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The Mineral King Mine

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ABSTRACT

The Mineral King Mine, owned and operated by Sheep Creek Mines is located in the Purcell mountain range of southeastern British Columbia. Production of lead and zinc concentrates commenced in 1954 some 60 years after the initial discovery. Low cost mining methods have been employed to extract 16,000 tons of ore monthly from large ore bodies formed by replacement in limestone. Extensive areas in the limestone are yet to be explored and success in the search for new ore bodies is anticipated.

* * *

INTRODUCTION

THIS paper describes the Mineral King lead-zinc operation of Sheep Creek Mines Limited, Nelson, British Columbia.

Sheep Creek Mines Limited was formed in 1933. It has successfully operated since that date the Queen Gold Mine at Salmo, B.C., the Lucky Jim Mine at New Denver, B.C., the Paradise Mine at Invermere, B.C., and the Mineral King Mine, also near Invermere.

In December of 1954 Mr. F. R. Thompson, at that time General Superintendent of Sheep Creek Mines, presented a paper at the 60th Annual Convention of the Northwest Mining Association, Spokane, Washington, entitled "The Mineral King Mine: From Prospect to Production." Some of his material is included in this paper.

Location and Property

The Mineral King Mine is in the Golden Mining Division and 28 miles southwest of Invermere, B.C. To those not familiar with the southeast part of the Province we might describe Invermere as a small town in the Windermere Valley, some two hours drive by car north of the well known Sullivan Mine at Kimberley, B.C.

The property consists of 32 claims of which 25 are Crown Grants. The mine and camp are located on the southerly slope of the ridge which forms the divide between Jumbo and Toby Creeks. The camp elevation is 4400 feet above sea level. The hillside is steeply sloped, sparsely wooded, and the surrounding terrain rugged and mountainous.

History

The property first received mention in the report of the Minister of Mines for British Columbia in 1898. Prior to 1950, records indicate various prospectors and companies examined and owned the property. Work reported included 90 feet of surface trenching, 390 feet of tunnelling on two levels at elevations above 5600 feet, and shipment of one ton of high grade ore. In 1950 Sheep Creek Mines re-examined and optioned the property.

Early Developments by Sheep Creek Mines

In the fall of 1950 four short holes, drilled with an X-ray drill from an open cut, intersected ore. The following spring the Toby Creek road from Invermere was rehabilitated and three additional miles built to the mine site. By March 1952 the road was completed, a small camp established, and underground exploration started.

By May of 1953 development work to the extent of 6800 feet of diamond drilling 1200 feet of tunnelling, and 100 feet of raising, from the two levels above the 5600 foot elevation, had been completed. Approximately 325,000 tons of ore assaying 1.4 oz. Ag, 2.1 per cent Pb, and 5.2 per cent Zn were indicated at a total exploration cost of $0.31 per ton.

A decline in the metal content in the ore at the Lucky Jim Mine, and a depressed zinc metal price, decided the Sheep Creek Company to begin production at the Mineral King, utilizing plant, equipment, and personnel from their other operations.

Plant and camp sites were prepared at the 4400 foot elevation. Construction of plant buildings and movement of equipment started simultaneously. A new 3 level adit at elevation 5450 feet was started for ore haulage purposes and for new ore development possibilities. A 1500 foot surface incline with 40 cu. ft. skips in counterbalance connected 3 level and the mill ore bins.

In March 1954 this work was completed. Total expenditures on mine and plant by using machinery from the other operations were slightly under $500,000.00. The concentrator was started in April and since the tune up period has operated continuously.

The following year the 7 level main haulage adit was started at mill elevation. Upon completion an internal 800 foot two compartment inclined service shaft, paralleled by ore and waste passes, was driven to 3 level. Three additional levels, 4, 5, and 6, were then developed. All ore and waste are now sent to their respective passes and trammed ont via 7 level.

Geology

The Mineral King Mine is in the Purcell mountain range about twenty miles west of the Rocky mountain trench. The area is underlain by moderately metamorphosed Pre-Cambrian sedimentary rocks of the Upper Purcell and Windermere series. The general geology of the area has been described by J. F. Walker and J. E. Reesor. (1) (2)

The formations of the Upper Purcell series lie in tight folds, plunging in a northwesterly direction, and above these unconformably lie the Windermere series. Major faults are near vertical, parallel to the strike of the sediments, and are pre-ore in age. Small greenstone and lamprophyre dykes, highly altered and probably Pre-Cambrian, occur. Regional deformation and mineralization are Jurassic.

The ore bodies were formed by replacement of limestone near the lower contact of the Mount Nelson formation. Replacement is closely associated with layers of olive-green, grey, or black schist which represent the quartzite and argillite of the Mount Nelson formation. Structure is not apparent in the grey dolomitic limestone but the bands of schist in-

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dicate a series of minor folds on the east limb of a major syncline. Interpretation of early exploration indicated four parallel ore zones. For mining purposes they were named from west to east, A, B, C, and D. The ore horizon is continuous between zones in some areas.

The ore is composed of galena, sphalerite and a bourncnite-tetrahedrite mixture in a gangue or barite, quartz and calcite. Composition of the ore varies from west to east. The A zone averages 2 per cent Pb, 3 per cent Zn and 1 oz. Ag per ton in a well banded siliceous gangue; the D zone averages 1 per cent Pb, 6 per cent Zn and less than 0.5 oz. Ag per ton in a predominantly barits gangue. The buornonite-tetrahedrite mixture, containing copper, is associated with barite. The C and D zones locally become almost pure barite, with negligible sulphides.

The ore ranges in thickness from one foot to 40 feet. The A zone, which has been a major source of ore, has been mined from surface for 1200 feet down the plunge. It is up to 50 feet wide and as much as 150 feet high.

Between 3 Level and 4 Level, there is a transition in the A zone from replacement along schist layers to replacement along a major strike fault in the limestone. Mineralization is similar throughout, although the Lower A zone is much higher in grade. This lower ore was probably formed along the channel followed by the mineralizing solutions. The ore developed in this zone has a vertical extent of 325 feet, an average length of 200 feet and widths up to 30 feet.

Development from the lower levels has indicated that the productive limestone is 500 feet thick. It has been partially explored for 2000 feet down the plunge, and over a horizontal extent of 800 feet. No work has been done on scattered surface showings, or below 6 level. Large favourable areas of the property have yet to be explored.

TABLE OF FORMATIONS

Jurassic		Mineralization — barite, quartz, sulphides Greenstone and lamprophyre dykes
Pre-Cambrian	Windermere series Toby conglomerate	
	Unconformity	
	Upper Purcell series	Mount Nelson formation — limestone, minor argillite and quartzite
		Dutch Creek formation — black slate.

Mining

The mine to date is developed and serviced from 6 levels and the internal service shaft.

Due to the parallel nature of the ore bodies, and their overall width, practically all drifts and crosscuts have been driven on line and grade with minimum radius curves of 60 feet and 8 feet by 8 feet in cross section. Raises are driven 6 feet by 6 feet in cross section, untimbered, at 50 degree pitch.

Track gauge is 24 inches. Main haulageways have 30 pound rail, carrying 4 ton diesel locomotives and 60 cubic foot Granby type cars. Intermediate levels have 20 pound rail, 1½ or 3 ton battery locomotives, and 35 cubic foot side dump cars.

Power is transmitted underground by armored cable at 2300 volts to a central transformer station. Here voltage is transformed to 440 volts and transmitted by armoured cable to permanent control panels on each level. From these panels power is distributed in rubber covered power cable to the various electrical installations. The control panel also provides 110 volt lighting outlets, and 220 volt blasting circuits.

Ventilation is maintained at 15,000 c.f.m. of air, downcast from 2 level to 7 level, by means of a 10 h.p. axivane fan installed on 2 level. Additional fans along the fresh air current provide ventilation to inaccessible headings.

Light weight stopers are used on raise work and, with the exception of one long hole machine, all other drilling is done with jackleg mounted drills.

Tapered sockets, four wing tungsten carbide bits, and 7/8 hexagon alloy drill steel are used throughout. A 70 per cent Cilgel explosive is used on all headings. Shots are fired electrically in most instances using milli-second delay caps in stopes.

Air and electric scraper hoists, ranging in size from 10 h.p. to 50 h.p., are employed in stopes pulling 36 inch to 48 inch heavy duty scrapers. The larger chutes are air operated, smaller ones hand operated.

Stoping Practice

Actual stope mining practices have changed very little since the mine commenced operating in 1954. Above the 4 level standard open stoping methods have been employed. The general procedure is to make draw point and manway entries. From this point a face is advanced by slashing to the upper limit of the ore. The stope is then slashed to full width and the back cleaned down to complete safety. Where practical a ventilation raise is driven to the level above. When the broken ore ceases to run by gravity, scraper hoists are employed. When a top slice has been completed, benches are then advanced in succession until the footwall of the ore is reached.

Because of the irregularity of the ore bodies, several deviations from this general plan can occur. It is seldom possible, or practical, to completely outline an ore body by diamond drilling. One to three holes are usually attempted. If the ore back cannot be reached on the first upward stope advance a spiral system becomes necessary. If the ore continues to climb, a new entry from the level above may be desirable. Where stope widths become excessive pillars are left along the stope walls in a suitable pattern to maintain a scraping channel. In some instances the stope heights become excessive and horizontal rib pillars are left at narrower ore zones. Stopes vary in length from 50 feet to 300 feet, in width from 10 feet to 70 feet, and in height from 20 feet to 150 feet.

In 1957 one area looked particularly adaptable to long hole drilling methods. It was decided to experiment on a block of ore containing 80,000 tons. Holes 2¼ inches in diameter by 32 feet in length with 6 feet of burden were drilled. The high dilution factor resulting has not encouraged further long hole drilling except for pillar removal.

Below the 4 level, ore encountered to date, appears to be controlled by a near vertical major fault. The ground adjacent to this fault zone is incompetent and only shrinkage stoping methods have been tried. Irregularity of this ore zone has required extra stope development in the form of scraper drifts and additional draw points.

In most ore zones a fairly well defined wall is apparent, and mining limits can be controlled visually. If the ore zones become marginal in grade sampling is necessary. Samples from test holes 10 feet to 15 feet in length drilled intermittently in walls and back check complete ore recovery. Average mining dilution is 8 per cent although some fault zones have caused up to 23 per cent dilution. Pillars and sills contain 15 per cent of the present ore reserves.

All underground diamond drilling, development, stope mining, scraping and haulage is done on a bonus system. The bonus is paid twice monthly and averages $4.50 per man shift.

As mentioned previously the predominant gangue mineral in C and D zones is barite. Substantial tonnages of marketable barite are mineable in these two zones above 3 level. A contract was negotiated for delivery of 10,000 tons of uncrushed barite ore, in 1959. The mined barite is moved through worked-out lead-zinc stopes or raises and trammed to the old surface incline where it is lowered to a newly constructed ore bin. No underground development work will be required to fill this order.

Concentrator

The concentrator, capable of handling 625 tons per day of average mine grade ore is currently operating at 85 per cent capacity.

Ore from two 400 ton capacity coarse ore bins is belt fed to an 18 inch by 36 inch jaw crusher. The crushed product is conveyed to a double-deck vibrating screen over a 500 ton capacity fine ore bin. Oversize from a 7/16 inch screen is returned to a three foot gyratory crusher in closed circuit with the screen.

The crushed ore is belt fed to a 7 foot by 5 foot ball mill operating in open circuit with a 4 foot by 23 foot rake classifier. Classifier sands are fed to a 6 foot by 3 foot conical ball mill in closed circuit with the same classifier. Classifier overflow is pumped directly to a nine cell lead flotation circuit. Lead concentrate is pumped to a thickener tank and the thickened product to a 4 foot, 4 leaf filter. The filter cake discharges directly to a 400 ton capacity lead concentrate storage bin.

The tailings from the lead circuit discharges into a 6 foot by 7 foot conditioner tank where it is conditioned and sent to a 10 cell zinc flotation circuit. Zinc concentrate is pumped directly to a 6 foot, 4 leaf filter. The filter cake discharges to an 800 ton zinc storage bin. Concentrates are loaded from storage bins into trucks by scraper hoists and trucked 28 miles to a railway siding in Invermere. Tailings are discharged by gravity to disposal stack.

MINE AND MILL PERFORMANCE

The following operating statistics may prove of interest to other operators:

Mining Costs 1958

Direct Mining	$2.191
Direct Milling	$1.186
Concentrate Haul	$ .321
Indirect Expenses (administration, insurance etc.)	$ .673
Total	$4.371

Underground Performances 1958

Tons won per foot of development	55.0
Tons won per foot of Diamond drilling	32.0
Average feet advanced per manshift drifting	3.6
Average feet advanced per manshift raising	4.1
Average feet drilled per miner shift	171.0
Pounds of explosives per ton stoping	0.72
Average feet drilled per throwaway bit	183.0
Total H.P. connected load	750

Mill Performances 1958

Average tons milled per day			567
Percent Running time			95.12
Ratio of concentration			11.6
Average mill heads assay	Ag	oz/ton	0.58
	Pb	%	2.20
	Zn	%	5.57
Average Concentrate Assay	Pb	%	62.6
	Zn	%	55.8
Average tailings assay	Ag	%	0.07
	Pb	%	0.35
	Zn	%	0.48
Mill Recovery (metal paid for)	Ag	oz/ton	84.31
	Pb	%	77.31
	Zn	%	89.60
Total H.P. connected load			540

Mine and Mill Services

A 40 foot by 82 foot steel building houses the diesel-electric power plant, compressor plant, and work shop. Five diesel generating units have a generating capacity of 1275 k.v.a. Three electric driven compressors furnish 2250 c.f.m. of compressed air. Steel buildings are also used for office and steel sharpening shop.

Camp buildings are mainly of prefabricated construction and include change house, cookhouse, staff house, commissary, school, community hall, five bunkhouses and 17 family dwellings. A trailer camp was established in 1958 and five employee-owned trailers are now installed.

Main camp and plant buildings are steam heated from a central coal stoker-fed heating plant.

An adequate water supply is obtained from Stark Creek. The water is delivered to a pumping station near Toby Creek at 100 foot head and pumped by a 50 h.p. two stage centrifugal pump against a 425 foot head to mill and storage tanks.

Besides a community hall, recreation is available to employees in the form of a ball park, curling rink, and a billiard room.

Mine and Mill Performance

The operating statistics tabulated above may prove of interest to other operators:

Summary

Production at the Mineral King Mine commenced in April 1954 and since that date the concentrator has treated 725,000 tons of lead-zinc ore.

Open stope mining methods in large ore bodies, have been the key to low mining costs. Results of recent exploration in the lower levels indicate slightly higher mining costs, offset by a higher metal content. A continued successful operation is expected.

References

(1) Walker, J. F. Memoir 148-1926 Geological Survey of Canada.

(2) Reesor, J. E. Map 12-1957 Geological Survey of Canada.

Undercutting at the King Mine of Asbestos Corporation, Limited

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ABSTRACT

The purpose of an undercut is to undermine a block of ore which will then proceed to cave into draw holes just below the undercut. Undercutting is the final and most vital phase in the development of an ore block. Since its purpose is to undermine and induce caving into a block of ore, great care must be exercised to avoid premature caving by removing only enough rock to permit undercut development to proceed.

Several undercutting methods, used over a twenty-year period, are reviewed and their suitability is compared with ideal requirements.

* * *

Introduction

THIS paper will review undercutting methods at the King mine of Asbestos Corporation, Limited, over a twenty-year period.

Efforts to increase undercutting efficiency at King were first directed toward the pillar drilling techniques, which were greatly improved by the introduction of new drilling and blasting equipment.

Lately, effort has been directed to reducing development drilling to a minimum by eliminating the bell out. This has been achieved with considerable success.

Main emphasis will be placed on our latest method, which was introduced some eighteen months ago.

The King mine is the underground component of the recently established King-Beaver operation. King-Beaver operations consist of a 5,000-ton asbestos mill on the former Beaver property, an underground mine on the former King property, as well as open pits on the Beaver and Bennett Martin properties.

Block caving has been carried on at the King mine since 1932 when the first block was undercut on the 455-foot level(1). Ore is presently being mined on the 700-ft. and 855-ft. levels.

Present-day block layouts are quite similar to the original, although the shapes have been altered sometimes to conform with variations in the ore contact(1, 2). Many changes have been introduced in the design of rock openings and support methods, especially those openings under caving blocks(3).

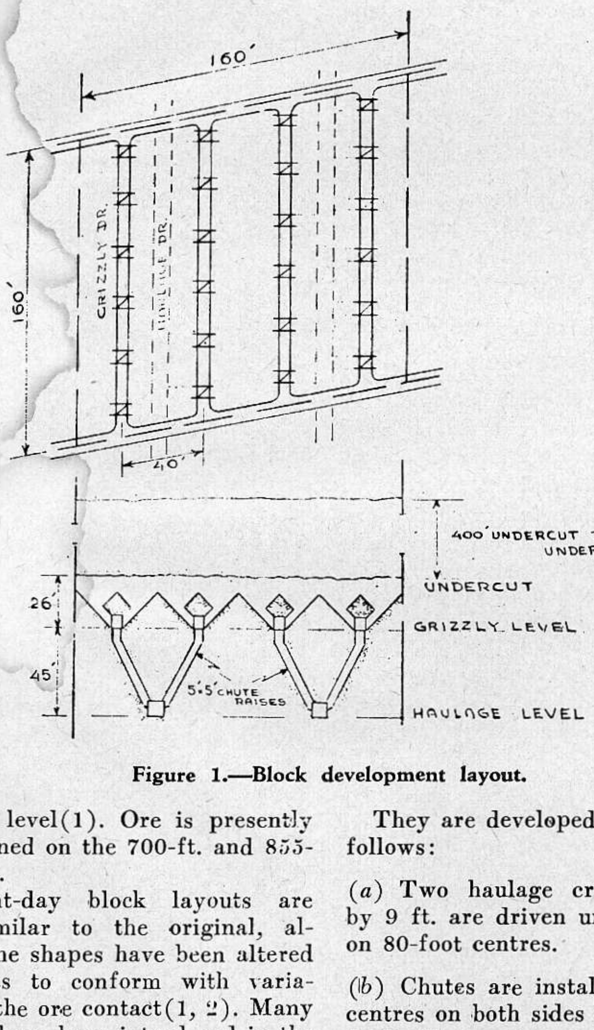

Figure 1.—Block development layout.

Block Layout — Grizzly and Haulage Levels

(Figure 1)

A typical block in plan view has 160-foot sides. Blocks are generally 400 feet in height.

They are developed for mining as follows:

(*a*) Two haulage cross-cuts 8 ft. by 9 ft. are driven under the block on 80-foot centres.

(*b*) Chutes are installed on 26-foot centres on both sides of the haulage drift. Chute raises 5 ft. by 5 ft. are driven to the grizzly level.

(*c*) Four grizzly drifts are driven on 40-foot centres, parallel to the haulage drifts and 45 feet above them.

(*d*) Steel draw points are constructed at the top of each chute raise. The grizzly drift between draw points is supported by an elliptical concrete section (3).

Upon completion of this development work, the work of undercut-

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(1) For this and other references see end of paper.