Switches, West Kootenay Substation, Tadanac, B.C.

The works cover an area of about 250 acres and include, in addition to the departments already mentioned, an acid plant which manufactures sulphuric, hydrofluoric and hydrofluosilicic acids required in the operations. There is also a copper-smelter of four furnaces, two converters, and a refining-furnace as well as a 70-ton copper-refinery, with its melting and casting equipment, and a copper rod-mill.

At the present time, with the copper-plant shut down, there are about 1,200 men on the pay-roll at Tadanac.

The plant is served by the Canadian Pacific railway, but all interdepartmental transportation is accomplished by means of electric locomotives using an 18-inch gauge track. The standard V-bottom cars have a capacity of 48 cubic feet.

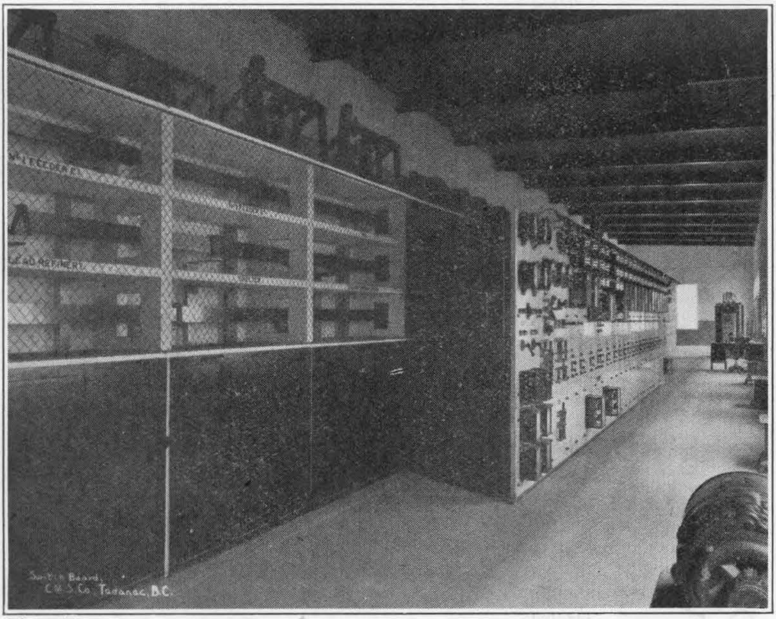


Plate 12

Switch Board and Equipment, West Kootenay Substation, Tadanac, B.C.

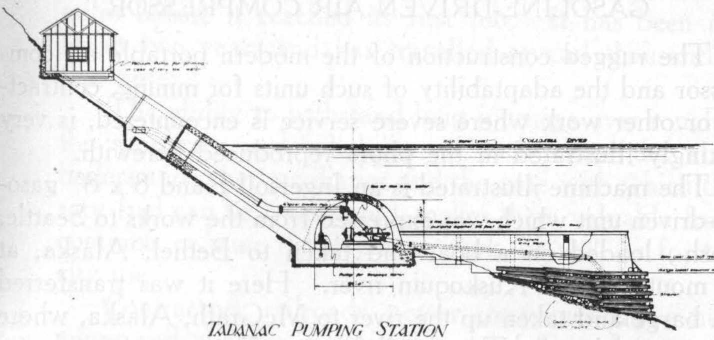
Power is obtained from a subsidiary company, the West Kootenay Power and Light Company, from their plant at Bonnington Falls, some 30 miles away on the Kootenay river. Two lines at 60,000 and two at 20,000 volts deliver power to a central sub-station. Here it is transformed to 2,200 volts and distributed to departmental sub-stations, where it is again stepped-down to 550 volts, which is standard throughout the plant, excepting in the zinc-plant, refinery generator-rooms and rod-mill, where 2,200 volts is used. The total power consumed for the present operations is about 14,000,000 kilowatt hours per month.

Water is delivered by flumes from several creeks in the vicinity, but this having proved insufficient, in 1920 a pumping station which embodies some novel features was installed. At high water the river level is about 30 feet over the floor of the pumping station. As the station was put in during a period of very low water there is never much suction lift on the

TABLE SHOWING ANALYSES OF PRODUCTS AND BY-PRODUCTS MADE FROM SULLIVAN ORE

	Silver, oz. per ton	Copper percent	Lead percent	Zinc percent	Sulphur Percent.	Silica percent	Alumina Percent.	Iron percent.	Manganese Percent.	Lime percent.	Magnesia Percent.	Arsenic Percent.	Antimony Percent.	Cadmium Percent.	Bismuth Percent.
Mill feed	3.0	....	11.0	10.5	....	....	....	36.5	....	....	....	....	....	....	....
Lead concentrates ..	20.0	....	66.0	6.3	18.8	1.0	....	7.5	....	....	....	....	....	....	....
Zinc	2.2	....	5.3	41.0	33.2	1.0	....	19.0	0.4	....	....	....	....	....	....
Zinc residue	4.8	....	10.9	19.1	3.5	2.4	....	37.0	0.6	....	....	....	....	....	....
Lead sinter	....	.05	33.0	8.6	1.5	7.0	....	17.0	....	4.4	....	....	....	....	....
Lead slag	0.15	0.10	1.6	16.2	1.5	18.9	3.5	32.7	....	8.9	....	....	....	....	....
Lead bullion	....	0.04	97.9	....	....	....	....	....	....	....	....	0.057	1.44	....	0.017
Refined lead	0.15	0.001	99.9946	0.001	....	....	....	0.0009	....	....	....	....	0.0019	....	0.0001
Refined zinc	....	0.001	0.02	99.956	....	....	....	0.011	....	....	....	....	....	0.022	....
Fine silver	995.5 fine														
Fine gold	990.0 fine														

pumps. There are three four-stage 10-inch 1,500-gallon Reese roturbo pumps, each driven by a 200 h.p. motor. The lift is 315 feet.



TADANAC PUMPING STATION

Fig. 11

The present production of zinc is about 100 tons per day. The lead production is about 200 tons of bullion, but as soon as sufficient power is obtained to run the Sullivan concentrator to capacity this will be increased to 300 tons per day, as there is sufficient blast-furnace capacity to smelt all the lead concentrates produced.

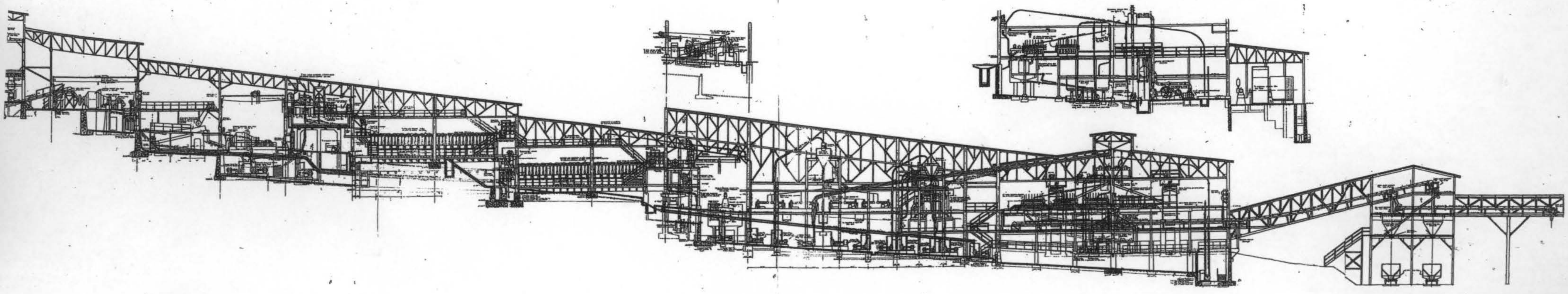


Fig 13

THE SULLIVAN CONCENTRATOR
KIMBERLEY, B.C.

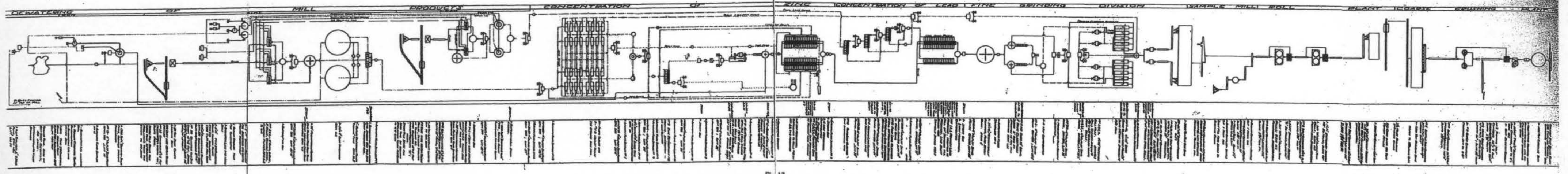


Fig. 12

THE CONSOLIDATED MINING & SMELTING COMPANY OF CANADA, LIMITED
TRAIL, BRITISH COLUMBIA
Flow Sheet of Sullivan Concentrator, Kimberley, B.C.