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CURRENT OPERATIONS AT UNITED KENO HILL MINES

by

The Staff

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October 19th, 1960

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CURRENT OPERATIONS AT UNITED KENO HILL MINES

by

The Staff

A.E. Pike. - General
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ABSTRACT

The United Keno Hill Mines Limited operation is located in the central Yukon 220 miles north of Whitehorse and some 200 miles south of the Arctic Circle. Silver-lead-zinc mineralization in the Galena Hill - Keno Hill area was first discovered in 1906 but the isolated location, long winters and problems of access, transportation, communications, permafrost and costly services slowed the development of the area.

The original discovery and others were subsequently developed and steady production was carried on from 1921 to 1942 when all production ceased. In 1945 United Keno Hill Mines acquired these holdings and commenced production in 1947. The property is currently Canada's largest producer of silver. Total silver production from the present Company holdings is in excess of 110 million ounces.

Geologically the area consists of Precambrian or Palaeozoic quartzites and schists intruded by greenstone dikes and sills. A quartzite member between two schist strata is the most favourable host rock. Vein faults and transverse faults form the major fault pattern of the area.

Ore occurs as lenses along vein faults and attains greatest widths at vein junctions and in loop structures. Primary ore minerals are galena, sphalerite and freibergite with numerous secondary minerals in the oxidized zones.

Currently the Hector, Calumet and Elsa are producing mines with development being done at four other properties. Mining has been carried to a depth of 1165 feet with stoping largely by square sets in the wider ore sections and by a down dip slotting method in the narrow vein sections. In all mines fractured and quite heavy ground is the rule and close timbering is practiced in all workings.

The concentrator, located at Elsa, produces a lead and a zinc concentrate by selective flotation. Flotation tailings are treated in a cyanide plant producing a silver precipitate.

Concentrates are shipped in collapsible containers which are returned and re-used. Precipitates are shipped in drums. Both are transported by truck to Whitehorse, then by railway to Skagway and by steamship to Vancouver for furtherance to the smelters.

Power is purchased from the Northern Canada Power Commission's hydro plant on the Mayo River. Off peak power is used in electric boilers to augment the coal fired boilers at the Elsa and Calumet town-sites. Timber requirements are supplied by local contractors.

The Company has its own transportation division operating out of Whitehorse and handling outgoing shipments and incoming freight, coal from the Company's coal mine at Carmacks, and forest products.

INTRODUCTION

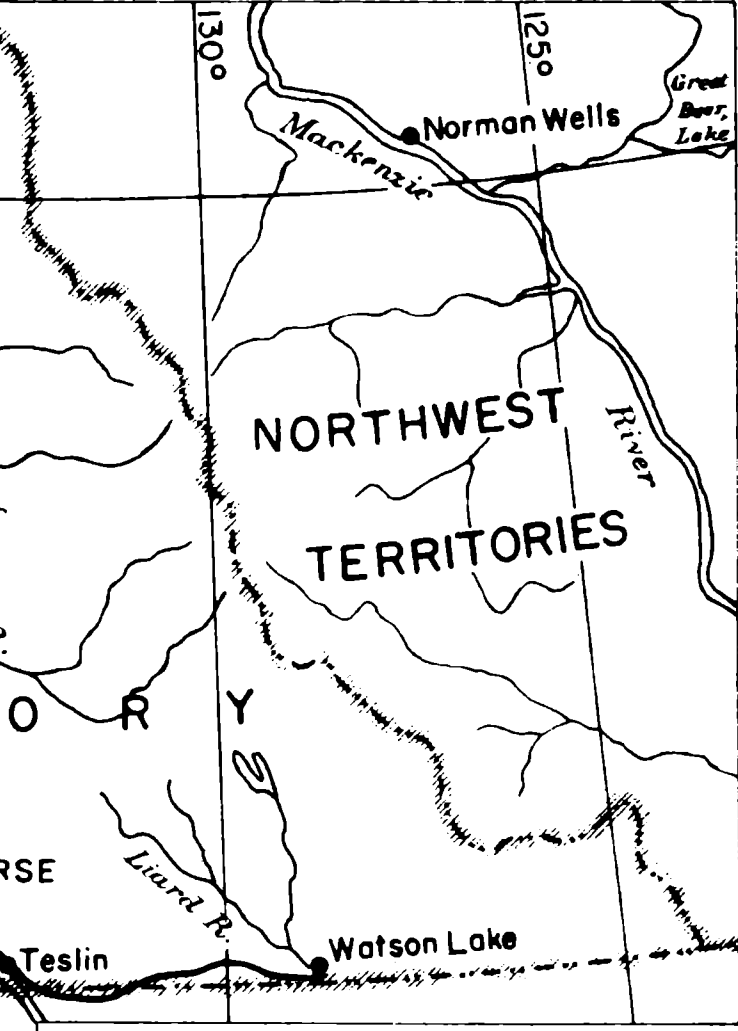
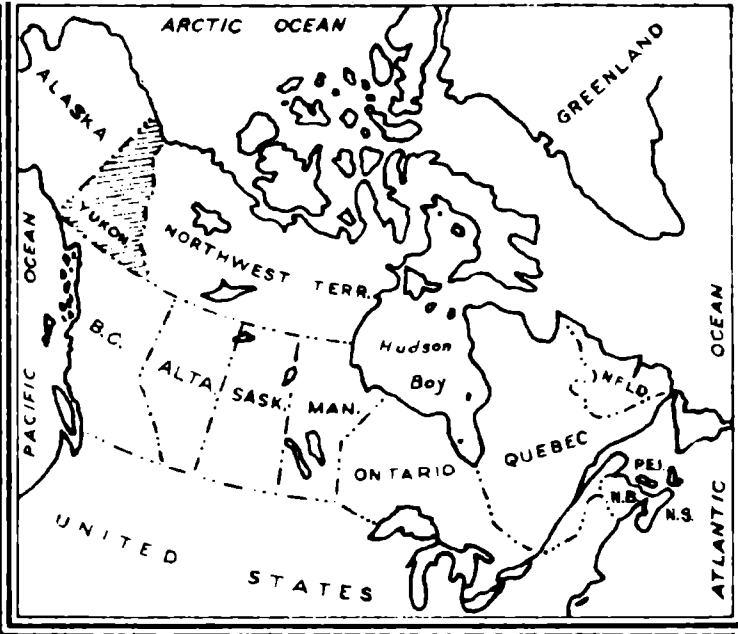
The United Keno Hill Mines operation stems from the discovery in 1906 of silver-lead-zinc mineralization in the Galena Hill area of the Mayo district in the central Yukon. Mining activity progressed slowly until 1923 when the Treadwell Yukon Company commenced important production which later included returns from several properties. Production continued until 1942 when all operations were suspended as most of the known ore bodies had been exhausted, labour was in short supply, metal prices were low and general conditions brought on by the war were very unsettled.

United Keno Hill Mines Limited acquired the Treadwell Yukon and other holdings and since 1946 have carried out a vigorous program of development and mining. Successful development of new ore bodies allowed for important production and today United Keno Hill Mines stands as the largest primary producer of silver in Canada.

The location of the property, the long winters, the presence of permafrost, the generally heavy ground, together with problems of transportation and communications and other factors are, while not peculiar to the operation, most important operating considerations. Reference is made to these factors in describing the United Keno Hill Mines operation in this paper.

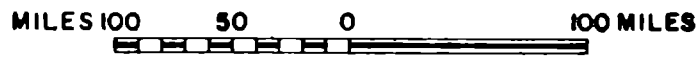
LOCATION

The United Keno Hill Mines Limited operation which includes several small mines, is centred in the Galena Hill - Keno Hill area which lies approximately thirty-two miles north of the village of Mayo on the Stewart



YUKON TERRITORY

Scale: 1 inch to 120 miles



LEGEND

- TRAIL OF '98
- HIGHWAYS
- WHITE PASS & YUKON ROUTE RAILWAY

river and is 220 miles north of the capital city of Whitehorse. The main office and concentrator are at Elsa at $63^{\circ} 51'$ north latitude and $135^{\circ} 31'$ west longitude, thus placing the Company's mining operations within 200 miles of the Arctic Circle (Figure 1).

PROPERTY

The property consists of 625 mineral claims and fractions and covers much of a known mineralized belt over a length of some 16 miles. Many of the mineral claims were secured by the purchase of the property from our predecessors, the Treadwell Yukon Company Limited, and these have been added to by purchases and by staking.

ACCESS AND COMMUNICATIONS

The mining district is served by an all weather road which leaves the Alaska Highway nine miles north-west of Whitehorse. This highway branches at the Stewart River Crossing with the road to the north-west going to Dawson City and the road north-east to Mayo, Elsa and Keno City. Crossings of the Yukon, Pelly and Stewart Rivers are made and until 1959 when the Yukon River was bridged at Carmacks, it was necessary to use ferries in summer and ice bridges in winter. The completion this fall of the bridges across the Pelly and Stewart Rivers has eliminated the use of ferries and closures of the road during freeze-up and break-up periods, and now provide for year round traffic to and from the district.

A bi-weekly air service by Canadian Pacific Airlines DC-3 aircraft serves the Mayo area from Whitehorse. Most personnel, first class mail and urgent repair parts and small supplies are transported by air.

Prior to this fall, communication with the "Outside" was by telephone to Mayo and thence wireless telegraph to Whitehorse. The recently completed land line from Whitehorse by Canadian National Telegraphs now provides direct telephone communication with the rest of Canada.

Completion of the land line also allows for the installation of a low power relay transmitter in the area by the Canadian Broadcasting Corporation. This project, which should be completed shortly, will provide much-needed radio reception in an area where general reception has hitherto been erratic and unreliable.

TOPOGRAPHY AND CLIMATE

The Galena-Hill Keno-Hill area lies in the Yukon Plateaus south of the Ogilvie Mountains and west of the Gustavus Mountains. The plateau is a mountainous region, the majority of the well rounded peaks averaging 5,000 to 6,000 feet above sea level with isolated peaks rising to as high as 6,755 feet. The valleys are wide and flat with the lowest elevation in the area of operations at 2,300 feet above sea level.

There has been heavy glaciation of the lower slopes of the mountains and glacial till and gravel cover the valleys and much of the slopes. Above the 4,500 foot level the mountains are largely covered with float rock and alpine vegetation. Below this elevation there are few rock outcrops due to the almost complete mantle of till, soil, muskeg and vegetation. Vegetation in the area is mainly scrub growth of conifers, birch, aspen, buck brush, alders and willows.

This part of the Yukon is in a region of permanently frozen ground with depths ranging to several hundred feet on the north slopes while some

small areas on the lower south slopes are sometimes free of permafrost.

The climate of the Mayo area is of moderate extremes, with long cold winters and warm and fairly dry summers. The mean annual temperature at Mayo is 26 degrees with a recorded range of from 80 below zero to 95 above. The average annual precipitation is about 12 inches. Snow seldom exceeds 2½ feet on the level and there is comparatively little wind. At Elsa at the higher elevation the temperature range is less extreme and has ranged from 62 below zero to 80 above with somewhat higher precipitation. Days are long in summer with almost total daylight in late June, but shorten to only four hours in December, a climatic feature which has had a marked effect on morale.

HISTORY

The Gold Rush to the Klondike in the 1890's was indirectly responsible for the discovery of the silver-lead-zinc deposits in the Mayo area. While placer gold had been discovered in Duncan Creek on the east side of Galena Hill in 1895, the main body of prospectors entered the area after the turn of the century as production in the Dawson area declined. A high grade silver-lead showing was discovered in Galena Creek in 1906. This later became the Silver King Mine.

The original claim was allowed to lapse and was relocated in 1913 as the Silver King claim. Development work was started and a small shipment of 59 tons of high grade ore which returned \$269.00 per ton was made to the smelter at Trail, B. C. During the winter of 1914-15 new owners worked the property and shipped from a high grade shoot 1,180 tons averaging 270 ounces of silver per ton. Though this shoot was worked out and the property shut down, it supplied the impetus to prospect the area. Prospecting gradually

increased and in 1919 a showing called the No. 9 vein was discovered on top of Keno Hill. This mine produced 9,000 tons of hand cobbled ore averaging 200 ounces of silver per ton. The richness of the ore attracted wide attention and both Keno Hill and Galena Hill were quickly prospected and staked.

The Treadwell Yukon Company entered the area and started mining on the Ladue claim in 1921. This company gradually acquired most of the better showings and commenced shipping hand cobbled ore in 1923 and in 1925 erected the first concentrator. As the good ore in the Ladue, the Sadie-Friendship and Lucky Queen Mines on Keno Hill was exhausted, the mill was moved to the Elsa Mine on Galena Hill and in 1936 commenced production from the Calumet, Elsa and Silver King Mines. Production continued until 1942 when all operations were closed due to exhaustion of ore reserves. Total production by the Treadwell Yukon Company during its twenty-one years of operation totalled 625,000 tons from which was recovered over 44 million ounces of silver and 96 million pounds of lead.

In 1945 Frobisher and Conwest interests formed the Keno Hill Mining Company Limited and purchased the holdings of the Treadwell Yukon Company. The company was later re-organized as United Keno Hill Mines Limited.

Initial work was directed toward the development of the Hector and Calumet Mines and production from these sources started in 1947. Since that time the Elsa Mine has been actively developed and in 1958 commenced production. Extensive development has been carried out on several other properties including the Sadie-Ladue, Shamrock, No. 9 (Keno Mine) and the Onek Mine all on Keno Hill and on the Birmingham and No Cash Mines on Galena Hill. The Onek Mine was developed to the production stage but has not been brought into production due to the low tenor of the ore which runs well to zinc. In 1958 the holdings of the Galkeno Mines Limited, an adjoining property, were purchased by the Company.

GEOLOGY

Much has been written on the geology and ore deposits of Galena and Keno Hills particularly by members of the Geological Survey of Canada. The following report, therefore, dwells briefly on the general geology of the district and is more concerned with the details of the main producing mines which are Hector, Calumet and Elsa.

General Geology

Rock Formations









The area of Galena and Keno Hills is underlain by metamorphic rocks considered to be late Precambrian or early Palaeozoic in age. These rocks can be divided into three general formations (see Figure 2). The lowest is known as the Lower Schist and is composed mainly of thin-bedded quartzite, graphite schist and an upper portion consisting of several hundred feet of sericite-chlorite schist.

The middle member is known as the Central Quartzite and is composed mainly of blue to grey quartzite beds varying in thickness from six inches to five feet or more. The total thickness of the formation approximates 2,600 feet. Most of the individual quartzite beds are separated by one to twelve inches of graphite schist, containing numerous minor drag folds and mullion. The quartzite beds are uniform in character and display few distinguishing features by which they may be used as marker horizons. There is one rock type known in mine terminology as pale siliceous quartzite that is believed to consist of several beds in the rock assemblage. While the ordinary quartzites are a blue-grey color these particular beds are white to pale grey in color. They have been observed underground and mapped extensively at Hector, Calumet and Elsa which are at the same stratigraphic horizon

GALENA AND KENO HILLS

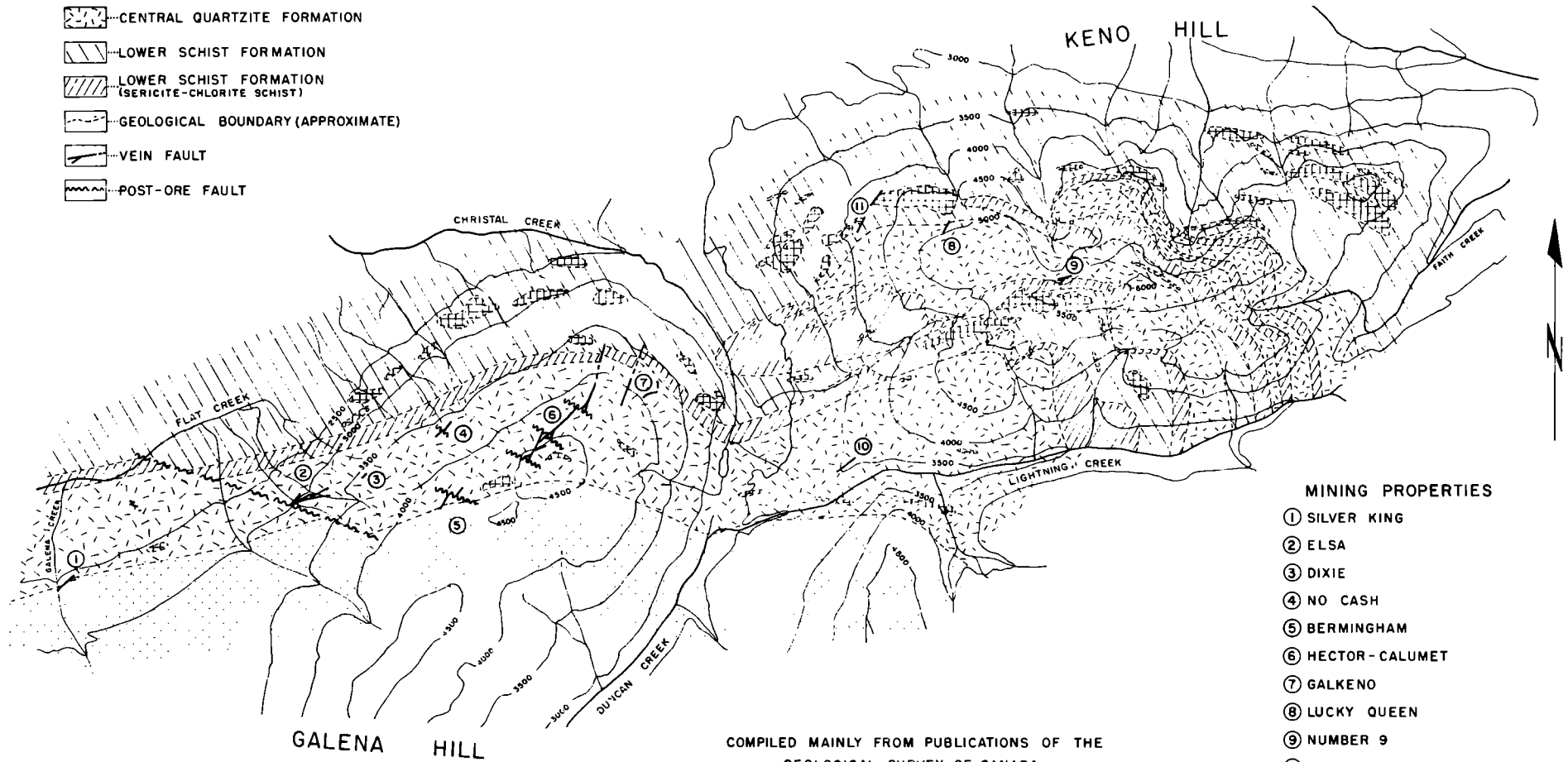
YUKON TERRITORY

LEGEND

-  GREENSTONE
-  UPPER SCHIST FORMATION
-  CENTRAL QUARTZITE FORMATION
-  LOWER SCHIST FORMATION
-  LOWER SCHIST FORMATION
(SERICITE-CHLORITE SCHIST)
-  GEOLOGICAL BOUNDARY (APPROXIMATE)
-  VEIN FAULT
-  POST-ORE FAULT

4000 2000 0 4000 8000

SCALE IN FEET



MINING PROPERTIES

- ① SILVER KING
- ② ELSA
- ③ DIXIE
- ④ NO CASH
- ⑤ BERMINGHAM
- ⑥ HECTOR-CALUMET
- ⑦ GALKENO
- ⑧ LUCKY QUEEN
- ⑨ NUMBER 9
- ⑩ ONEK
- ⑪ SADIE-FRIENDSHIP-LADUE

COMPILED MAINLY FROM PUBLICATIONS OF THE
GEOLOGICAL SURVEY OF CANADA

within the quartzite formation. However, in or near vein zones it is difficult to distinguish these beds from ordinary quartzites that have been leached to a white or pale grey color.

Above the Central Quartzite lies the Upper Schist formation. It, like the Lower Schist formation, is composed of thin-bedded quartzite, graphite and sericite-chlorite schist.

Particularly present in the Lower Schist formation and to a lesser extent in the Central Quartzite and Upper Schist formations are a number of sill-like greenstone intrusives. These were probably diorite or gabbro bodies but have been mainly altered to a chlorite schist. A few small aplite dykes are present, more so on Keno Hill.

The three formations and the greenstone intrusives are conformable, striking almost east-west and dipping twenty to thirty degrees to the south. This assemblage is considered to be the south limb of an open anticlinal structure. In turn this anticline is a smaller one found on the west limb of a much larger anticline extending thirty miles to the south-east from the east end of Keno Hill. North of Keno Hill in the Davidson range there is evidence of overturning in the anticlinal structure but in underground operations at Galena Hill no such evidence has been observed.

The Central Quartzite formation is the most important economically and with a few exceptions contains all the important ore bodies. Of the properties now owned by United Keno Hill Mines Limited, the Silver King, Elsa, Dixie, No Cash, Birmingham, Hector, Calumet and Galkeno on Galena Hill and Lucky Queen, Number 9 and Onek on Keno Hill are situated within this quartzite formation. The Sadie-Friendship-Ladue is an exception where ore occurred in Lower Schist formation. However, there it was closely associated with greenstone bodies.

The Upper and Lower Schist formations are unfavorable host rocks. Some small good grade pods of mineralization have been found but none has proven to be extensive. Veins that can be followed quite easily in quartzite tend to tail out and disappear when the schist formations are encountered.

While greenstone is a fairly competent rock, exploration work to date has proven it to be less favorable than quartzite.

Vein Systems

The veins found in the area can be divided into one system with a strike of N. 60° E. and another with a strike N. 30° to 45° E., both systems dipping from 50 to 80 degrees south-east. These veins are found in older faults which are classified as vein-faults to distinguish them from later offsetting cross-faults. The vein-faults were formed over a long period of time and from slickensides seen in the ore it is evident that movement continued after mineralization.

Mineralization

There have been several ages of mineralization with the first being a quartz, arsenopyrite and pyrite type carrying a little gold. This early mineralization is more evident in the veins on Keno Hill than those on Galena Hill. Further movement re-opened the vein-faults permitting the main carbonate, lead, zinc and silver mineralization.

Hector and Calumet Mines

Vein System

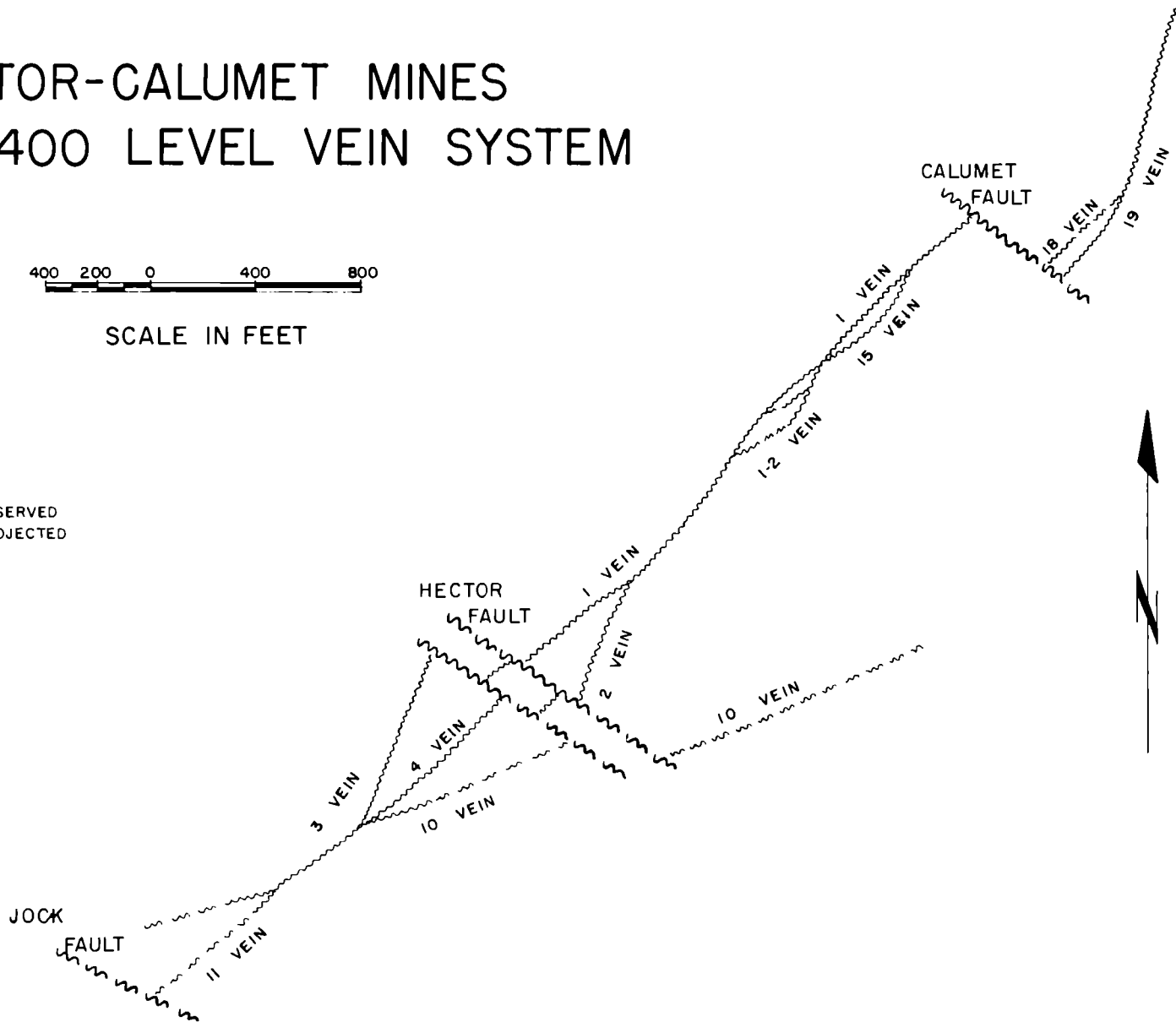
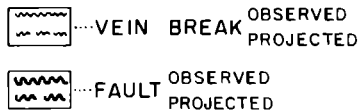
At Hector and Calumet the vein systems have been traced laterally for 6,000 feet. The general strike of the system is N. 45° E. dipping an average of 65 degrees to the south-east. There are many branching veins with some rejoining both horizontally and vertically to form loops. Figure 3 is a plan of the vein system on the 400' or adit level. The 3 and 4 veins with the 1

HECTOR-CALUMET MINES PLAN OF 400 LEVEL VEIN SYSTEM



SCALE IN FEET

LEGEND



and 2 veins form a loop that has been offset 280 feet by the Hector fault. Another loop is formed by the 15 and 1 veins. At the north-east end the vein system continues as the 18 and 19 veins after being offset 320 feet by the Calumet fault. At the south-west end the Jock fault has offset the vein system a distance thought to be 400 feet.

Most of the movement on the vein-faults is horizontal with the south side moving relatively to the east. North-east of the Calumet fault the horizontal displacement along the vein-fault is approximately 700 feet.

The Jock, Hector and Calumet cross-faults strike north-west and dip 45 degrees to the south-west. They are normal faults with right hand displacements. The fault zones consist of five to thirty feet of crushed quartzite, graphite and gouge. Little is known of the Jock fault but both the Calumet and Hector faults develop footwall branches at depth.

Ore Bodies

Ore bodies vary in size from fifty to hundreds of feet vertically and laterally with widths from five to fifty feet. They are found irregularly along strike and dip and do not form any recognizable pattern. In a few cases some seem to follow the intersection of the vein-fault with the bedding. However, the random distribution of ore bodies along a strike length of 2,300 feet and a dip distance of 1,100 feet within the 1 vein does not indicate any favorable series of beds. Figure 4 is a longitudinal section of the 3 and 1 veins showing the positions of the ore bodies.

Most ore occurs as deposition in open fractures so structures producing these openings are the most favorable. This control is apparent at junctions where veins have joined at an acute angle so that in addition to two main outside veins there are numerous cross-over or tension fractures.

Wide ore bodies result when enough of these fractures become mineralized.

The best example of structural control is in the 3 vein where ore is continuous from surface to just below the 900 level. Above the 650 level the widest ore is contained in the junction area of the 3 and 4 veins. From the 650 level down the widest ore is within the 3 vein in a loop structure which reaches its maximum width of 57 feet on the 775 level. Other examples are the 1 and 2 vein junction, the 1 and 15 vein loop and the 9-1A-556-573 stope area, another narrow loop structure which has produced over 60,000 tons from the 900 to the 775 levels.

Mineralization

The main ore minerals are silver-bearing galena, sphalerite and freibergite which occur in a gangue of manganiferous siderite with some quartz and brecciated or leached quartzite. Galena varies from a fine-grained to a coarse-grained type commonly with a laminated appearance. Sphalerite varies in color from a light to dark brown and contains appreciable cadmium but little silver. Freibergite occasionally occurs disseminated in a carbonate gangue but is usually intimately associated with galena. Minor ruby silver is present at various locations, some of these being the 775 and 1040 levels. It has been observed in depth to the 1165 level. Coarse crystalline pyrite may occur anywhere within the ore bodies but the massive fine-grained variety generally occurs at the extremities. Within ore bodies the presence of pyrite bears no relation to silver values.

In many of the large ore bodies such as the 3 vein where massive sulphides attain a width of over 50 feet, galena and sphalerite occur in alternating bands several feet thick but in many of the smaller ore bodies the ore minerals are widely disseminated.

Oxidation extends to a depth of approximately 400 feet and within this zone secondary minerals such as cerussite and anglesite are common with some native silver and argentiferous jarosites. There has been some enrichment of silver but the effect is not significant. One noticeable feature is the absence of zinc. The zone is characterized by the presence of considerable limonite and black manganese stain produced by oxidation of manganiferous siderite.

Within the vein zone the quartzite may be present as hard brecciated fragments cemented with carbonate or it may be leached to a loosely consolidated sand. In some cases the quartzite fragments retain their strength but have been altered to a white color. Beyond the vein zone the quartzites are mostly fresh and unaltered. Occasionally they have been fractured and contain veinlets of carbonate with some galena. If the wall rock contains much graphitic material it tends to be contorted and dragged into the vein.

Zoning Effects

The following table shows the aggregate tonnage produced by levels from the various veins. With the exception of the 15 vein the figures given for the 100, 300 and 400 levels are only for a portion of the ore mined but they are indicative of what enrichment there has been. The veins from the 525 level down are below the zone of oxidation and not influenced by surface enrichment.

HECTOR AND CALUMET MINE PRODUCTION

LEVEL	1 VEIN				2 VEIN				3 VEIN				4 VEIN				15 VEIN			
	Tons	Ag,oz.	Pb,%	Zn,%	Tons	Ag,oz.	Pb,%	Zn,%	Tons	Ag,oz.	Pb,%	Zn,%	Tons	Ag,oz.	Pb,%	Zn,%	Tons	Ag,oz.	Pb,%	Zn,%
100	7,453	41.0	5.7	1.0	3,774	36.6	6.0	3.7	1,680	21.4	5.5	0.2	5,996	29.5	5.4	1.1	1,028	36.3	4.6	-
300	29,945	32.3	4.9	1.2	10,283	27.5	4.7	4.2	11,523	54.4	9.8	1.9	50,886	43.4	8.6	4.7	10,900	26.2	3.6	0.3
400	12,252	23.7	2.8	3.3	19,265	32.3	5.3	6.2	10,389	63.1	14.9	8.1	67,433	34.8	6.1	6.3	29,379	37.1	5.6	0.3
525	53,436	33.4	5.0	2.0	24,632	27.2	4.8	5.9	51,290	39.8	12.0	8.5	43,993	38.2	8.6	6.7	38,855	42.7	6.1	0.3
650	32,786	30.9	6.2	5.6	30,552	39.7	8.5	8.1	91,030	39.2	12.0	8.9	23,958	30.3	5.9	6.4				
775	70,329	32.4	6.9	7.6	67,795	41.4	9.2	8.6	132,214	42.0	11.9	12.8	15,065	38.3	7.7	7.0				
900	90,648	34.9	8.8	12.0	29,402	31.1	6.7	8.2	60,768	34.0	9.9	14.2								
1040	14,130	41.8	10.6	11.7																

TABLE I

The tonnages and grades for the 3 vein from 525 level to the 900 level represent the bottom part of the largest continuous ore body in the mine.

The figures for the 2 vein, 4 vein and 15 vein represent continuous ore bodies while those for the 1 vein are totalled from a number of individual ore bodies most of which are not over 200 feet laterally or vertically (see Figure 4).

For individual ore bodies there is a zoning effect with a definite increase in zinc and a decrease in lead and silver content at the bottom

HECTOR-CALUMET MINES LONGITUDINAL SECTION I AND 3 VEINS

LOOKING WEST



SCALE IN FEET



although the lead-silver ratio remains almost the same. In the 1 vein which represents totals of many individual ore bodies the aggregate content of silver, lead and zinc increases with depth. The silver-lead ratio shows a decrease from approximately 6 ounces of silver to one per cent lead for the 525 and 650 levels to 4 ounces of silver to one per cent lead for the 1040 level.

In summing up it may be said that although individual ore bodies tend to bottom with high zinc and lower silver values there is no reason why other ore bodies found at lower depths should not be average mine grade.

Zoning

Elsa Mine

Vein System

At Elsa the vein system is more complex than at Hector and Calumet. Figure 5 shows the various veins as they are known on the 200 level. The 2 vein strikes N. 70° E. and dips 65 degrees to the south. Branching from it is a vein striking N. 45° E. dipping 65° S. This in turn has another branching vein, the 15, striking N. 20° E. and dipping reversely 40 degrees to the west. In addition, there are numerous other branching structures with various dips and strikes. Some veins have attitudes so close to the bedding that they follow along it for short intervals.

The vein system has been traced for 2,300 feet laterally. It has been offset by two east-west cross-faults, the Brefalt Creek Fault at the south end of the veins and the Porcupine Creek Fault 650 feet farther north. The Brefalt Creek Fault dipping 50 degrees to the south consists of a crushed zone 120 feet wide. While the vein system has not been located south of the fault the offset is indicated by surface geology to be at least 2,000 feet.

ELSA MINE

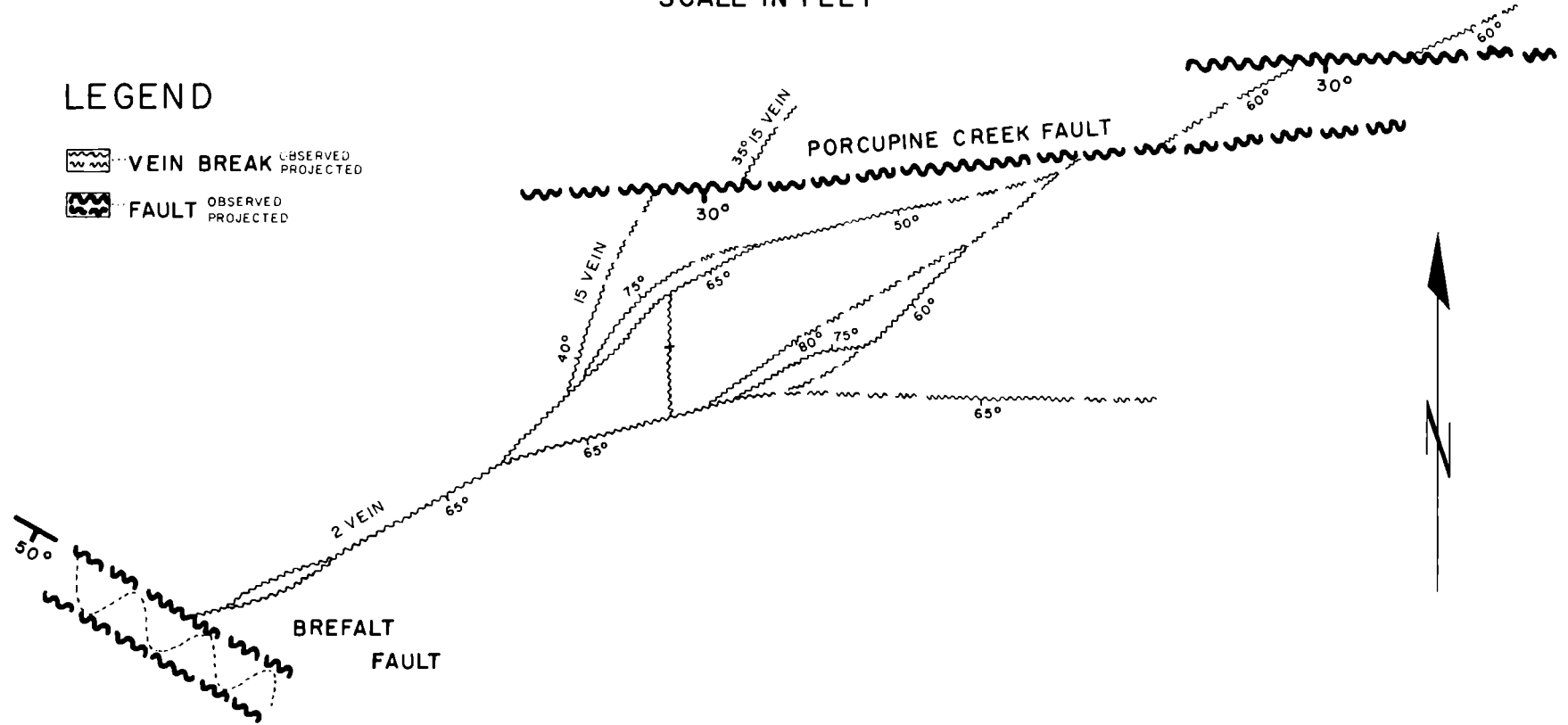
PLAN OF 200 LEVEL VEIN SYSTEM

200 100 0 200 400

SCALE IN FEET

LEGEND

-  VEIN BREAK OBSERVED PROJECTED
-  FAULT OBSERVED PROJECTED












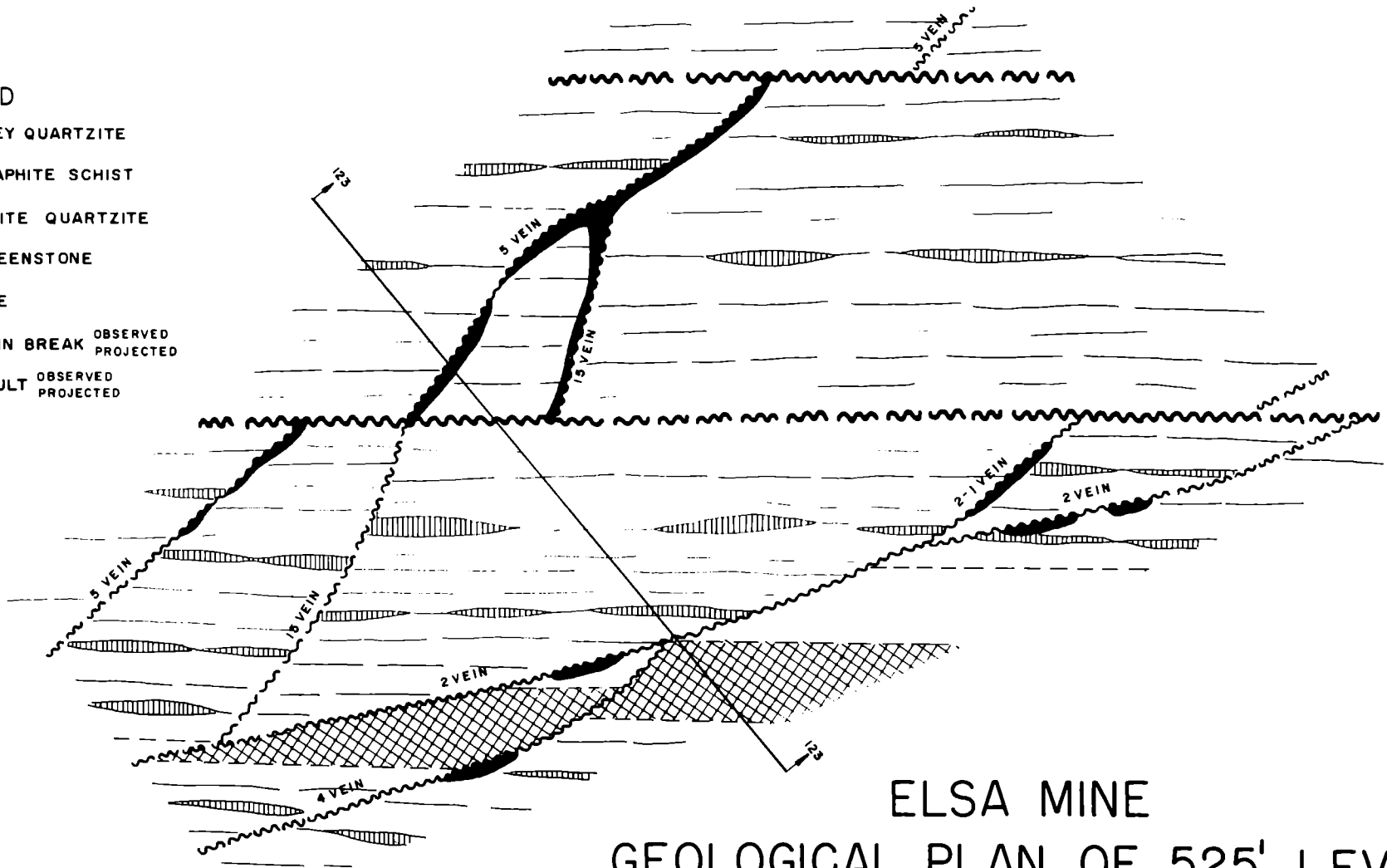
The Porcupine Creek Fault dips 30 degrees to the south making it almost a bedding fault. The fault zone consists of one to twenty feet of graphite and crushed quartzite. From the displacement of the veins and mullion seen in the fault zone it would appear that the fault is produced by thrust resulting in a right-hand horizontal offset of 90 feet. Figure 6 is a section illustrating how the veins have been offset by the Porcupine Creek Fault.

Ore Bodies

The most important location of ore appears to be at vein junctions although numerous ore bodies are found elsewhere in the veins. Much of the present work in the mine is confined to the exploration of the veins in the footwall of the Porcupine Creek Fault. In this area the largest ore bodies are found at the junction of two veins, the 5 vein striking N. 60° E. dipping steeply south-east and the 15 vein striking N. 10° E. dipping reversely 30 degrees to the west. At the junction of the two veins the quartzites have been shattered and filled with numerous veinlets of galena and freibergite. There, ore has attained a width of 40 feet and a height of 45 feet, also extending outward in the individual veins to a distance of 100 feet. Figure 7 is a plan of the vein system on the 525 level and Figure 8 represents a section through the vein junction showing the concentration of ore. This structure has been traced from the 200 to the 650 level along a slope distance of 1,200 feet. A longitudinal section along the steep-dipping 5 vein is shown in Figure 9 with ore at the junction of the two veins shown by cross-hatching.

LEGEND

-  GREY QUARTZITE
-  GRAPHITE SCHIST
-  WHITE QUARTZITE
-  GREENSTONE
-  ORE
-  VEIN BREAK OBSERVED
-  VEIN BREAK PROJECTED
-  FAULT OBSERVED
-  FAULT PROJECTED



ELSA MINE
GEOLOGICAL PLAN OF 525' LEVEL



SCALE IN FEET

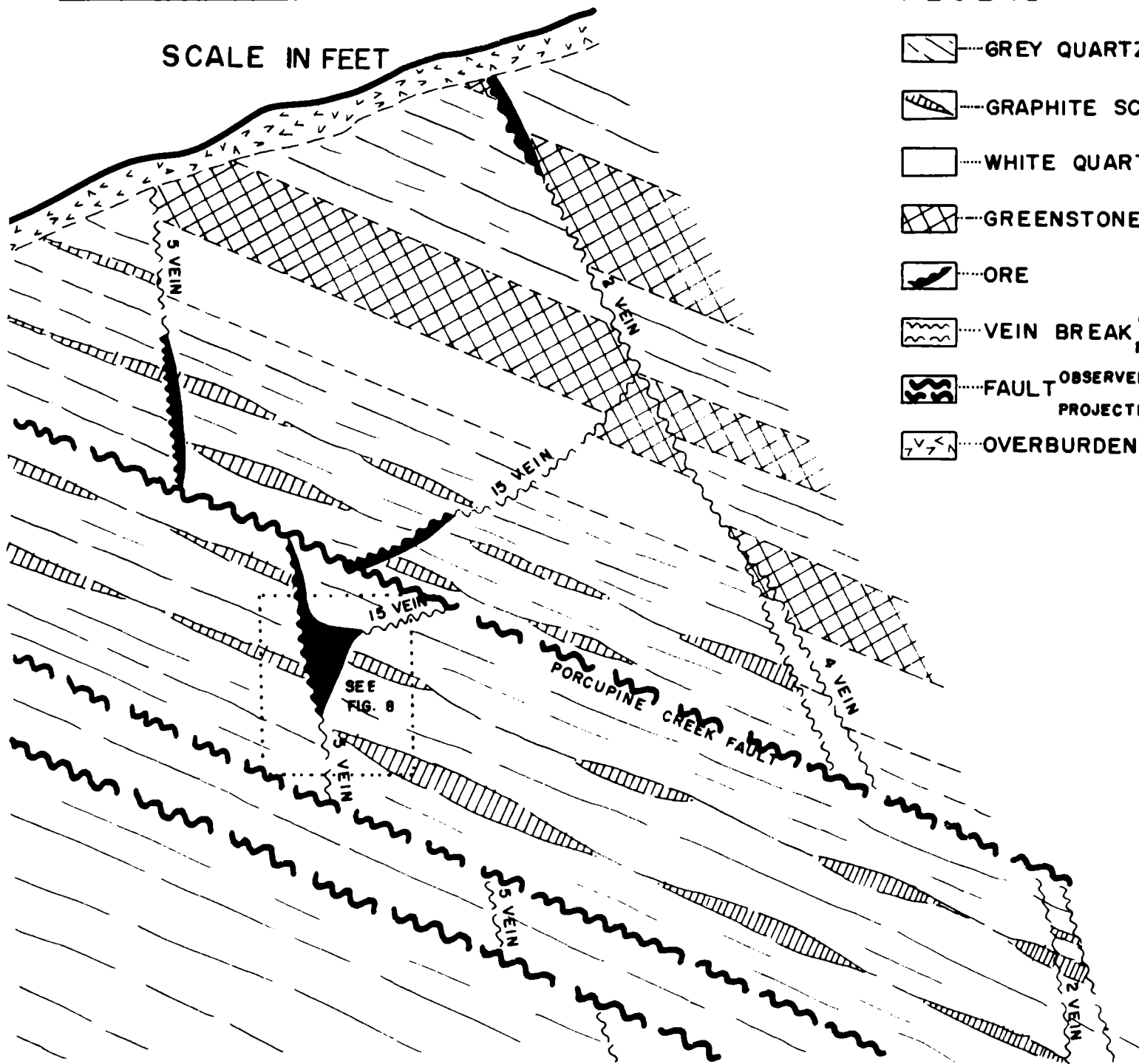
ELSA MINE

VERTICAL SECTION 123









LOOKING NORTH

100 50 0 100 200

SCALE IN FEET



LEGEND

-  GREY QUARTZITE
-  GRAPHITE SCHIST
-  WHITE QUARTZITE
-  GREENSTONE
-  ORE
-  VEIN BREAK
-  FAULT
-  OVERBURDEN

OBSERVED
PROJECTED

OBSERVED
PROJECTED

200' LEVEL

400' LEVEL

525' LEVEL

650' LEVEL

775' LEVEL

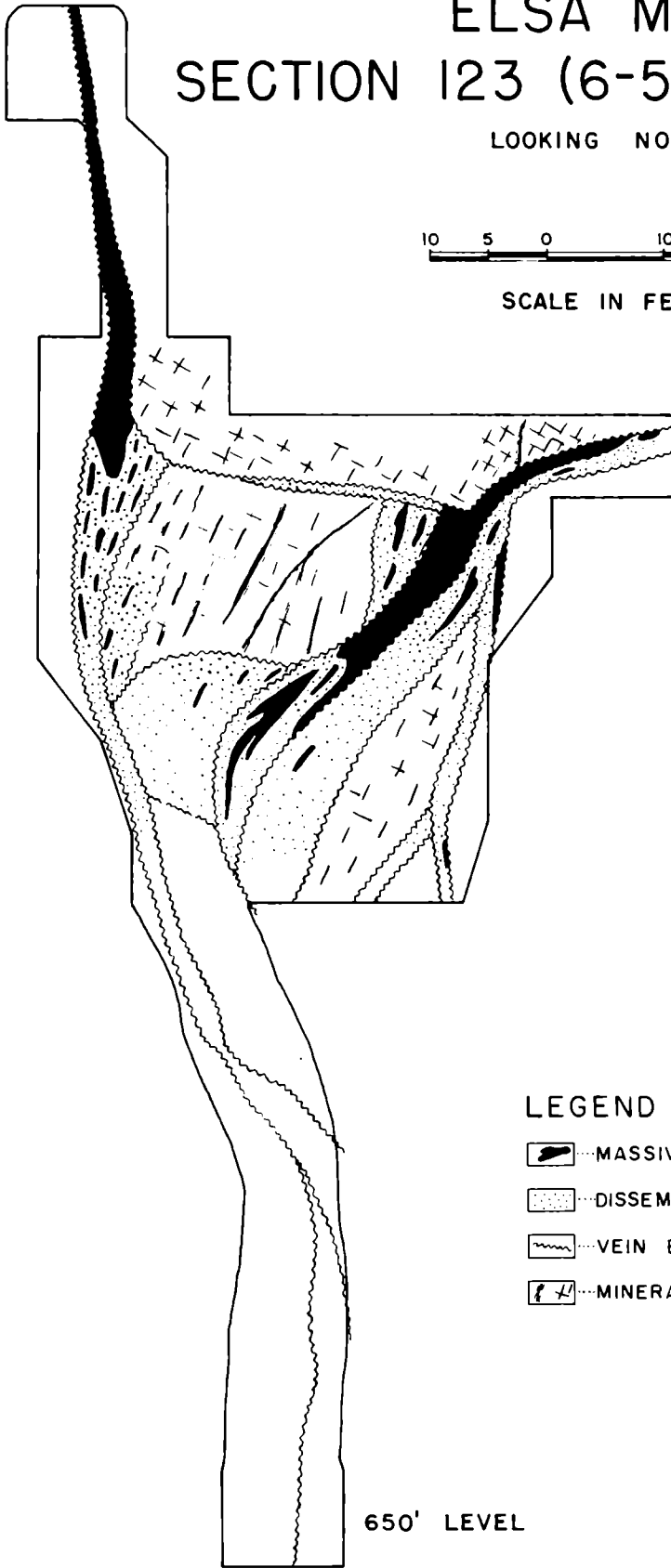
525' LEVEL

ELSA MINE SECTION 123 (6-5-235 STOPE)





LOOKING NORTH



SCALE IN FEET



LEGEND

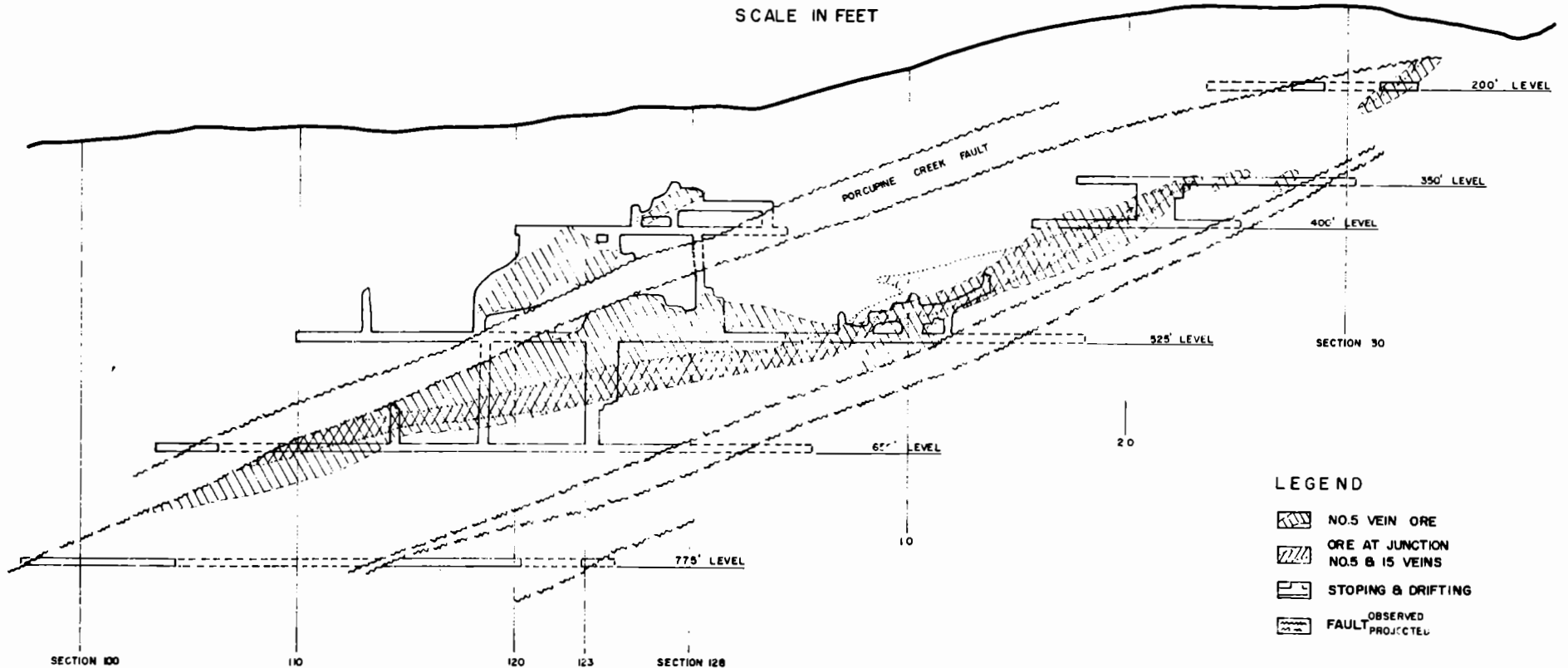
-  MASSIVE SULFIDES
-  DISSEMINATED SULFIDES
-  VEIN BREAK
-  MINERALIZED JOINTS

ELSA MINE LONGITUDINAL SECTION NO. 5 VEIN





LOOKING WEST



SCALE IN FEET



LEGEND

-  NO. 5 VEIN ORE
-  ORE AT JUNCTION
NO. 5 & 15 VEINS
-  STOPPING & DRIFTING
-  FAULT
OBSERVED
PROJECTED

Mineralization

Mineralization at Elsa differs markedly from that of Hector and Calumet. There is minor zinc and the lead content is lower while the silver content is higher. Silver is derived mainly from abundant freibergite which occurs in almost solid veinlets and as irregular masses in carbonate and quartzite with or without galena. When occurring with galena it is usually so coarse that it can easily be distinguished megascopically. More rarely it is closely associated with pyrite. Minor chalcopyrite is associated with freibergite. Galena is commonly medium to coarse-grained and exhibits a laminated appearance. It occurs as disseminated particles, small veinlets and as massive bands up to 4 feet thick. The little zinc present is in the form of dark sphalerite associated with galena.

Oxidation extends to approximately 400 feet below surface and in this zone along with primary freibergite and galena are found secondary native silver, ruby silver, and argentiferous jarosites all of which contribute significantly to the total silver content. Malachite staining is common throughout the zone and is a good ore indicator. As in Hector and Calumet the effect of surface enrichment is not too significant. The secondary silver minerals are partly deposited in joints within the quartzite wall rocks resulting in more ore rather than an enrichment of existing veins.

Ore is contained in a gangue of manganiferous siderite, brecciated quartzite and minor quartz. There has been little leaching and the quartzite fragments within the vein zone are hard and fresh. Occasionally a vein will appear not as regular vein but rather a quartzite horizon mineralized with numerous small veinlets of galena and freibergite. Outside the vein zone

the quantities are unaltered, although in places heavily jointed. Where thick bands of graphitic schist are present beds are greatly distorted by movement along vein faults.

Too little tonnage has been mined to come to any definite conclusions regarding zoning effects but some interesting features are evident. To the depth of the 650 level, which is 500 feet below surface, the silver content has varied very little. It ranges from a low of 61.6 ounces per ton on the 400 level to a high of 69.5 ounces per ton on the 650 level.

The lead content varies from 2.6 percent on the 400 level to 8.0 percent on the 525 level. It would appear, therefore, that there is little relationship between the ounces of silver and the percentage of lead. This reflects the occurrence of freibergite which is the source of most of the silver. As mentioned previously, freibergite occurs in veinlets and irregular masses with or without galena.

This lack of relationship between silver and lead contents is in direct contrast to the ore at Hector and Calumet mines where the ratio is approximately 4 to 5 ounces of silver to 1 percent lead.

Sphalerite is nowhere abundant and zinc reaches its highest content of only 0.2 percent on the 525 level.

Mapping

The complex nature of the vein system along with extensive use of timber requires every development heading and most stopes to be mapped each day so that any horizontal or vertical branching vein may not be missed.

In square set stopes every second vertical slot is mapped giving a complete section of the stope every eleven feet. In longwall stopes the exposed ribs are mapped as mining progresses downward. Face-mapping is carried on in drifts and the geology at waist height transferred to level plans. Waist height mapping is used because in re-mapping and checking certain areas the walls can be seen through the timber but the backs are almost completely obscured by tight lagging. Underground mapping is done on 20 scale and then transposed to permanent maps consisting of 40 and 100 scale level plans and 40 scale sections spaced at 50 feet intervals.

A surface area of almost 30,000 acres has been mapped on 100 scale but because of scarcity of outcrop it is difficult to do any detailed work.

Diamond Drilling

Two underground drills operated by Boyles Bros. are stationed on the property, one at Calumet Camp and one at Elsa, although at various times they may be moved to other underground operations. By use of the non-rotating AXF core-barrel in conjunction with the standard AX it is possible to average 75 to 80 per cent core recovery. In solid galena and sphalerite or carbonate with disseminated sulphides core recovery is good but in crushed leached vein material it is poor. Sludges are taken every five feet and these in conjunction with core give satisfactory results. Much ore at Hector and Calumet and a large portion of the ore at Elsa was found initially by diamond drilling.

MINING

Mining at United Keno Hill Mines Limited is noteworthy chiefly for the soft heavy ground and the large quantity of timber needed for

ground support.

The producing mines are the Hector, Calumet and Elsa, with the Hector and Calumet being the main producers at a rate of approximately 12,000 tons per month. The Elsa mine produces about 2,800 tons per month.

Access to the vein zones in the Hector and Calumet mines is by way of a 1,600 foot adit on the 400 level of the Hector mine with an internal shaft extending 930 feet below this level. A number of service raises and manways extend to surface from the upper levels. The Elsa mine is serviced by adits on the 200 foot and 400 foot levels with an internal shaft extending from the 400 foot level to the 775 level.

The Hector and Calumet mines, with the more difficult ground, consume about 30 board feet of timber per ton of ore mined. At the Elsa mine, with slightly better ground, the consumption is about 25 board feet per ton mined. These figures represent stulls, lagging and rough lumber including square sets, and includes only that timber actually used underground.

All rough lumber and round timber used underground is spruce, cut, sawn and delivered to the mine timber yards by local contractors. (See Table 2) Timber yards are large enough to carry supplies to cover the periods cutters cannot haul during the spring breakup and the winter freezeup. Usable timber stands occur on the valley bottoms along the courses of creeks and rivers. Fortunately, adequate supplies are available locally as the cost of importing the quantity needed would be prohibitive. Shaft sets and shaft guides are imported B. C. fir. Planed guide timber costs approximately \$500.00 per 1,000 bd. ft.

APPROXIMATE COST OF TIMBER LAID DOWN AT MINE

Round Timber

Lagging (4-6 inch tops)	4 ¢ per lineal foot
Cribbing (6-8 inch top)	12 ¢ per lineal foot
Stulls (8-10 inch top)	19 ¢ per lineal foot
Stulls (10-12 inch top)	25 ¢ per lineal foot

Rough Lumber

8' x 8'	\$77.00	per 1,000 bd. ft.
6 inches and over	\$88.00	per 1,000 bd. ft.
under 6 inches	\$100.00	per 1,000 bd. ft.

All timber normally delivered in 16 ft. lengths.

TABLE II

Ladders, wedges, and square sets for all the mines, are prepared in the framing shop at Calumet. At the Hector and Calumet mines a crew of twelve men, under a work boss, load and deliver timber to the working places. Drift caps and posts are cut to length with an electric chain saw. Round timber, lagging, square sets, etc. are loaded on timber trucks which are made up into trains. The first train reaches the shaft and the timber is ready for lowering after waste drift rounds have been hoisted.

All development and production timbering is done by the mining crews. In heavy ground, timber repair is extensive and costs about \$0.91 per ton mined.

To increase the strength of the drift caps in heavy ground rail

beams are used in combination with timber caps. The rail beam is installed on top of the drift posts, and the round timber cap is put on top of the beam. The rails are either triple 45 pound rail or double 56 pound rail welded at the ends and clamped together at the center with a strap loop and bolt. Initially these beams were also welded in the middle but under pressure the beam failed at the center weld. Since the center clamp replaced the weld, failures are rare.

Permafrost

Most of the underground workings in the Keno Hill district have encountered permanently frozen ground or permafrost. The depth of the permafrost varies from 0 to 450 feet, and appears to be determined by a number of different factors; character of the bed rock, depth and character of overburden, amount of exposure to the sun (south slope or north slope), and the leaching action of surface waters.

Rock temperatures below freezing provide many problems. Water lines must be completely drained after use. Surface seepage and drilling water will freeze in ditches, on track and on ladder rungs. Diamond drilling must be done on a 24 hour cycle or water must be kept in circulation down the hole at all times. In severe cases calcium chloride or other chemical agents must be added to the water. At times drilling water, with chemicals added, must be transported to the site in mounted water tanks.

Permafrost, in certain oxidized, fractured and crushed ground, can be an aid in the preliminary development and production. It acts as a cementing agent and ground that would otherwise be considered difficult can be mined with a minimum of timber. This type of ground, when frozen, tends to be "rubbery" and difficult to break, but proper attention to

drilling and blasting practices eliminates this problem. In frozen areas where the levels are connected and normal ventilation is established, the permafrost starts to retreat and a considerable amount of material begins to spall from ribs and back. It has been found by bitter experience that frozen ground, despite its apparent competence when first opened, must be tightly blocked in anticipation of rock conditions after the retreat of permafrost.

The presence of permafrost presents certain difficulties in stoping. Broken ore will freeze in place in the stope or in mill holes, therefore mill holes must be pulled on a regular schedule and, in severe cases, be left empty. Shrinkage stoping has been avoided because of the freezing problem although one stope at the Elsa mine has been mined by this method with little trouble. It is interesting to note that the Treadwell Yukon Company, at the Silver King mine, used shrinkage stoping but abandoned it below the permafrost horizon because of the increased dilution from the hanging wall.

In permafrost areas any entry subject to seasonal thawing, or open to surface drainage, will gradually fill with ice. When the new adit at the 200 level of the Keno mine entered the old workings of Keno Hill Limited, closed in 1923, these were found to be completely ice filled. It was necessary to mine the ice to examine the old workings but it was found that the ice had served as a support and preservative, and the headings and timber were in good condition.

Haulage

In all producing mines, track gauge is 18 inches. All present drifting is done using 20 pound rail but in some of the older workings 16

pound rail is still in use. Thirty pound rail is used for main haulage track.

At the Hector and Calumet mines, ore is pulled from hand chutes and trammed to the main ore pass system at No. 3 shaft. Trains of 30 or 26 cubic foot rocker dump cars are hauled by 1.5 ton Mancha battery locomotives. The ore pass system connects all producing levels below the 400 (Adit) level, and feeds to twin measuring pockets at the shaft below the 1165 level. From the measuring pocket the ore is hoisted in 40 cubic foot bottom dump, skip-cage combinations, to an ore pocket above the 400 level.

On the 400 level the ore is pulled from an air operated, undercut, arc gate chute. Trains consist of six 60 cubic foot Granby type cars hauled by a 3.5 ton Mancha battery locomotive. The ore is hauled 2,690 feet to the outside bins at the upper tramline terminal.

Excess development waste is car hoisted to the deck where it is dumped into a waste pocket. Waste is drawn from an air operated, undercut arc gate chute. The waste train consists of ten 30 cubic foot rocker dump cars hauled by a 3.5 ton Mancha battery locomotive. The waste is hauled 3,000 feet to the waste dump.

At the Elsa mine ore and waste produced below the 400 (adit) level is car hoisted to the 400 level. All haulage is handled by 1.5 ton Mancha battery motors hauling 26 cubic foot rocker dump cars. The ore is trammed 1,120 feet to the outside bin above the mill crushing plant.

At the Hector and Calumet mines tramping crews work on a different shift from the regular mining crews. This allows continuous use of the available haulage equipment. Entering the mine about four hours after the regular crews, the tramping crews are able to utilize all haulage equipment after the drift development crews have finished their mucking cycle.

Ventilation

Ventilation in all operating mines is natural, downcasting from upper surface openings in summer and upcasting in winter. Ventilation doors are installed where required and are used to control air volume and distribution throughout the mine. There is a short period, during the spring and fall, when the ventilation is relatively poor. For most of the year, large variations in the outside and mine temperatures and appreciable differences in elevation between various mine entries, provide a satisfactory motive column.

Extremely low winter temperatures cause some icing problems in the incast entries. In all mines this includes the main adit with the main mine drainage ditch. The problem is handled in several ways. Incast air is restricted and at the Elsa Mine heated by a steam space heater. At the Hector mine there is sufficient heat from the main air line to slightly warm the restricted flow of air. In addition, mine drainage water is caught in a sump, well inside the mine, and pumped outside at sufficient velocity to prevent freezing.

Certain dead end headings are supplied with mechanical ventilation by 16 inch Meco air-driven fans connected to 10 inch collapsible rubber tubing. Long exploration adits are supplied with air by 7 1/2 h.p. electric fans connected to 15 inch galvanized ventilation pipe. It is interesting to note that most headings, without mechanical ventilation, develop an appreciable indraft along the back and outdraft along the floor. This is most noticeable in new drifts in which the rock has not been warmed appreciably and is probably due to the cooling action of rock on the generally warmer mine air.

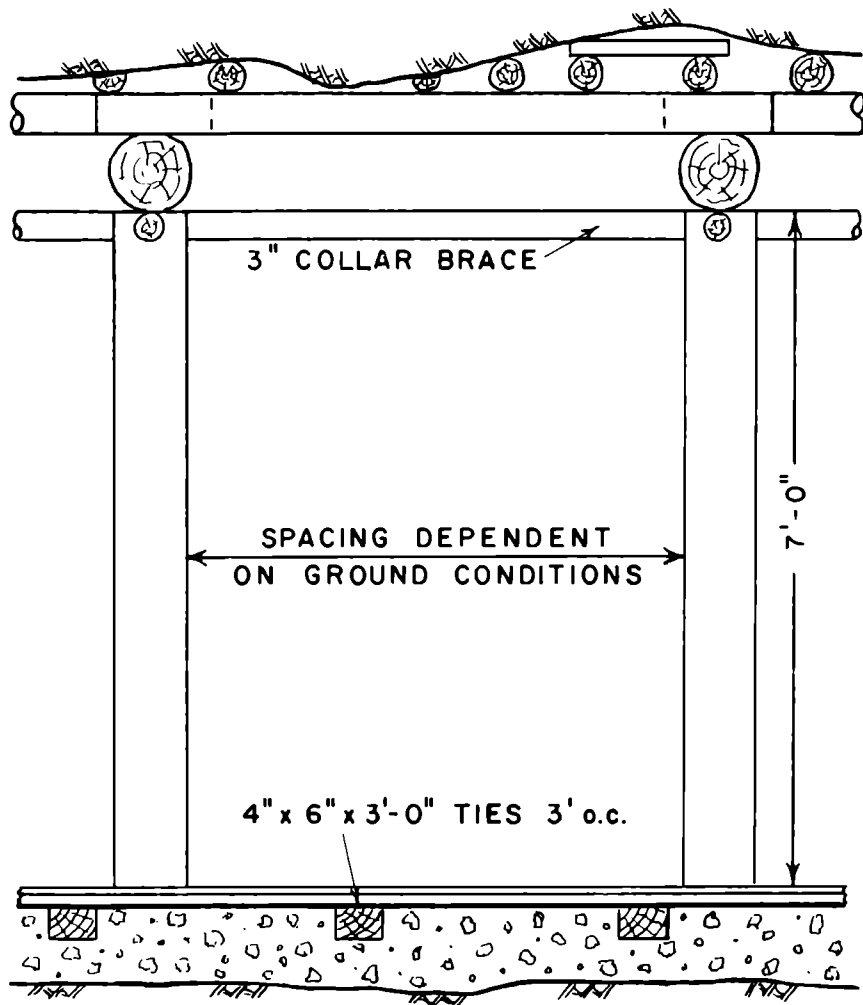
Producing mines are operated on a two shift basis, with a minimum of 3 hours between shifts to allow the powder smoke to clear.

Ventilation doors and seals are normally made of timber. However, plastic sheeting, on light timber frames, has proved useful for temporary seals. Some permanent seals, in untravelled sections of the mine, are also made with plastic sheeting. The plastic does not stand up to human curiosity or vandalism but otherwise serves as a superior brattice cloth.

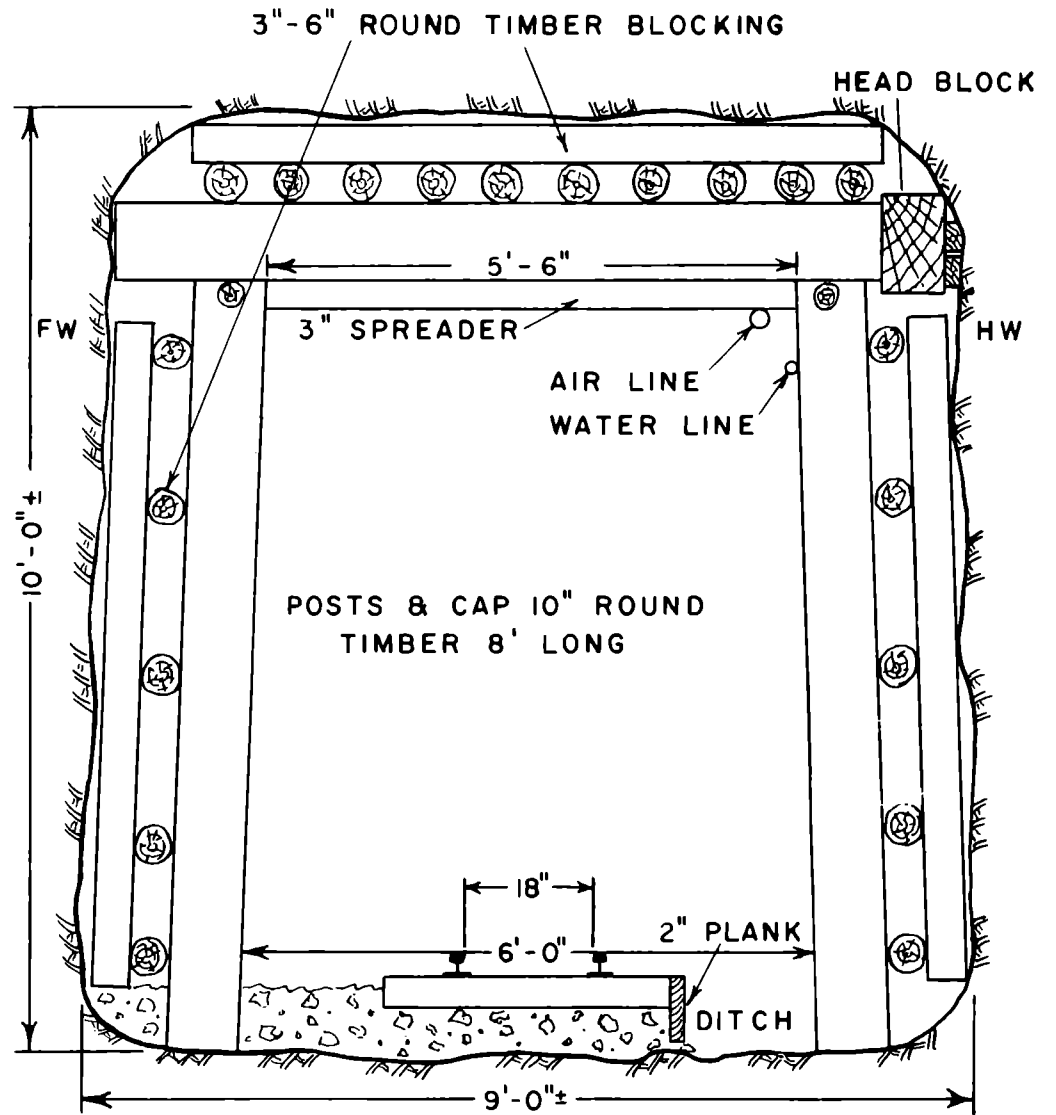
Development

All horizontal development headings are driven with airleg equipment and carbide tipped integral steel. Ground strength and hardness varies widely in the district and development practices vary to suit conditions.

Most crosscuts are driven in relatively competent rock in the footwall or hanging wall of the vein zones and follow usual development practice. A two man crew of miners muck out, scale and timber, drill off and blast a six foot round in a shift. The headings are driven 9' by 10' leaving 6' by 7' inside timber. (Figure 10). Track laying, digging ditch, and installing water lines are considered part of the normal work of the drift crew. Except in the strongest ground all headings are timbered with set intervals dictated by the strength of the ground. In a few exceptional cases, crosscuts have been driven without timber. In other cases, where there is no need of side lagging, bullhorn sets and back lagging are used. Back blocking is used in relatively competent ground to avoid constant scaling. Small "loose" develops regularly due to an intricate fracture



SECTION ON C



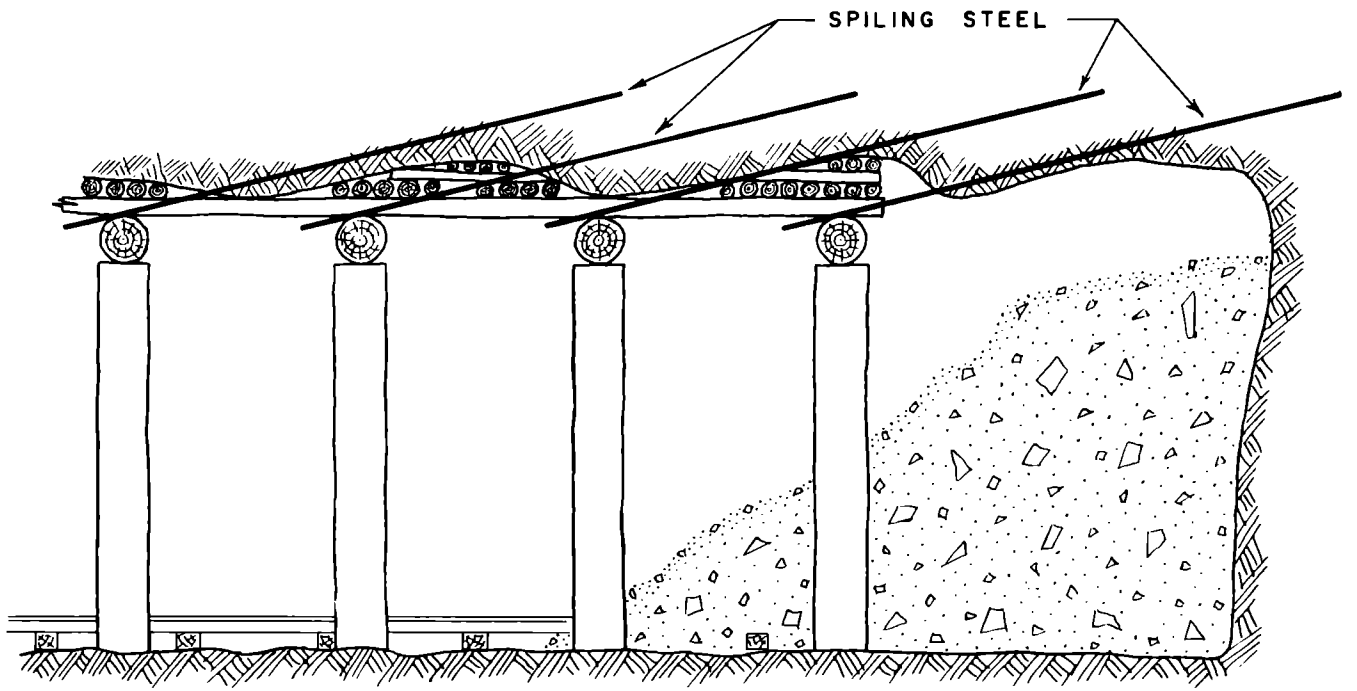
CROSS SECTION

STANDARD LAYOUT FOR DRIFTS AND CROSSCUTS

pattern, abundant schist bands, wide spread oxidation and the gradual retreat of permafrost.

Drifts are driven in the veins, or to give them a more descriptive designation, in the vein faults. The ground varies from moderately hard to gouge, with drill penetration rates seldom below 20 inches a minute. Drilling is often the smallest part of the cycle but loading time is equal to or greater than drilling time. Holes fill with angular fragments or with gouge and it is common for the miner to lose some holes in each round. Drifts are driven the same size as crosscuts and the set details are similar. However, in drifts both ribs are always lagged and and at least one set of timber is installed per round.

In veins where there is much gouge or graphite schist, particularly where water is present, spiling steel is drilled on the back and sometimes on the ribs (Figure 11). On completion of mucking, the crew stands a set of timber as close as possible to the face. They then drill over the cap with 1 inch quarter octagon steel and detachable steel bits using a steel about twice the length of the round to be drilled. The spacing varies from one foot to two inches between steels. When the steel is run out, it is withdrawn if possible and a piece of 1 inch round mild steel rod is inserted in the hole. If the drill steel cannot easily be removed it is left in place to be recovered after the drift advance. When necessary, spiling steel is driven along the rib in the same way. When the spiling steel has all been driven, the ends of the steel above the cap are wedged tight, and then the crew drills off the round and blasts. The round usually breaks up to the spiling steel which holds the back until timber and blocking can be installed. In particularly soft ground, the back holes are not blasted with the round.



SPILING STEEL IN PLACE AFTER BLAST

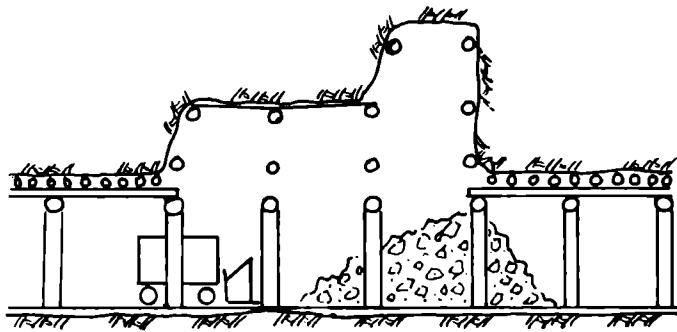
In all drifts and crosscuts where timber is required and where spiling steel is not used, crews are required to advance lagging booms and block the back before they are allowed to work ahead of the last set of timber.

Raising

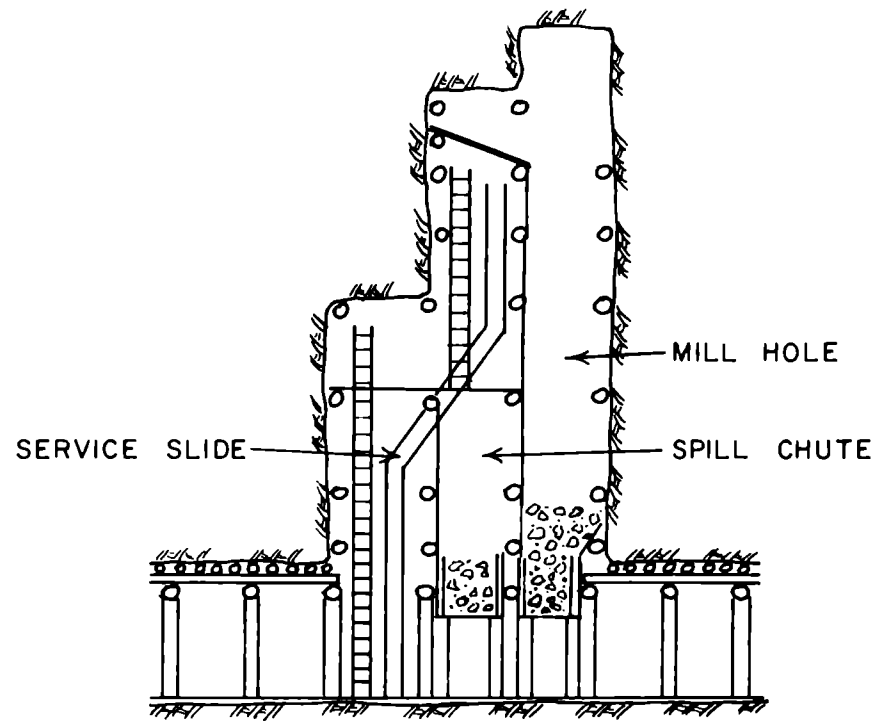
There are four main methods of raising used. The method chosen for any particular raise depends on the width of the vein, the character of the ground, and the purpose of the raise. Raises are normally placed at 100 foot intervals where required and are driven directly up the dip of the vein.

In wide veins, square set raises are driven. These are started in the same way as a normal square set stope, and are advanced in two slots which extend from the hangingwall to the footwall of the vein. One set in each lift must be drilled with a stoper but all other sets are taken with air leg equipment. As the raise advances a plank lined mill hole is carried up in one slot and the other slot is used for manway and supply slide.

In narrow veins either crib or stull raises are used. Both types of raise are started in the same fashion. Backs are taken down for approximately 18 feet and caps and blocking installed. (Figure 12). In the first 3 lifts the muck is blasted onto the track and picked up by a mucking machine. Three lifts are taken to a point about 28 feet above track. In two adjacent slots standard hand gate chutes are installed. Each slot is separated from its neighbor by a plank lining. The end slot without a chute becomes the manway with supply slide. The second slot is a spill chute and the last



INITIAL STAGE



START OF RAISE COMPLETED

START OF ROUND TIMBER RAISE

slot is the mill hole. In the third lift the manway and supply slide are moved into the center slot and the first slot is dropped in the fourth lift. This leaves a pocket under the manway over the spill chute, to collect muck which leaks through the mill hole lining or through the raise bulkhead. It has been found that the cost of installing the second chute is more than saved in eliminating drift cleanup. Above the manway crawl-through the raises are continued as a crib or as a stull raise.

The stull raise is advanced in two slots with a center line of caps and cap lines along each rib. Posts are installed between the caps on both footwall and hangingwall. These serve to help support the caps and also serve to hold the hangingwall blocking. In particularly bad ground the footwall is also blocked. The mill hole is separated from the manway with plank lining which is spiked to the center line of caps. The manway is kept within 5 sets of the face with a standard bulkhead and a crawlthrough into the mill hole. Because of the H.W. blocking it is impossible to make a completely tight bulkhead so there is always some leakage of broken muck into the manway and hence into the spill chute. The mill hole side of the face is advanced using a standard burn cut round. The rib cap and center cap are installed and the face, back, and rib are blocked. Then the other side of the face is drilled off and blasted.

The crib raise is used as an alternative method in particularly soft and friable ground. The start of the raise is the same as for the stull raise. Above the crawlthrough the manway is continued as a single crib with outside dimensions of 5 foot by 6 foot. The raise is drilled 6 foot and the mill hole is left untimbered. Crib material is unpeeled

local spruce cut to length and capped on the ends. Cribbing is advanced to the face and several pieces of crib are used to block the face during drilling. All holes are drilled from inside the crib and a modified draw cut round is used to throw the muck towards the millhole. When the round is loaded a complete bulkhead of cribbing is built across the manway before blasting. There are two disadvantages to the crib raise. In veins wider than 6 feet the crib raise does not expose the whole vein and there is a chance of missing vein splits. If the crews are not checked constantly the H.W. of the raise does not get blocked tight to the crib and in subsequent stoping, loose back conditions and overbreak are sometimes encountered.

A fourth type of raise is used where ore or waste passes are driven in competent rock. This is a standard 6 foot by 10 foot raise divided into two compartments by a single line of stulls and plank lagging. No blocking is required and standard raise methods are used.

Stoping

In general, heavy friable vein material and competent wall rocks necessitate close timbering and back filling in the stopes. The Elsa Mine has some exceptions where hard quartzite forms the walls in some of the stopes.

The present company and previous operators have used a variety of stoping methods; Timbered filled rill, timbered horizontal cut and fill, and shrinkage, to name a few. These may be considered to be of historical interest only, although one small shrinkage stope is currently in operation. Current practice over the past several years has concen-

trated upon two principal methods, with the width of the vein dictating the type of stoping to be used: Square setting in veins wider than ten feet; long walling in narrower veins. Occasionally both methods are used in one stope where an unexpected change in width is encountered. Also, a minor amount of timbered flat-backing is done in small lenses of ore.

In all stopes, drilling is done with air-leg equipment using either quarter-octagon steel with detachable steel bits, or carbide-tipped chisel-bit steel, depending on the type of ground. Holes are loaded with either 55 % Stopsite or 40% Forcite and detonated by No. 6 caps on tape fuse. Fuses may be either 10 feet long which are trimmed after loading, or 8 feet long for use with igniter cord. Most of the stopes work on a 2-shift basis with the usual crew being a miner and a helper; crews are expected to be proficient in all phases of stope work.

Long Wall Stopping

Long Wall is a local term for a method which could be called "Vertical cut and fill" or "down dip slotting". The word "long wall" is not inappropriate since the method resembles the long wall practice of coal mines except that the veins are steep (as opposed to more or less flat seams of coal) and mined-out sections are filled (rather than being allowed to cave).

The method consists of mining two parallel six-foot long slots down the dip for the full width of the vein, with a minimum working width of five feet. The slots are timbered with a stull cap and head-board placed slightly steeper than normal to the hanging wall, and standard cap spacing is 6 feet by 6 feet; posts are placed so that the hanging wall (and the

foot wall, if necessary) as well as the open rib, may be lagged and blocked. All caps and posts are collar-braced. Minimum sizes of round timber used are 8-inch stulls and 4-inch lagging.

A long wall stope can be started from either a stull or a crib raise. In the first slot alongside a crib raise it is necessary to enlarge the mill hole of the raise so that timber can be placed at the standard spacing. The drilling for this first slot is done by down holes and the cribbing of the raise must be braced prior to blasting. In subsequent slots (and in the first slot alongside a stull raise) drilling is done into the ribs with the holes sloping upward at approximately 15 degrees so that the broken rock is nearly all thrown clear of the bench and hand mucking is minimized.

Normally, while two slots are carried down in one operation, the first slot is kept no more than three benches below the second so that rib lagging is not always necessary between the two working slots. Crews may drill off extra holes while set up, but no more ground is blasted than is required for one set of timber.

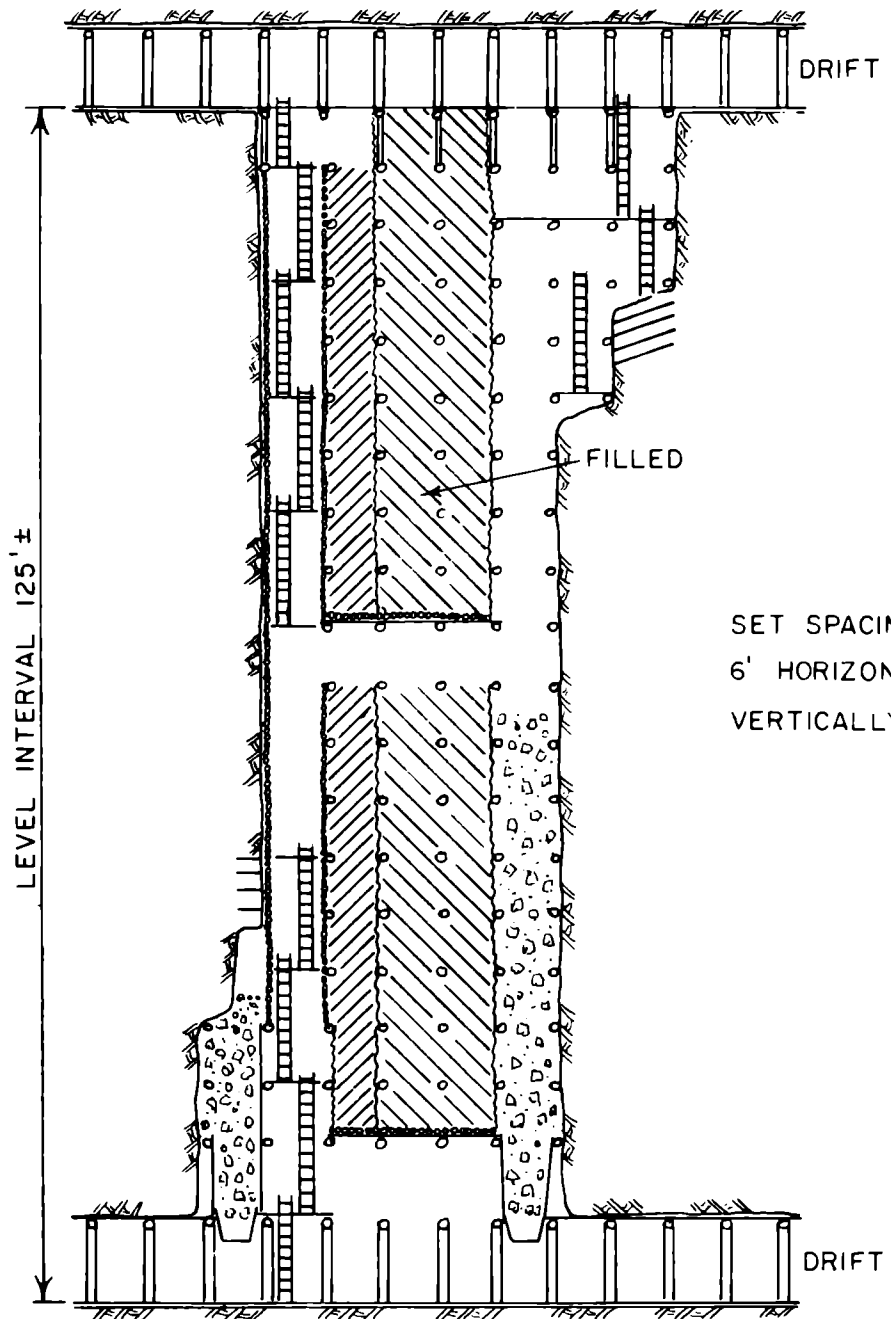
Formerly, long wall stopes were mined overhand, but safety considerations necessitated changing to underhand mining so that the crew's exposure to open ground would be minimized. A further advantage gained from underhanding is improved efficiency as the stope can be cleaned down of fines as mining progresses and fill fencing can be installed in the crew's spare time, thus reducing the non-productive time at the completion of a double-slot.

Safety precautions taken in long wall stopes include: spragging of open hanging wall before the set is timbered; installing safety bulkheads

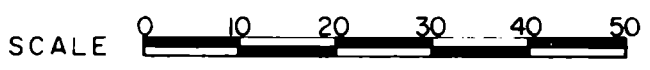
in the open slots no more than 25 feet above the working bench; and, in the steeper stopes, the mill holes are left as full of muck as practicable. The latter cannot be done in stopes flatter than 65 degrees because the fine muck tends to build up on the foot wall and eventually cause a hang up in the mill hole.

The bottom two or three benches of the slots are blasted directly on the drift track except where it is necessary to keep the drift clear, in which case china chutes are used. When a double slot is completed and the final clean-up is done, the chute is removed and fill bulkheads are installed in two slots, leaving the last-mined slot open to serve as a mill hole for the next double slot. The bulkheads consist of two or three stulls laid across the first cap above the drift set and covered with round lagging or such scrap timber as may be available. (The opening left above the drift sets is desirable since it facilitates any later timber repair work which may be required in the drift). Then the remaining portion of fill fence is installed. Fill fencing is two-inch mesh chain link-fencing supported on round lagging which is spiked on the stope caps. The chain link fencing is three feet wide and is purchased in 50-foot rolls. This is readily installed and provides a strong fence. A surprisingly small amount of fines comes through the mesh as fill is dumped. When the slot is half-filled a second bulkhead is installed and the opening below provides a secondary entrance to the stope.

In some crib raises it is necessary to start a long wall stope on the side opposite the raise mill hole. In such cases the crew drills and blasts three-foot or four-foot deep holes on a two-foot spacing, and in this manner a mill hole can be slashed out in a few shifts. (Figure 13



VERTICAL LONGITUDINAL SECTION
OF A LONGWALL STOPE



illustrates various phases of long-walling).

Spiling steel is occasionally used in long wall stopes at times when the hanging wall is overbreaking. In such instances eight-foot or ten-foot holes are drilled over the hanging wall post, and one-inch round spiling is inserted in the holes.

Square Set Stoping

This method was adopted in 1953 to mine the wider veins which were then being developed. Sets are cap butting and measure 5'4" by 5'4" by 6'11". Timber used is 8" x 8", which is the largest size of timber available locally in the quantities required, and is rather light for some of the ground. However, the sets can be readily reinforced with pony sets.

Stopes are started from square set raises and are mined overhand, with a liberal use of spiling steel, in various sizes of panels; a panel being two or more vertical slots mined the full width of the vein from level to level.

Stope Preparation - In starting a new stope panel no attempt is made to tie the new square sets to the old sets of the raise or of the previous stope panel, because, even in raises, ground pressures can force sets out of alignment in a relatively short time. Hence a off-set of two feet or so is made between the old and the new columns of sets. Ten-foot long spiling is drilled over the caps of the first lift of the previous slot, the drift back is taken down to a height of 13 feet above track, the back boomed and blocked, and the old drift set is removed. New sets are placed at 5' 4" intervals and are reinforced with rail beam; both the rail beams and the caps span the full width of the drift. Across the stull caps are placed 8" x 8" squeeze blocks so positioned as to be under the columns of posts of the

square sets. Then 8" x 8" sills of the jacket set are laid parallel to and directly over the drift sets and are braced with 6" x 8" or 8" x 8". The back is then slashed to a size sufficient for one set, which is immediately installed. Thus spiling, slashing and timbering continue until the first lift of the new panel is completed. All spiling steel is recovered either when the set is being blocked, or when the next lift is being mined.

In wide vein zones which have been developed with HW and FW drifts, the first lift of a stope is taken across at the elevation of the drift backs leaving the crown pillar to be mined with the stope below.

Actual mining of the stope panel follows immediately upon completion of the preparation work.

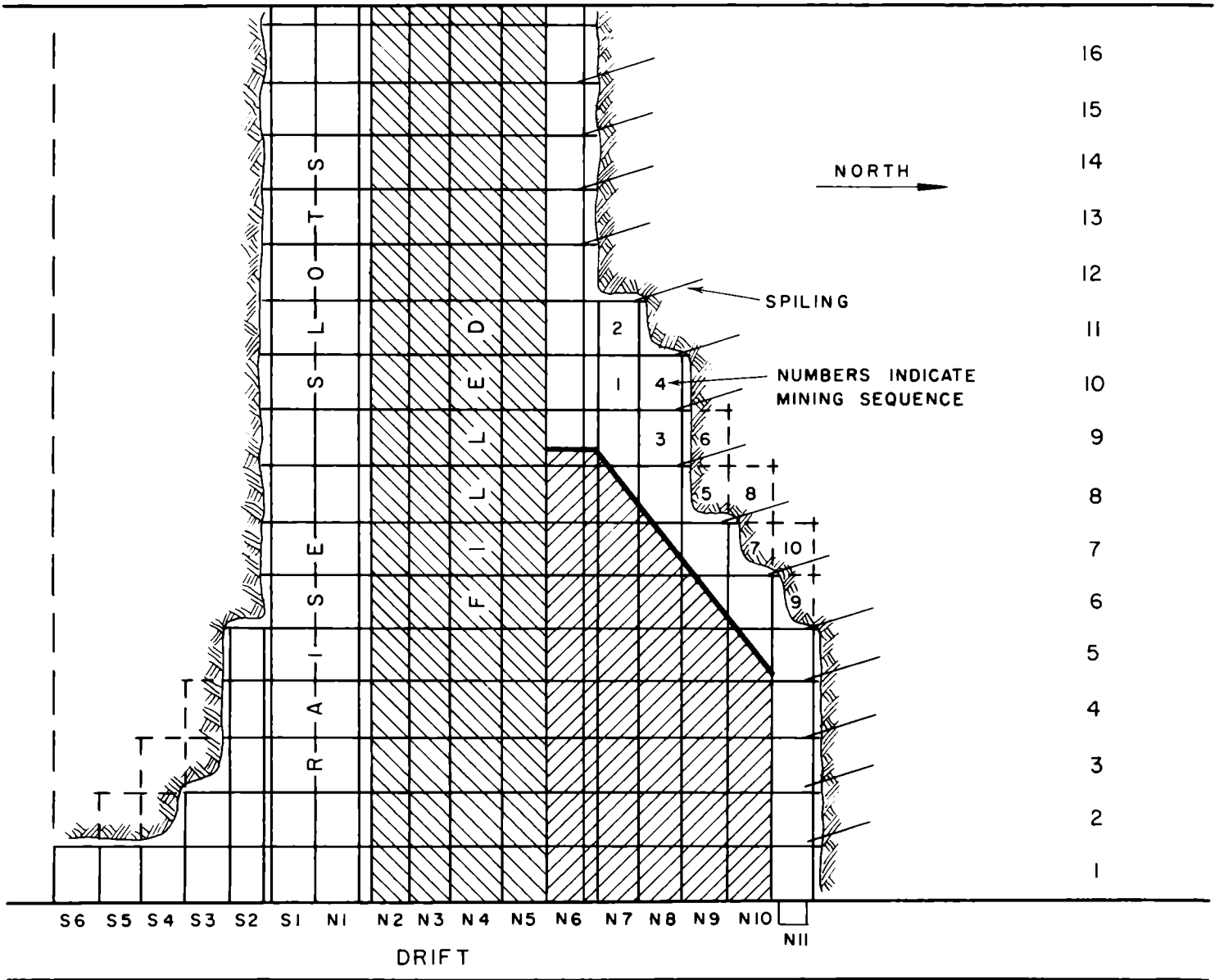
Types of Square Set Stopes

Flat Back - This type of mining was tried initially in the 3 vein but collapse of one stope due to heavy weight of the back caused this form of square set stoping to be abandoned in that part of the mine. The method does find some application in small wide lenses of ore in scattered locations, and it might be practicable in the 18-19 vein zone where vein material appears to be fairly competent.

Rill Stope - Rilling was adopted to mine successfully the 3 vein at the Hester Mine. In the ore blocks stoped by this method, about 95% recovery has been obtained. Five slots comprise the usual rill stope panel.

Figure 14 shows, on the left, a rill stope being started. The first slot (S 2) is mined up to lift 5, the next to lift 4, and so on. During this stage, the muck is moved by alushers and/or slides to temporary

DRIFT



chutes. When the rill is established, fill bulkheads are laid in the first four slots. Over the drifts, bulkhead stulls are laid at right angles to the jacket sets, and the spaces between the drift cap and the jacket set is filled in with 8" x 8" blocks. Over the crown pillars, 12-foot to 16-foot long bulkhead stulls are laid parallel to the stope caps, with the joints staggered to facilitate the later mining of the pillar below. Bulkhead stulls are covered with lagging (or scrap timber when available).

Fill fence (chain link fencing on lagging) is installed at both ends of the section to be filled leaving the end slot open; the latter serves as a mill-hole and second manway for the panel. At the lower end of the section being filled, fencing is installed to a line two feet above the caps and fill is dumped through a fill pass which has been built in slot S1. To obviate later settling of the fill, pneumatic tampers are employed during the first filling; tamping has been found to compress the fill as much as one foot in a single lift. When the fill has covered the caps to a depth of one foot or so, sloping floors are laid. These consist of 3" plank nailed at the upper ends only to lagging which is placed across the posts and rests on the fill. The lower ends of the planks overlap the flooring of the set below, so that the entire floor "floats" on the fill, and the broken ore falling on this floor has a further tamping effect on the back fill. With the first filling completed, a permanent chute of standard design is built in the end slot; slides (3" or 4" plank on edge) and grizzly rails (45# or 56# rail) are installed, and mining is resumed.

In this type of stope, it is the general practice to mine a double-rill, as indicated on the right of Figure 14: two lifts are mined in the highest slot (N 7), two lifts in the next slot (N 8), and so on, with the work proceeding down the rill so that a minimum of open timber is exposed. As a lift is completed in each slot, spiling steel is drilled into the rib where conditions warrant. When mining is finished on a double-rill, the fill floor is lifted, fines are cleaned from the top of the back fill, fill fence is installed, fill is dumped, and flooring is relaid. Two lifts in the last slot of the previous panel are filled in conjunction with the sets in the current panel, leaving one set open at the hanging wall for a manway. The mill hole is raised using 3" or 4" plank for lining; so far as is possible, the mill hole is kept only one set wide. In particularly heavy ground, it is necessary to fill the foot wall part of the mill hole slot, leaving two sets open immediately below the hanging wall for manway and mill hole.

When the highest slot of the rill is within three sets of the level, it is mined through and the slot behind is filled. After the panel is completed and filled, the mill hole is cleaned down and repaired for use as a fill pass for the next panel.

Vertical Double Slot Steps - Here, a panel consists of two slots. This form of mining was developed to provide better efficiencies in those parts of the mines where both the vein material and the walls are more competent, and where the vein material is less heavy (i.e. less massive sulphides) than in the 3 vein of the Hector Mine.

Most stopes in which this method is used average four to five sets in width. However, a variation of the method is required in the

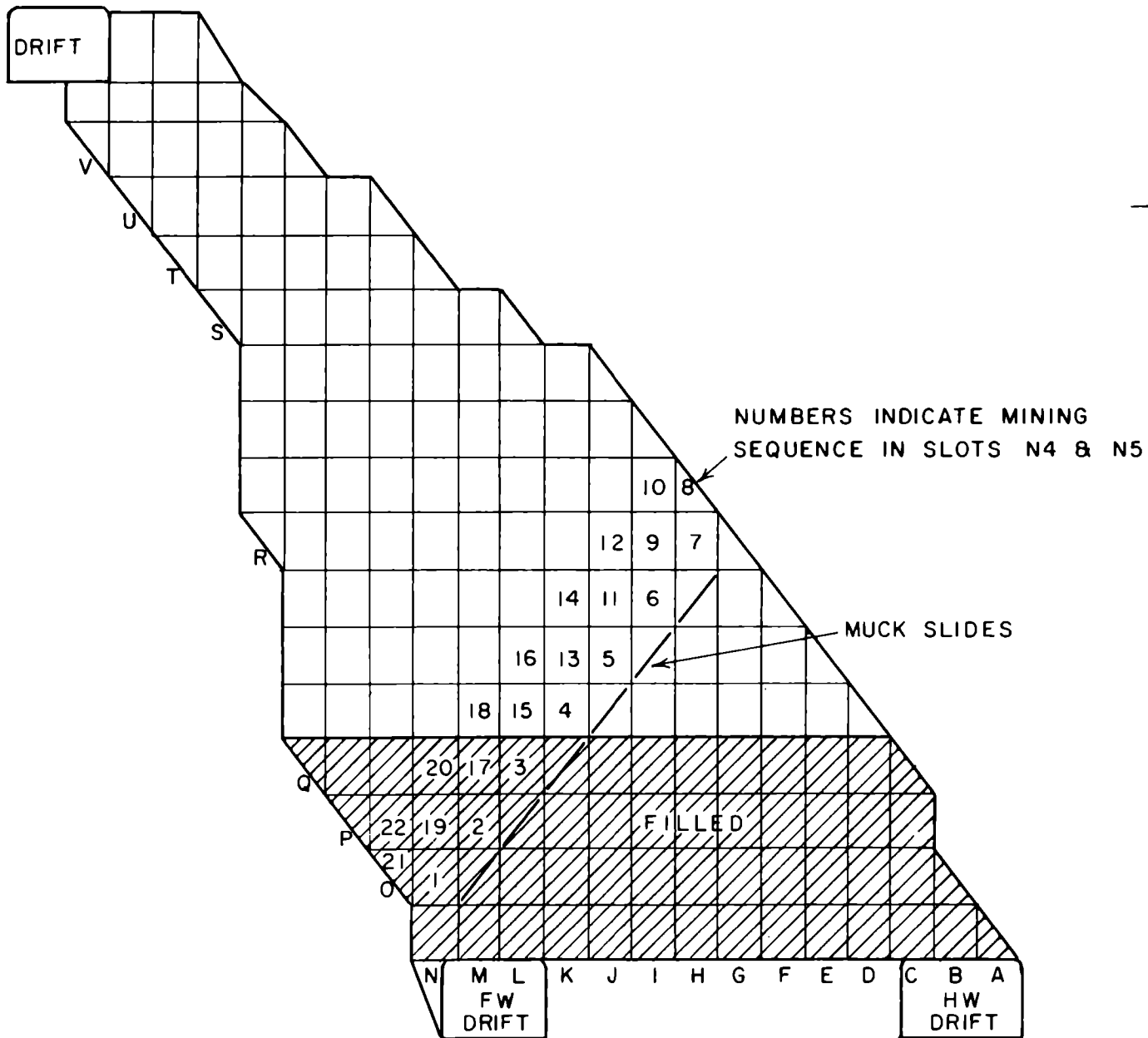
15 vein at Calumet mine where widths of up to fourteen sets are encountered.

In a narrow vein stope of this type, the permanent chute is constructed after two or three lifts have been mined; mill hole lining plank, grizzlies and muck slides are then installed, and mining proceeds. After every third or fourth lift, the grizzlies, slides and mill hole are raised, and, since the mill hole is kept to one set in size, very little timber is broken during the mining operation.

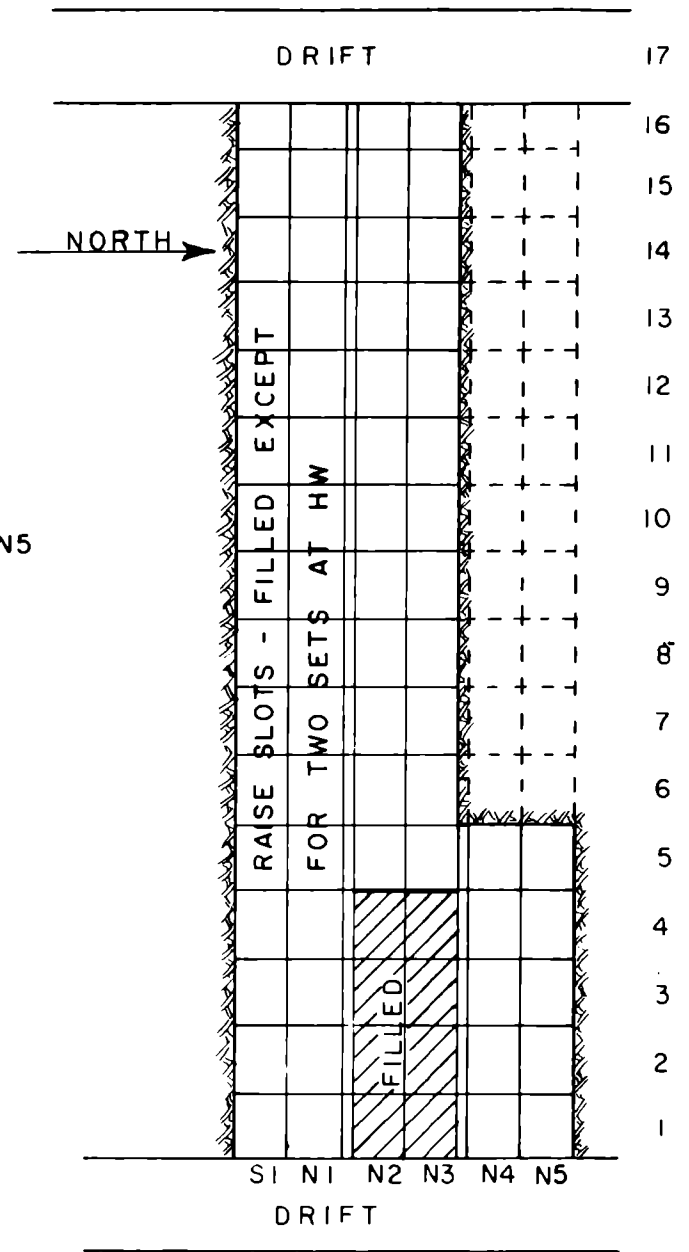
If the panel is the first one mined alongside a raise, a second panel is completed before the first is filled. Prior to filling, the standard bulkheads and fill fencing are installed. Back fill is tamped wherever possible to minimize settling of the square sets. Nevertheless a varying amount of subsidence occurs in the fill; in this case, the two-foot offset between the panels has been found to form a "break line" for the settling so that the sets alongside the filled sets are little affected.

In double slot stopes in the wider parts of the 15 vein, Calumet mine, mining is carried out on a rill from hanging wall to foot wall. When the first lift of the panel is being mined, and while the rill is being established, all muck is slushed to temporary chutes. When the toe of the rill has reached the foot wall a permanent chute is built over the foot wall drift. At this time the initial fill is placed in the previous panel, where a timber crew has already installed bulkheads and fill fencing. Fill is dumped into a lined waste pass at the foot wall of the stope, slushed into place and tamped.

The sequence of mining is shown in the cross-section in Figure 15.

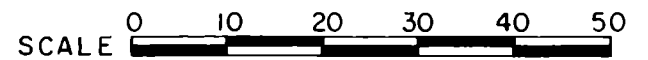


CROSS SECTION SLOT N3



LONGITUDINAL SECTION

TYPICAL DOUBLE SLOT STOPE IN WIDE VEIN ZONE



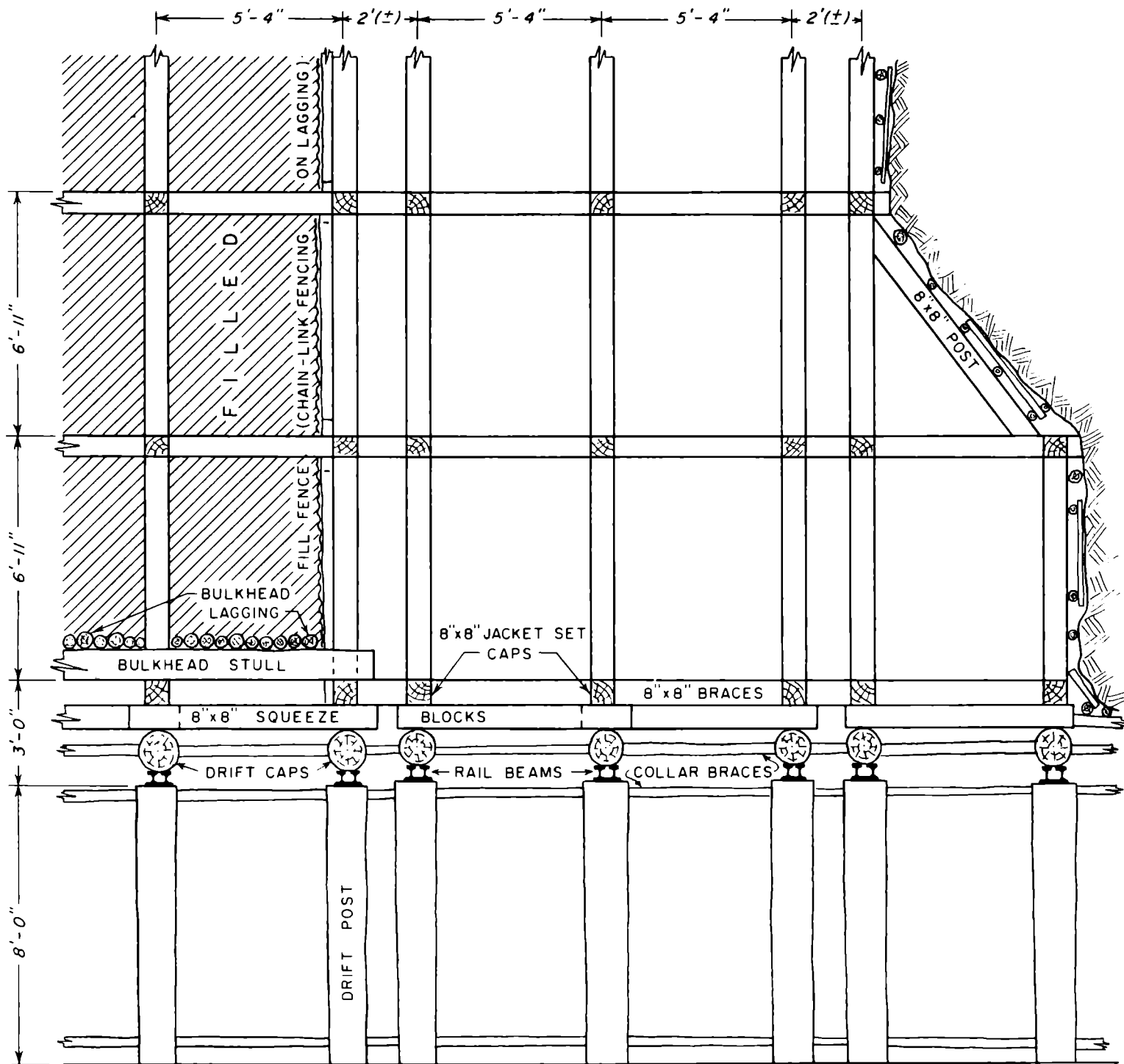
As each of sets 1 through 6 is mined (working up the rill), slide planks are installed below the succeeding set. Then a double lift is mined down the rill using the same slides. Filling of the previous panel advances concurrently with the mining. When a panel is completed the mill hole (one set in size, at the foot wall of the stope) is cleaned down and repaired for use as a fill pass for the succeeding panel.

General Remarks on Square Set Stopping

Seldom is more ground blasted than is required for one set of timber. Backs are blocked with 8" x 8" or 6" x 8" and lagging; ribs are kept tightly lagged. In cases where the back overbreaks it is generally on a slope, and the angle posts and lagging are installed (see Figure 16); this provides more positive holding of the ground and obviates the use of excessive amounts of timber for blocking.

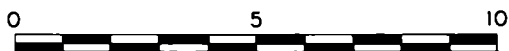
Safety precautions taken in square set stopes include spragging and/or booming of the back immediately after scaling of every set, laying complete floors in at least every third lift - with all other open sets being fenced off with guard rails.

Occasionally, in stopes in extremely friable ground, the rib will slough up in spite of spiling steel. In such instances, long 8" x 8" booms are run out at the back of the sloughed section and blocking is placed over the booms. Then sufficient muck is removed to permit placing booms in the next lower lift. The upper booms are posted from the lower ones, and walls are spragged. In this manner work proceeds down through the sloughed zone to the top of the square sets below. New sets can then be installed to replace the sprags and booms.



LONGITUDINAL SECTION

DETAILS OF SQUARE SET - JACKET SETS & FILL BULKHEADS



Reinforcing of square sets is often necessary in heavy ground. This is done with pony sets (8" x 8" caps and 8" round posts) or by cribbing up the sets.

Ore control in the stopes is obtained from daily chip samples, geological mapping, and visual inspection. Also, six-foot test holes are drilled regularly in both HW and Fw, with all ore values in the sludges being investigated. When each stope panel is completed an ore outline of the rib is drawn up to serve as a guide for mining the next panel.

In the 15 vein area, Calumet Mine, the over-all mining plan follows a level-to-level rill. Figure 17 shows this rill in the process of being established.

Standard chutes in all stopes are fitted with steel hand-operated gates.

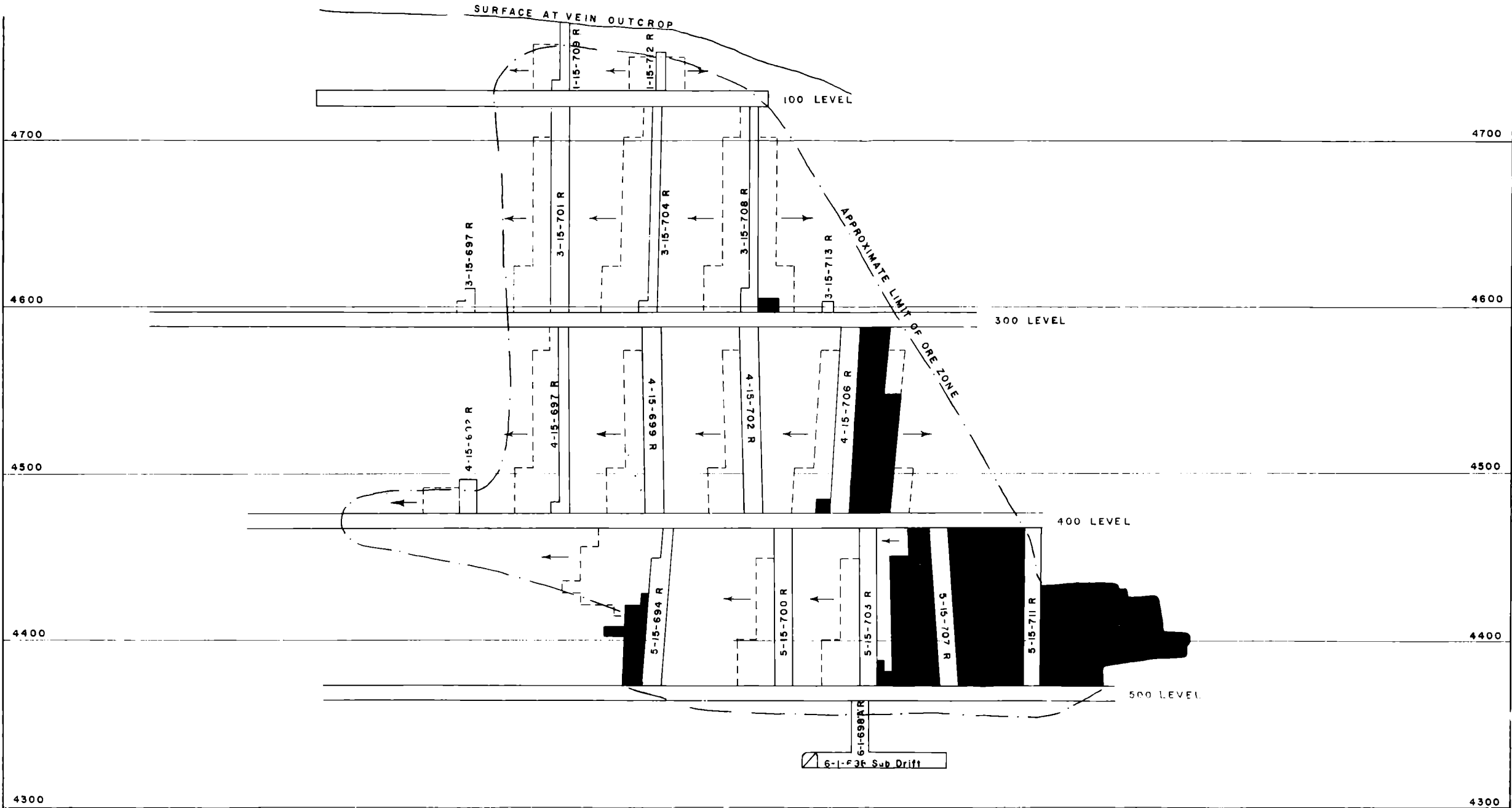
PRODUCTION STATISTICS

Oct. 1, 1959 - Aug. 31, 1960

	<u>Hector & Calumet</u>	<u>Elsa</u>	<u>Total</u>
Daily mine production* ore:	390.4 Tons	91.5 Tons	481.9 Tons
waste:	169.9 Tons	52.1 Tons	222.0 Tons
Average No. of employees total mining:**	215	51	266
Average No. of employees stoping:	86	18	104
Productivity total mine:**	2.16 tons/man shift	2.17 tons/man shift	2.16 tons/man shift
Productivity stoping:	4.34 tons/m. s.	5.71 tons/m. s.	4.57 tons/m. s.

* Calendar days
** Does not include mechanical, electrical, surface and staff personnel

(Continued.....)



LONGITUDINAL SECTION OF NO. 15 VEIN



Stoping Productivities (Tons/man shift)

	<u>Hector & Calumet</u>	<u>Elsa</u>	<u>Total</u>
Square Set	4.72	5.88	4.93
Round Timber	3.41	4.96	3.61
Shrinkage	-	<u>6.25</u>	<u>6.25</u>
TOTAL	4.34	5.71	4.57

<u>Breakdown of Stoping by Type</u>	<u>Hector & Calumet</u>	<u>Elsa</u>	<u>Total</u>
Square Set	77.3%	78.4%	77.5%
Round Timber	22.7%	18.0%	21.7%
Shrinkage	-	<u>3.6%</u>	<u>0.8%</u>
TOTAL	100.0%	100.0%	100.0%

TABLE IIIMINE COSTS (\$ per Ton)

	<u>Hector & Calumet</u>	<u>Elsa</u>
Development	\$ 3.99	\$ 4.81
Stoping	6.91	6.00
Haulage	2.23	1.78
Hoisting	0.78	1.69
Pumping	0.15	0.55
Ventilation	0.00	0.02
Diamond Drilling	0.25	1.17
Timber Maintenance	0.91	0.91
Rehabilitation	-	0.57
Mine General and Indirect	<u>3.77</u>	<u>4.35</u>
TOTAL	\$18.99	\$21.85

Stoping Costs (\$ per ton)

	<u>Hector & Calumet</u>	<u>Elsa</u>	<u>Total</u>
Square Set	\$8.22	\$6.83	\$ 7.93
Round Timber	8.92	5.61	8.35
Shrinkage	-	<u>4.56</u>	<u>4.56</u>
TOTAL	\$8.39	\$6.52	\$ 8.00

TABLE IV

INTER-PLANT ORE DELIVERY

Ore is transported from the Hector and Calumet Mines to the mill at Elsa by a 2.7 mile aerial tramway. Production from the Galkeno Mine and the Keno Hill properties is trucked to the mill while ore from the Elsa Mine is dumped directly into the mill bin.

The tramway which was originally constructed to deliver ore from the Lucky Queen Mine to the old Treadwell Yukon mill at Wernecke camp, was dismantled in 1935 and re-erected and extended on its present site. It consists of a loading terminal at Calumet, a 15° angle station, a tension station and a discharge terminal at Elsa. There are 42 pyramid type timber towers supporting two track cables and an endless traction cable. The towers are located on permanently frozen ground and are supported on cribbed foundations. The tramway is powered by an electric motor at the top end which also regenerates power when acting as a brake on the loaded line.

A summary of pertinent tramway data is as follows:

Length	14,200 ft.
Difference in Elevation	1,500 ft. in favour of the load
Buckets	0.5 ton; rounded bottom; anvil grip on hangers; 2 wheel trolleys.
Average number of buckets on line	45
Approximate bucket spacing on line	600 ft.
Traction rope speed	406 ft./min.
Average tonnage trammed per hour	20
Track cable (load side)	1 1/8" dia. 37 wire, smooth core, right hand lay, plow steel.
Track cable (return side)	3/4" dia. 6 x 7 hemp centre wire rope.

Aerial Tramway Costs - Oct. 1/58 - Sept. 30/59

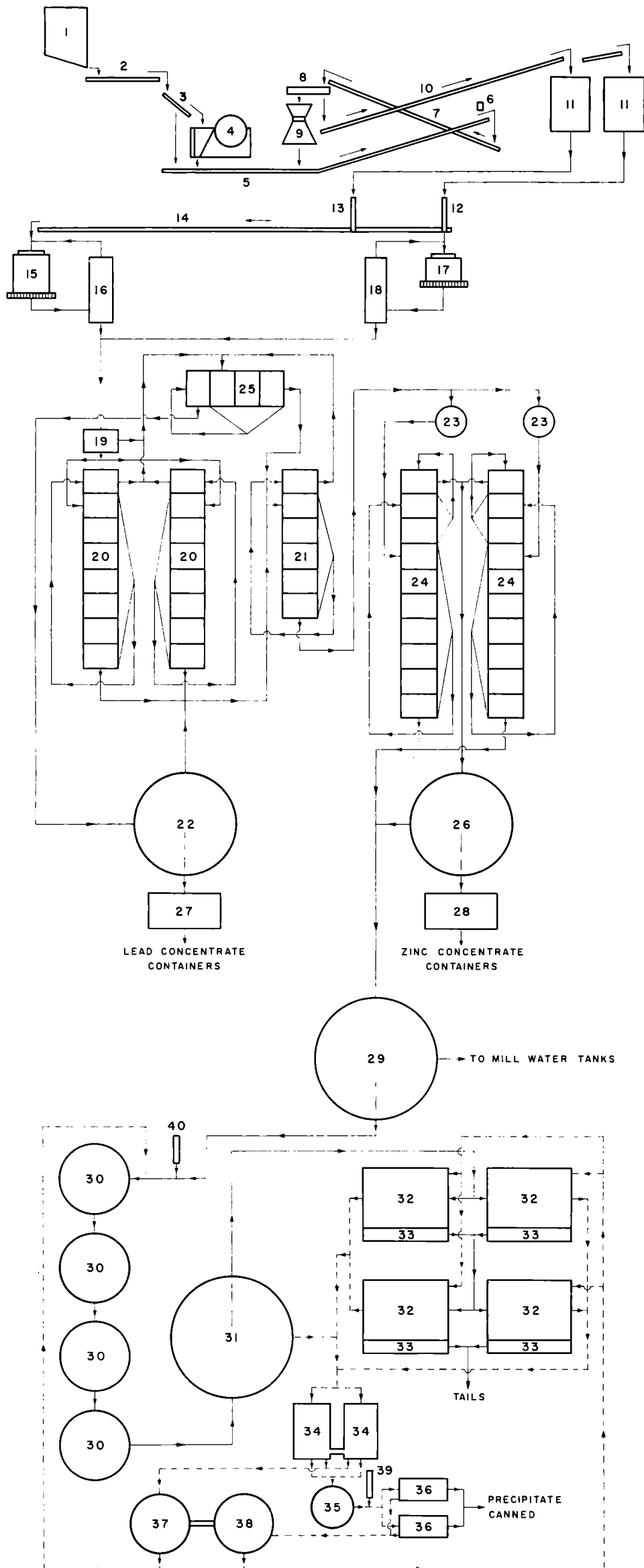
Tons trammed	144,615
Operating Labour	\$0.51/ton
Repair Labour	\$0.11/ton
Repair parts and Supplies	\$0.06/ton
Heat	<u>\$0.01/ton</u>
Subtotal	\$0.69/ton
Indirect expenses (fire protection, general shops, etc.)	<u>\$0.07/ton</u>
TOTAL	<u>\$0.76/ton</u>

MILLING

A paper on Milling Practice at United Keno Hill Mines was published in C.I.M.M. Transactions Volume LVIII pp. 140-146. This present paper will attempt to outline the alterations in the flowsheet since that time and the reasons for such changes.

The Elsa mill is a combination flotation and cyanide plant. At the present time, the flotation concentrator is treating 525 tons of ore per day and the cyanide plant 430 tons of flotation tailings per day. Figure 18 is a flowsheet of the mill.

The coarse ore bins are designed to receive ore from the aerial tramline, from the Elsa Mine in mine cars or from other properties by truck. The crushing circuit is of a typical closed circuit design with a jaw crusher, a gyratory crusher and a vibrating screen. The screen is operated normally with a 1" square opening bottom screen. Frequently, due to the high moisture content of the ore, it is necessary to change to a



LEGEND

- 1.....250 TON COARSE ORE BIN
- 2.....No.1 CONVEYOR, 30" x 52'
- 3.....5'-6" C.A.C. CANTILEVER GRIZZLY
- 4.....10" x 20" BLAKE TYPE JAW CRUSHER
- 5.....No.2 CONVEYOR, 24" x 80'-6"
- 6.....DINGS ELECTRO-MAGNET
- 7.....No.3 CONVEYOR, 24" x 63'
- 8.....3' x 8' A.C. LOW HEAD SCREEN
- 9.....1'-8" TRAYLOR TYPE TY CRUSHER
- 10.....No.4 CONVEYOR, 18" x 147'-6"
- 11.....2-300 TON FINE ORE BINS
- 12.....No.5 CONVEYOR, VARIABLE SPEED
- 13.....No.7 CONVEYOR, VARIABLE SPEED
- 14.....No.6 CONVEYOR, 18" x 34'
- 15.....7' x 9' OVERFLOW TYPE BALL MILL
- 16.....48" AKINS CLASSIFIER
- 17.....7' x 5' TRAYLOR BALL MILL
- 18.....48" AKINS CLASSIFIER
- 19.....No.500 DENVER UNIT CELL
- 20.....16-No.18 SP DENVER SUB A LEAD CELLS
- 21.....6-No.21 DEEP SUPERCHARGED LEAD CELLS
- 22.....18' x 12' DENVER SPIRAL THICKENER, LEAD
- 23.....2-6' x 6' DENVER CONDITIONERS
- 24.....20-No.18 SP DENVER SUB A ZINC CELLS
- 25.....4-No.18 SP DENVER SUB A LEAD RECLEANING
CELLS
- 26.....18' x 8' DENVER SPIRAL THICKENER, ZINC
- 27.....6' SIX-DISC AMERICAN FILTER, LEAD
- 28.....4' FOUR-DISC AMERICAN FILTER, ZINC
- 29.....40' x 14' DORR TWO-COMPARTMENT THICKENER
- 30.....4-20' x 22' DORR TYPE A AGITATORS
- 31.....35' x 12' DORR TWO-COMPARTMENT THICKENER
- 32.....4-11'-6" x 14' OLIVER FILTERS
- 33.....4 REPULPERS
- 34.....28-LEAF CLARIFIER
- 35.....CROWE TANK
- 36.....2-36", 22-FRAME PRECIPITATION PRESSES
- 37.....12' x 12' MILL SOLUTION TANK
- 38.....12' x 12' BARREN SOLUTION TANK
- 39.....ZINC FEEDER
- 40.....LIME & CYANIDE FEEDERS

—— PULP
 - - - SOLUTION

CONCENTRATOR & CYANIDE PLANT FLOW SHEET			
UNITED KENO HILL MINES LTD.			
	BY	DATE	NOT TO SCALE
DRAWN	JK	SEPT, 1960	
CHECKED			

3/4 inch slotted wire screen which has a better screening efficiency on wet muck. The crushing plant product is conveyed to one of two wood stave fine ore bins. Grinding is carried out in two parallel closed circuit units each with an overflow ball mill and spiral classifier. The combined classifier overflow has a grind of 58 percent minus 200 mesh. Classifier overflow is floated to recover a cleaned lead concentrate which is recleaned to produce shipping grade concentrate. The lead tailing is conditioned and a zinc concentrate floated. Concentrates are thickened, filtered and containerized for shipment.

The zinc flotation tailings are thickened to 60% solids and pumped to the first of a series of four agitators. The necessary lime and cyanide are added to the pump box. Overflow from No. 4 agitator is thickened and the thickener underflow is filtered, repulped and refiltered. The final filter cake is repulped and flows to waste. The thickener overflow, plus filtrate from the filters is clarified and the silver precipitated by zinc dust. The silver precipitate is recovered in plate and frame presses. The silver precipitate is sealed in cans and shipped to the smelter.

Crushing Plant

In the years prior to 1957 when the major part of the mill feed came from the upper levels of the Hector Mine, the capacity of the crushing plant was approximately 40 tons per hour. However, as the proportion of ore from the Calumet and Elsa mines with their greater quartzite content has increased to 68% of the mill feed, it has become

necessary to operate the crushing plant 24 hours a day. In view of the increasing hardness of the ore and the decrease in the primary fines in the ore, it has been decided to construct a new crushing plant.

The new crushing plant will have a designed bin capacity of 1000 tons and a crushing capacity of 75 tons per hour when crushing our present ore to minus 1/2 inch. The new flowsheet will be similar to our present plant with a jaw crusher and a secondary crusher in closed circuit with a vibrating screen. An efficient dust removal system will be incorporated in the new plant.

Grinding

The 7 x 9 ball mill was originally a grate type ball mill which has been converted to an overflow type. This conversion was accomplished by removing the grates, the lifters and the discharge end liners and installing newly designed discharge end liners. It was necessary to lengthen the one ring of shell liners adjacent to the discharge end of the mill and a new discharge trunion liner with a reverse spiral was installed. The conversion of this mill was carried out in an attempt to reduce undue shell and end wear, and also to increase the tonnage milled. The increased quartzite content of the ore and the decrease in primary fines had greatly increased the load on this ball mill.

There was considerable wear of the discharge end of the mill and of the shell adjacent to the discharge end. With the newly designed end liners, it is possible to get a much better liner fit and it has been

possible to stop most, if not all, of the pulp wash that used to occur in and around the lifter liners. While it is too soon to be positive, indications are that the shell wear has been greatly reduced if not stopped completely. Liner consumption has decreased markedly but not enough time has elapsed to determine exact figures. The converted mill has been equipped with a trommel screen on the discharge trunion. The plus 3/4 inch material thus screened out is practically 100 percent quartzite, contains no values and is discarded. By discarding this quartzite, the mill capacity has been greatly increased.

Flotation

The flotation process is being complicated by the increasing amounts of refractory minerals being received from the Calumet and Elsa Mines. Argentiferous plumbo-jarosite, from both these mines, does not respond readily to flotation nor do the non-sulphide lead minerals. Test work on some ore samples containing an intimate mixture of tennantite, tetrahedrite and pyrite has shown that these minerals respond to flotation very slowly. To increase the recovery of these minerals in the lead concentrate, the lead flotation time has been increased from sixteen to twenty-six minutes. During this additional ten minutes of flotation time, the pulp is subjected to a scavenging float with Aero Promoter 404 to obtain as much of the silver and lead as possible in the lead concentrate. This scavenging float has been successful in reducing the silver in the zinc concentrate by as much as ten ounces silver per ton of concentrate. This is being accomplished much more economically than by deleading the zinc concentrate.

Cyanide Plant

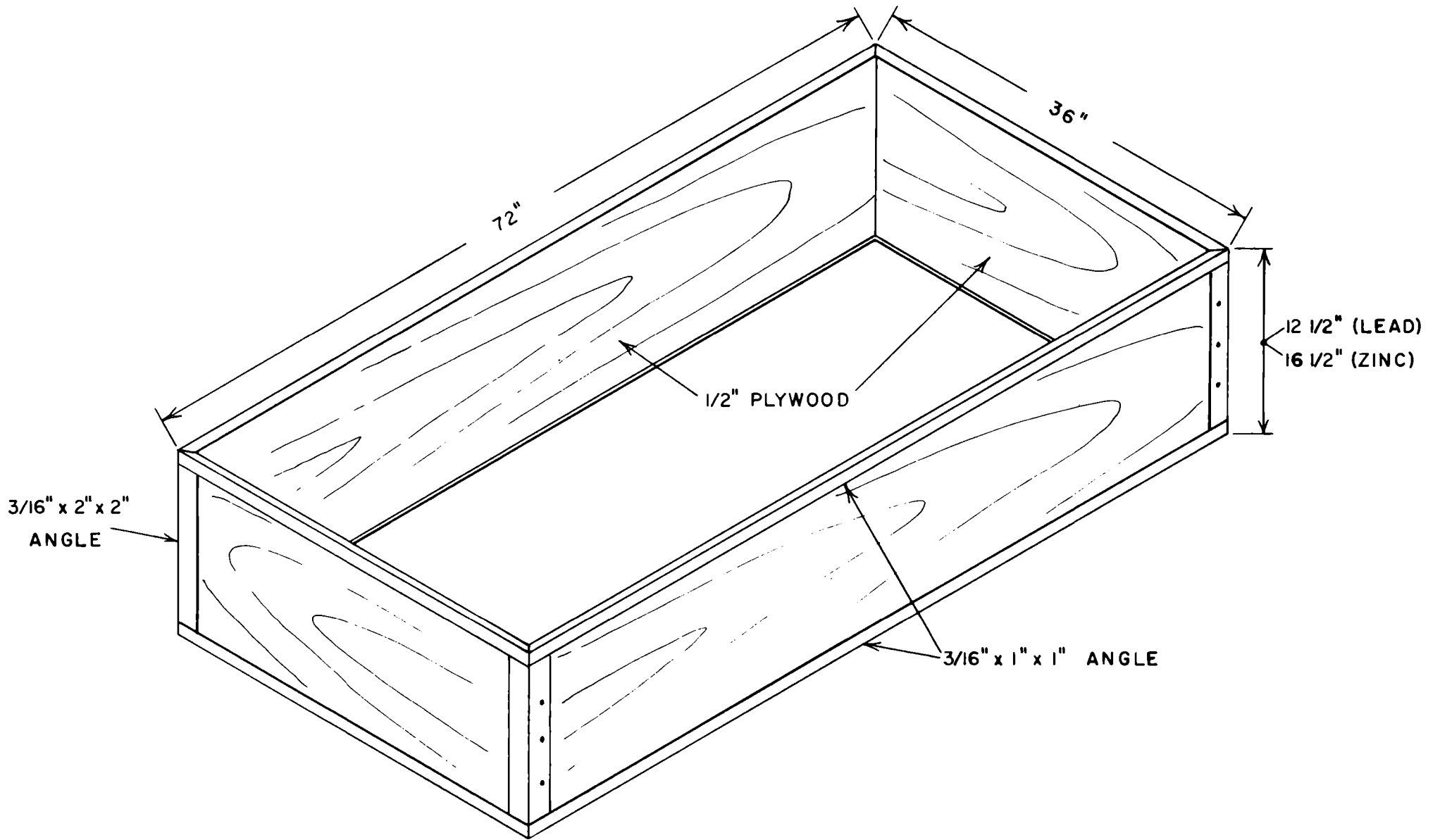
Sixty percent of the silver in the zinc flotation tailing is recovered in the cyanide plant. The relatively coarse grind of the cyanide plant feed has some bearing on the tailing loss but the major factor is probably a complex silver mineral which does not respond either to flotation or cyanidation. The silver precipitate being produced assays approximately 35% silver, 35 - 40% lead, up to 10% zinc and up to 14% copper.

Concentrate Handling

Until 1953, all concentrate was shovelled into burlap sacks. The sacks of lead concentrate weighed 132 pounds while the sacks of zinc concentrate weighed 120 pounds. These sacks were handled manually and individually while being loaded onto trucks in the mill, loaded onto railway cars at Whitehorse, unloaded at Skagway and loaded onboard ship. At Vancouver the sacks were emptied into railroad cars. In 1953, pallet boards were made available by The White Pass and Yukon Route and the sacked concentrate was shipped in two ton palletized packages. Thirty lead sacks or thirty-four zinc sacks were stacked on a pallet and strapped in place. With this two-ton palletized package, it was necessary to change to mechanical handling. In the mill, a three-ton overhead electric crane was installed for moving the palletized packages and for loading trucks. In Whitehorse, a Bull Moose was used to unload the trucks and load the railway cars and at Vancouver a fork lift was used to move the palletized concentrate. In the mill, the trucks were loaded while parked on a 50-ton truck scale and every load of concentrate was weighed.

Palletization and mechanical handling of the sacked concentrate resulted in some saving in costs but the cost of sacking and loading still amounted to 31.5 percent of the total milling cost. A significant saving in handling costs at Whitehorse, Skagway and Vancouver was also obtained. Once the procedure for palletizing the sacks of concentrate and handling the two-ton packages had been standardized, it was apparent that further savings could be obtained by replacing the sacks with a container. Therefore, an investigation was undertaken to find a container which would be more suitable and more economic than the sacks.

After considerable experimentation and testing of purchased containers, a suitable type was developed and put into use in 1957. This container, see Figure 19, consists of two angle iron frames made from 1"x 1" x 3/16" angle iron welded into a rectangle 36 inches wide and 72 inches long. Four pieces of one half inch thick plywood form the sides and ends of the container. To each end of each 72 inch long side is rivetted a 2" x 2" x 3/16" piece of angle iron. In assembling the container, one angle iron frame is placed on top of a pallet. The plywood sides are positioned in the angle iron and the other angle iron frame is placed on top. Once assembled, the plywood sides are prevented from moving outwards by the vertical members of the angle iron frames. The angle iron rivetted to the plywood sides forms a post which transmits the load from the top angle iron frame to the bottom angle iron and then to the pallet. At times, the loaded containers are stacked seven and eight high. A piece of plywood the same size as the ends is inserted in a vertical position in the center of the container



CONCENTRATE CONTAINER

to give rigidity to the container, especially while it is being strapped. The lead boxes have an overall height of twelve and a half inches and the zinc boxes have an overall height of sixteen and a half inches. The boxes were made these heights so that two tons of lead and of zinc could be packaged in their respective containers. The frames being the same size can be used for either lead or zinc containers and for either top or bottom frame.

In use, the container is assembled on the pallet and the top of the pallet is covered with burlap sacks, care being taken to bring the sacks up the side of the container. The assembled container, sacks and pallet is picked up by the three-ton overhead crane and placed on a small flat-car on an 18 inch gauge track. The container, pallet and car are moved under the head pulley of the conveyor onto which the concentrate has fallen from the filter. As the lead boxes are being filled, the concentrate is packed down with a shovel. The zinc concentrate is tamped with a pneumatic tamper three times while the box is filling. This tamping compacts the zinc concentrate to approximately three quarters of its untamped height. It is not possible to tamp the lead concentrate due to its fineness and moisture content.

The filled container is covered with 2 mil. polyethylene film. In summer, this prevents surface drying and dust loss and in heavy rains prevents concentrate washing out of the container. In the winter-time the polyethylene keeps the snow and ice off the concentrate. Four equally spaced steel straps are applied around the width and height of the container and under the top boards of the pallet and locked in position. This strapping helps hold the container on the pallet and is

necessary to hold the top frame in position. It also holds the polyethylene film in position. When the container is filled and completely packaged, the unit is lifted by the overhead travelling crane and is either loaded onto a truck for shipment to Whitehorse or is stored on the loading platform.

In Whitehorse, the palletized containers are removed from the trucks with a Bull Moose and loaded onto railway cars for shipment to Skagway. At Skagway, the palletized containers are loaded on a boat for shipment to Vancouver and unloaded at the company dock. At Vancouver the strapping around the container is cut and removed and the top frame and four sides are removed. The concentrate has solidified to the extent that there is no spillage. The pallet and concentrate is picked up by a fork lift truck with a rotary head which dumps the concentrate, center-board, bottom frame and sacks into a railway car for shipment to the smelter. The frame, center-board and sacks are removed from the railway car and packaged for return to Elsa. Sacks are packed in an empty concentrate container and the container components are packaged in lots of ten complete containers on a pallet, and strapped in place.

Conversion of concentrate handling from sacks to containers has reduced concentrate handling costs to approximately one third at the mill and considerably more at Whitehorse, Skagway and Vancouver. In addition, the present system has done away with all heavy physical work and reduced the manpower requirements.

Costs

Milling costs in dollars per ton of ore treated for the year 1959 are tabulated below. In table No. V, the cost of the various treatment steps are tabulated while in Table No. VI the costs are broken down into their various items.

Flotation

Crushing & Conveying	\$ 0.46
Grinding & Classification	0.80
Flotation, Thickening & Filtration	0.67
Containerizing & Loading	<u>0.71</u>
Total	\$ 2.64

Cyanide Plant

Primary Thickening	0.02
Agitation	0.82
Secondary Thickening & Filtration	0.24
Clarification & Precipitation	0.14
Shipping Precipitate	<u>0.01</u>
Total	\$ 1.23

General & Miscellaneous	<u>\$ 0.97</u>
TOTAL MILLING COST	<u>\$ 4.84</u>

TABLE V

Labour	\$ 1.16
Supplies	0.61
Reagents	0.89
Containers & Sacks	0.49
Power	0.72
General & Miscellaneous	<u>0.97</u>
TOTAL MILLING COST	<u>\$ 4.84</u>

TABLE VI

POWER

Power is purchased from the Northern Canada Power Commission's hydro electric plant on the Mayo river near the town of Mayo. The plant has two waterwheels with a rated capacity of 3000 H.P. each. Power is transmitted to Elsa a distance of 28 miles at 69,000 volts and on to Keno City a further seven miles at the same voltage. All power used at Elsa and Calumet is metered at Elsa.

At Elsa there are three banks of transformers of 2500 KVA, 1000 KVA and 1500 KVA which step down the voltage from 69,000 to 6,900, 6,900 to 2,300 and 6,900 to 460 volts respectively. In addition a three phase transformer rated at 2500 KVA reduces the voltage from 69,000 to 460 volts for use in electrode boilers.

Power is transmitted from Elsa to Calumet at 6,900 volts where it is stepped down by a 1500 KVA transformer to 2300 volts for mine usage. A 50 KVA transformer reduces the voltage from 2300 volts to 460 volts for surface use. A third bank of transformers of 600 KVA capacity reduces the voltage from 2300 to 460 volts for use in the electrode boilers.

Power used by the Company is paid for at the rate of 2 5/8¢ per Kilowatt hour for the output from the No. 1 unit while a flat charge of \$3800.00 per month is made for the output from the No. 2 Unit. During the 1959 fiscal year a total of 32,402,500 K.W.H. of power was used at an average cost of 1.41¢ per Kilowatt hour.

Three diesel driven alternator units totalling 1,273 KVA are housed at Elsa as a stand-by in case of power outages. These are sufficient only to handle essential services such as mine pumps, heat, light and water and the agitators and thickeners in the mill cyanide plant.

CAMPS AND SERVICES

Permanent townsites are maintained by the Company at Elsa and Calumet and in addition two small development mining camps are presently being operated.

Camp buildings generally are of insulated frame construction and in addition to the standard mine plant buildings of mill, compressor houses, machine shops, electrical shops, change houses, warehouses and offices, includes recreation halls, curling rinks, coffee shops, beer parlors and other sundry buildings. Staff residences are largely of 4 inch Panabode construction and are equipped with electric ranges and are fully modern. Other houses are of standard frame construction. All plant buildings and staff residences at both camps are heated from central steam heating plants. Steam, condensate, water, sewage and fire protection lines are carried in wooden boxes made of 2 inch shiplap with the pipes insulated by wrapping with blanket type ultralite insulation.

Both coal and electrical energy are used to produce steam for heating the plants and camps. A semi-bituminous coal having a calorific value of 11,500 B.T.U. is mined at the Company's coal mine at Carmacks on the Yukon River 185 miles south of Elsa. The coal is delivered by

incoming Transport Division trucks and is laid down at the property at an overall cost of \$25.00 per ton.

Electrode boilers were installed to take advantage of the excess and off peak power made available when the second 3000 H.P. water wheel was installed at the hydro plant late in 1957.

At Elsa the heating plant contains an H.R.T. steam boiler rated at 200 H.P. equipped with automatic controls and a Foster-Wheeler spreader stoker, two Supreme fire tube boilers rated at 100 H.P. each equipped with automatic controls and Iron Fireman underfeed stokers and two 1200 K.W. Bastian and Allen Electrode type boilers. At Calumet the heating plant consists of a 100 B.H.P. and a 150 B.H.P. Supreme fire tube boilers with automatic controls and underfeed Iron Fireman, and one each of 450, 150 and 70 K.W. Bastian and Allen electrode boilers. One of the 1200 K.W. electrode boilers at Elsa has automatic regulating equipment capable of shedding load during peak demand and picking up load during periods of slack demand.

Water supply for the plants is largely from underground workings and is sufficient for industrial and domestic use and for fire protection. Domestic water for Elsa is pumped from Flat Creek a distance of 4100 feet and is sufficient except in late winter when it is augmented by water piped from Calumet or Birmingham mines. The water has a hardness of 160 + p.p.m. and is chemically treated through a cold soda-lime softening process and calgon and chlorine are added. Domestic water for Calumet is pumped from underground

and has a hardness of 450 + p.p.m. and is treated in a similar treatment plant.

Both the Elsa and Calumet camps have two tanks or reservoirs of 100,000 gallons capacity for fire protection, one a gravity or primary tank giving a pressure of 80 p.s.i. on the system and a secondary tank which obtains its pressure from a gasoline driven centrifugal pump. The plants at both Elsa and Calumet are protected by a Grinnell sprinkler system using both wet and dry valves.

TRANSPORT DIVISION

When mining operations were commenced in 1946, supplies were shipped by river steamer from Whitehorse to Mayo and concentrates were taken out on the return trip. With the completion of the highway linking Whitehorse and Mayo a Transport Division was established in 1952 to handle trucking requirements and in the following year a similar division to handle the needs of Cassiar Asbestos Corporation, was set up under the same management. Today 126 pieces of equipment are operated by the joint Transport Divisions and 139 persons are employed.

The Transport Division supplies highway transportation and local needs at the mine. It operates a dispatching and maintenance terminal at Whitehorse and directs the operation of the West Indies Wharf in Vancouver where concentrates are unloaded and trans-shipped.

Since 1952 freighting costs have steadily decreased due to improved roads, installation of bridges over the major rivers, increased use of mechanized handling equipment, improved operating efficiencies and conversion from small gasoline trucks to large diesel tractors with

semi-trailers capable of hauling 26 ton loads. In 1959, 9500 tons of mining supplies, coal and forest products were trucked to the mine and 35,200 tons of concentrate were hauled from Elsa to Whitehorse. The cost of trucking the concentrate was \$18.57 per ton.

Operating and maintenance problems are ever present due to gravel roads, seasonal changes and winter weather. An interesting feature is the operation of highway equipment in temperatures down to minus 76° F in low areas. Successful operation can be maintained at such temperatures providing the units are kept moving and that exposure is less than four hours. Lower temperatures or longer exposures result in congealing of the diesel fuel. In the interests of safety, convoys travel at night to prevent "whiteouts" during the winter when it is difficult to distinguish snow covered roads in poor daylight. The last two units of the convoy travel together during winter as a precaution against a mechanical breakdown of the last unit.

*** **** **

Mining	8.50 / ton	} \$23.14 / ton] say \$25 / ton.
Milling	4.84		
Transport	\$18.50 to W.A.	} \$50 to smelter.	
Home	.76 / ton		

~~Agave~~

Mamfeld

"central gte 6" - 5" beds
1-12" graphic mat.

Hector Calmet ~~from fault~~ movement mainly horizontal | gte leached in places
S side is looks like loose sand

3 vein up to 57' massive S₂ - Loop structural belt
Zn up but Ag/Pb same @ depth.

oxidized
400'
dep

Also
Purcupine Pt is thrust fault (diff from most)

Set of - & is vein traced 200 to 600 level.

Ag higher Pb lower, hardly any Zn. compared to Calmet.

Ruby Ag° Tarsites make ore.

In Mn sideite at brecciated gte. | no leaching hard crushed gte

deepest - 650' level 500' below surface.

Ave Zn 0.1%

have to map every working piece so don't miss small branches
waist level - logging & sample.

75-80% recovery

Mining

→ Ag King used sinkage stope @ King till below
first, then abandoned it due to solution from the
55% recovery at ore by stoping method used.

Discussion:

Truck 18.50/ton.

x Flts.

movement, much is horiz
normal at least depth... }
thrust

Slides within
30° of horiz

Age - mostly post-ore.

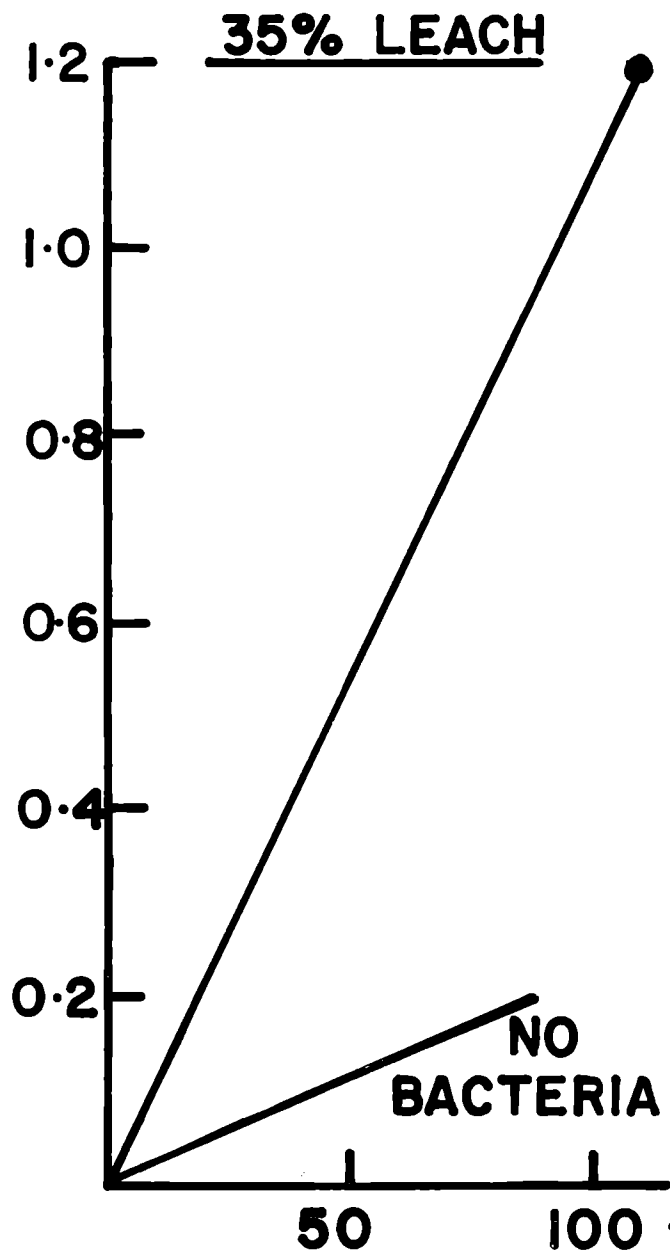


FIG. 5

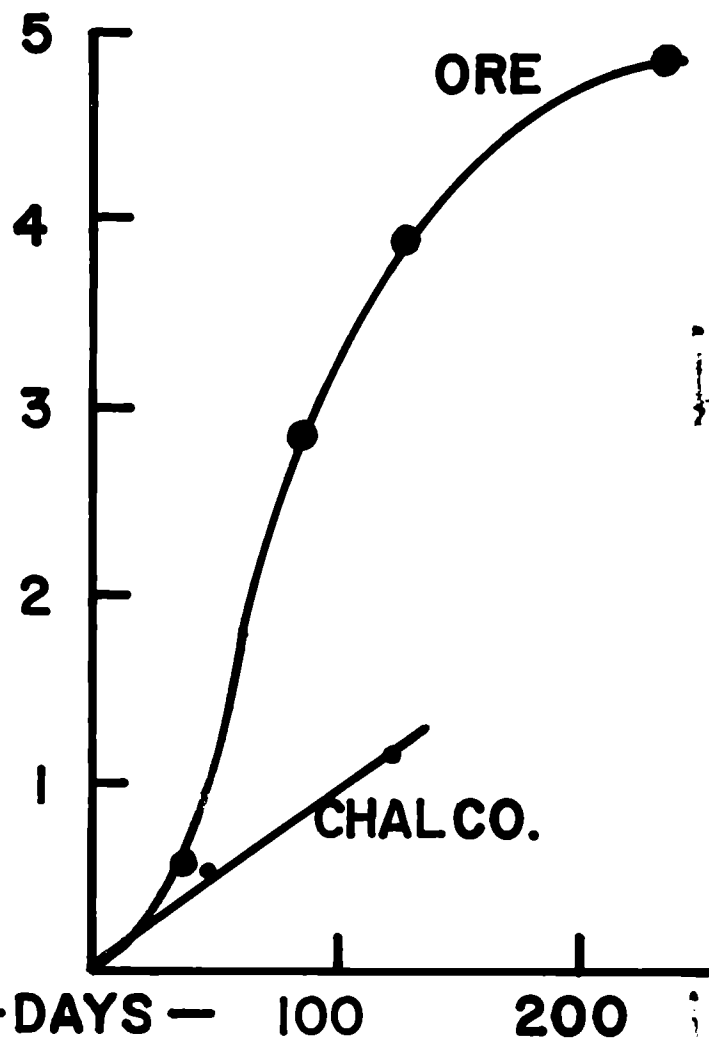


FIG. 6

GRAMS COPPER PER LITER
 FROM CHALCOPYRITE FROM ROCHER ORE
 STATIONARY
 BOTTLES