"THE CASSIAR STORY"

Tom Schroeter
Apr. 28/78

Introduction

Brian G. Pewsey, Mine Manager,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

Located in the northern part of the Province of British Columbia, the Cassiar Mine started development of its asbestos orebody in 1952, and is currently producing 105,000 tons of asbestos fibre annually. The townsite is nestled in a beautiful valley and supplies more than average facilities. The fibre is shipped to more than 40 countries and is used in many industrial products.

Location, Access and Climate

THE CASSIAR MINE is located at latitude 59°20′N, longitude 129°49′W, in rugged, mountainous terrain 50 miles (80 km) south of the Yukon border, 735 air miles (1177 km) northwest of Edmonton and 200 air miles (320 km) southeast of Whitehorse (see location map, Fig. 1).

Access to the property is by an all-weather road running 100 miles (160 km) southwest from Watson Lake (1044 km) on the Alaska Highway. The Cassiar road is a northern portion of the road to Stewart (Highway 37) on the Pacific coast.

Cassiar is in the Cassiar Mountain range, with the town and plantsite nestled in a typical glacially formed valley at an elevation of 3525 ft (1074 m). The open-pit mine located on McDame Mountain, at an average elevation of 6000 ft (1830 m), some 3 air miles (4.8 km) north of the plantsite.

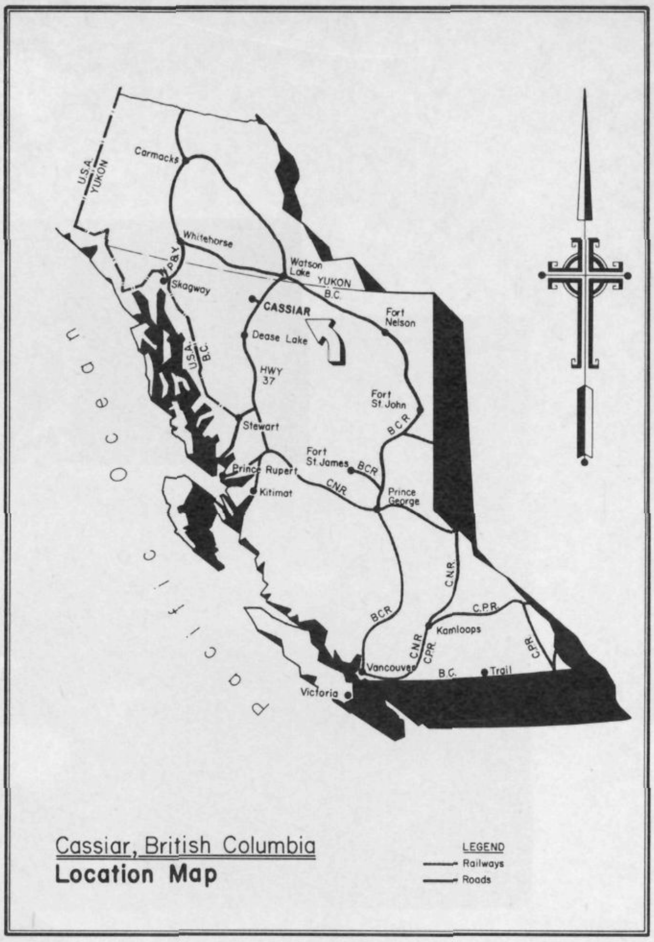
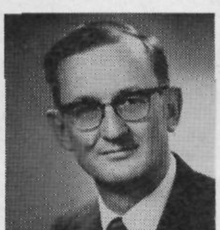


FIGURE 1 — Location of the Cassiar Mine.

The average winter temperature at Cassiar is in the range of -10 to -15°C. The winter season extends from October to May, and during this time there is an over-all snowfall of approximately 116 in. (295 cm). The mountain ridge to the south of the community permits only diffused sunlight to reach the town during December and January. During the spring and early summer months, temperatures reach 21°C under sunny skies. The latter part of the summer and fall is characterized by cloudy weather, with an average rainfall of 16 in. (40 cm).



Brian G. Pewsey was born in Zambia and received his primary education in South Africa. He joined Rhodesian Selection Trust in 1954 as a learner surveyor and by 1959 had gained his higher and advanced national certificates in mining, mining economics and surveying. In 1961, he entered the Camborne School of Metalliferous Mining, receiving his associateship in 1964 when he returned to Rhodesian Selection Trust in Zambia as an underground mine official. In 1966, he immigrated to Canada, joining the Iron Ore Company of Canada at Schefferville, Quebec, as mine engineer, progressing to supervising engineer, mines, by 1969. In 1970, he joined Rio Algom Ltd. in Toronto as mining engineer, working on feasibility studies and field exploration programs. In 1974, he joined Cassiar Asbestos Corporation Limited as mine superintendent and became mine manager in 1976. He is a member of the CIM and the Association of Professional Engineers of British Columbia and Ontario.

Keywords: Cassiar Mine, Asbestos, Townsite, Services, Manpower, Transportation.

History

The existence of the asbestos deposit had been known by native hunters for many years before development work commenced. Stories were told of mountain sheep bedding down in matted fibre that accumulated from the weathering of the outcrop. In 1950, four prospectors, Richard Victor Sittler, Hiram Nelson, and the brothers Robert and Ronald Kirk, staked the property, but it was not until 1951 that improvements in transportation and economics made development work possible. The Conwest Exploration Company Ltd. of Toronto acquired the holdings in 1951 and formed Cassiar Asbestos Corporation Limited, which commenced the development of the deposit.

The first ore was mined in the fall of 1952, when talus ore was trucked to a mill rated at a capacity of 250 tons per day. The transportation of the ore was originally by truck from the mine to the mill along 6 miles (9.6 km) of steep, narrow and inadequate roads. In 1953, a 180-degree concave steel chute was con-

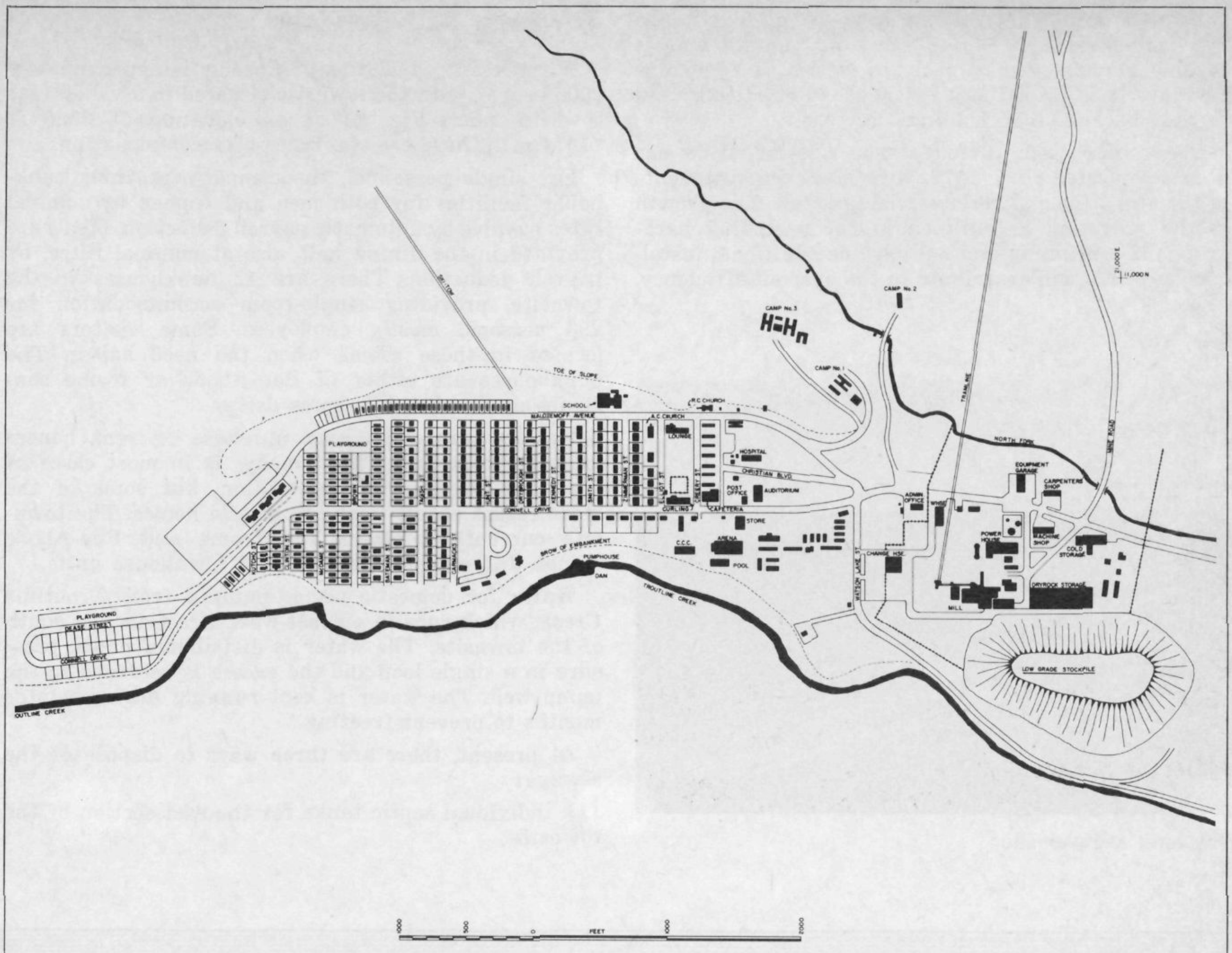


FIGURE 2 — Townsite plan at Cassiar.

structed and used to transport the ore from the 6400-ft (1950-m) elevation to the 4800-ft (1463-m) elevation, a drop of 1400 ft (427 m), where it was loaded and transported to the mill in 10-ton trucks. This chute, however, became constantly clogged and was abandoned in 1956 in favour of a 14,600-ft (4450-m) aerial tramline which extended from the mine at elevation 5800 ft (1768-m) to the mill at elevation 3525 ft (1074 m). The operating capacity of the tramline was 1800 tons per day, with each of the 180 buckets carrying ¾ ton net weight.

The mine had been developed and operated as a summer operation from 1952 until 1962. The reason for this part-yearly operation was the fact that sufficient ore could be mined and transported during the summer months to satisfy the yearly feed to the mill. By 1962, the strengthening of markets and a greater demand for the Cassiar fibre eventually resulted in a year-'round mining and milling operation. An important addition was a rock reject plant, located at the mine site, to concentrate the ore and improve the grade of the fibre transported to the mill stockpiles. During this time, the mill had progressed from a 250-tpd operation to a 1600-tpd operation. As improvements and additions were being made to all operations of the plantsite and mill, the tonnage increased yearly until it reached 2000 tons per operating day by 1966.

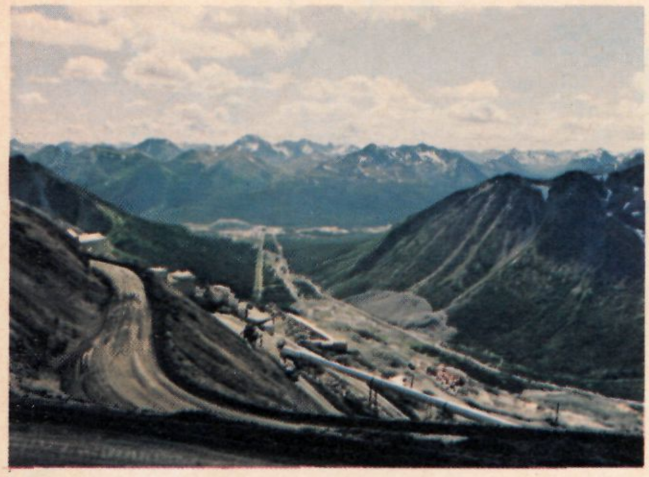
The progression, in a part of the province considered remote and hostile, from a small seasonal operation to a 12-month operation had shown that the efforts and perseverance of the Cassiar management team had been well justified.

The operation, by 1966, was increasing in productivity at a rate that required a change in over-all concept. It was becoming increasingly evident that the tramline was under capacity for the mill demand. The remedial approach was to improve and widen the existing 6-mile (9.6-km) haul road from the mill to the mine site and supplement the tramline with trucking, using trucks ranging from 28 to 60 tons capacity.

By 1970, the milling operation had progressed to a rated capacity of 2200 tons per operating day, with 36% of the total ore delivered from the mine by truck. The demand for fibre by 1970 had progressed to in excess of 75,000 tons of fibre produced; consequently, a new mill extension was constructed adjacent to the old mill, increasing the operating capacity of the mill as well as the fibre production from 250-275 to 400 tons per day and also increasing the amount of recoverable fibre by adding one additional cement fibre grade (AZ).

The tonnage processed by the new milling complex had, by 1975, increased to 3300 tons per operating day, with truck transportation outstripping tramline transportation by nearly 2 to 1. It was during this year that the new tramline was commissioned. Studies had been initiated as early as 1971 to determine the most economical method of transporting ore from the mine site to the mill. The tramline is 15,370 ft long (4685 m), travels at 600 ft (183 m) per minute and is capable of delivering slightly in excess of 6000 tons per day. It is loaded and unloaded automatically and has 142 buckets, each holding 1.75 tons.

The construction period at the Cassiar Mine has been accelerated since 1973, with many improvements to the operating and service-related areas. The growth of the operation as outlined in the preceding paragraphs is continuing and will include additional installations which will contribute to the over-all efficiency.



The town and plantsite.

Townsite and Services

The present population of Cassiar is approximately 2000 people, with the townsite situated in a valley (see townsite plan, Fig. 2) at an elevation of 3525 ft (1074 m). There are two types of accommodation.

For single personnel, the company provides bunkhouse facilities for both men and women at nominal rates payable by automatic payroll deduction. Meals are provided in the dining hall, also at nominal rates, by payroll deduction. There are 12 bunkhouses in the townsite, providing single-room accommodation for 293 persons, mostly employees. Some visitors are located in these rooms when the need arises. The bunkhouses are either of Pan-Abode or frame construction, or of mobile-home design.

Married employees may purchase or rent houses from the company. This housing is in most cases of frame or Pan-Abode construction, but some of the more recent acquisitions are mobile homes. The townsite currently contains 150 frame and Pan-Abode homes, 152 mobile homes and 18 townhouse units.

Water for domestic use is pumped from Troutline Creek, which runs in an east-west direction just south of the townsite. The water is distributed under pressure in a single loop and the excess is returned to the pump well. The water is kept running during winter months to prevent freezing.

At present, there are three ways to dispose of the sewage:

(1) individual septic tanks for the east section of the townsite;

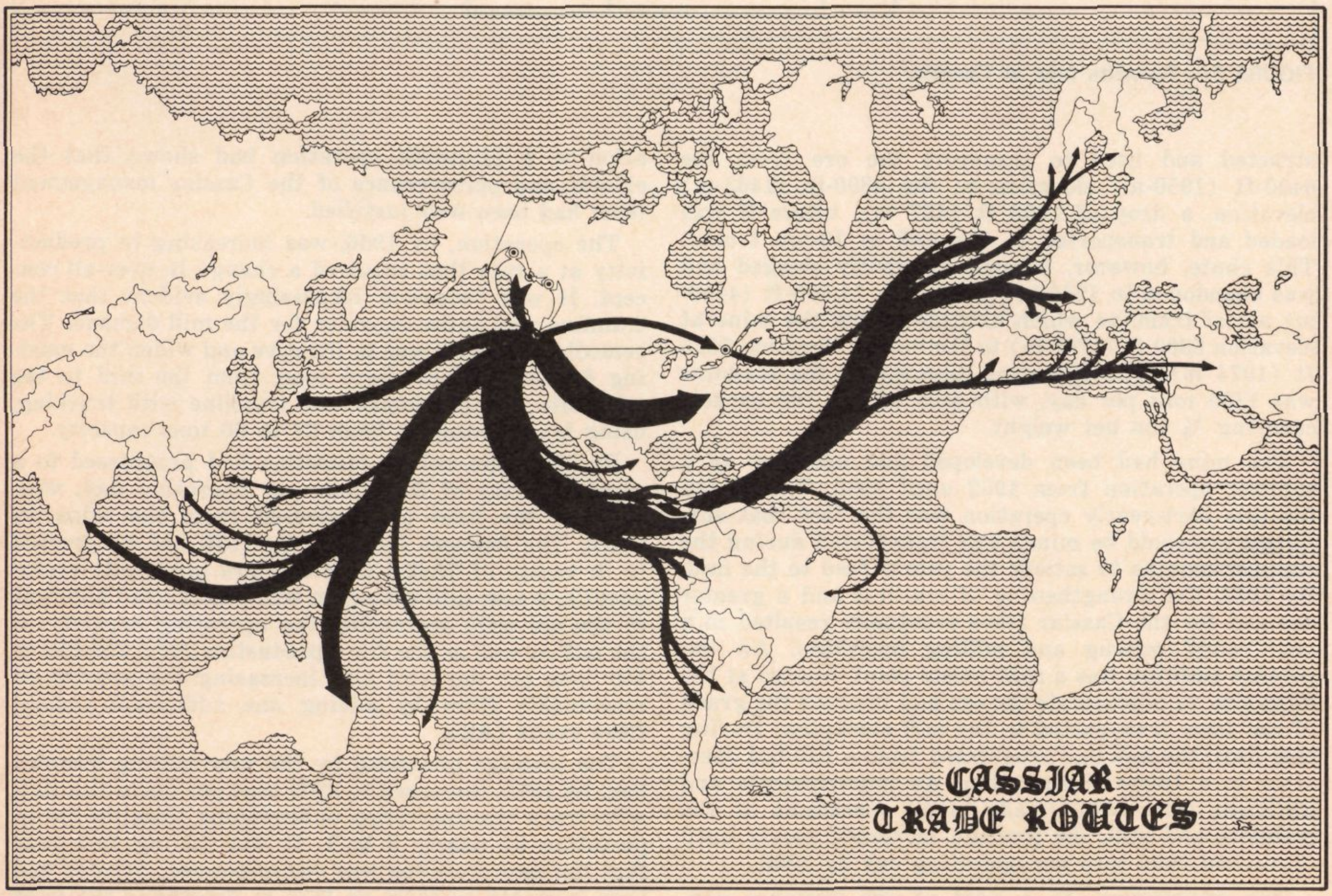


FIGURE 3 — Cassiar trade routes.

(2) collective septic tanks for the west section of the townsite;
(3) an aerobic treatment plant for the cafeteria and bunkhouses.

Cassiar has a small, but modern and fully equipped, six-bed hospital and dental clinic. Facilities include 2 two-bed wards, 2 single-bed wards, an operating room, 2 dental chairs, a nursing station and a kitchen. It affords complete provision for the thorough physical examination required by company safety and health standards, including equipment for chest X-ray, hearing, vision and respiratory testing.

The hospital staff includes two doctors, a dentist, head nurse and five staff nurses. In more serious cases or emergencies, the Rescue Coordination Centre of the Emergency Health Services Commission for the Province of B.C. is contacted and they dispatch an aircraft for transportation to Whitehorse, Prince George or Vancouver.

Other townsite services include snow removal, garbage disposal, and the operating and control of the dry goods and grocery stores.

Recreation facilities include: a community center with a library, lounges and a gymnasium; curling club (two sheets of artificial ice); swimming pool; arena with artificial ice; lounge; snack bar; ski hill with rope tow; tennis courts and playing field.

The business establishments in the townsite are: Royal Bank; C.N. Telecommunications; post office; transportation and bus company; liquor store; two gas stations; cable T.V. outlet; hairdresser; barber shop; two clothing boutiques; and many smaller and part-time stores providing sporting goods, snowmobile sales and repairs, radio and T.V. sales and repairs, and children's clothing.

An elementary-secondary school, with 17 teachers and 14 classrooms, offers instruction from kindergarten to Grade 12 level.

A small 3400-ft (1036-m) local air strip is available during the summer months. The CBC provides T.V. coverage through its ANIK satellite communication system.

There are two churches in Cassiar which serve all denominations, an R.C.M.P. detachment, and a fisheries and wildlife officer.

Fibre Transportation and Use

The transportation of the asbestos fibre from Cassiar, approximately 400 tons per day, is an ongoing and continuous process. Truck, train and ship transports are used to move container and palletized loads of fibre to Vancouver for onward shipment to overseas and North American markets. Currently, the fibre is routed by truck from Cassiar to Whitehorse in the Yukon Territory, where it is loaded on the White Pass & Yukon Route railway system and transported to Skagway, Alaska. At the Alaskan port, the asbestos containers are reloaded onto container ships for delivery to Vancouver. A certain amount of fibre is also transported by truck to Fort Nelson and Dawson Creek, British Columbia, where it is loaded onto the British Columbia Railway system for delivery to Vancouver, with the exception of a small amount going directly to the United States. Some fibre is trucked to the Cassiar Asbestos wharf in North Vancouver, but this is strictly as a back-haul situation.

The fibre is unloaded at the company's North Vancouver wharf, where it is stored by grade for final shipment to customers.

Cassiar Asbestos Corporation Limited ships its fibre to over 40 countries in the world (see attached plan of Cassiar trade routes, Fig. 3), where the product is used for spinning in the textile industry, mixing with cement, and in many other industries (as outlined in Table 1).

TABLE 1 — Cassiar Mine — Products and Industrial Use

Products	Industrial Use
AAA, AA, A, AC, Ac-60, AK-100	Spinning Fibres. Products are used in the textile industry for yarn, cloth, rope tape and wicks. Yarn and cloth are used for protective clothing, blankets, etc. The rope tape and wicks are mainly used for packing materials in water faucets and high-pressure pumps.
AK, AS, AX, AY, AZ	These products are used in asbestos cement products for pipe, flat and corrugated sheets and hand-moulded products such as pipe joints, flower pots, rain gutters and electric switch boxes.
AK 60, AS 60, AX 60	Products are used in gasket products mainly. However, some is used in brake linings for heavy industrial equipment.
AK 100, AS 100, AX 100, AY 120	Used in filter products for filtering beer, wine and water. Some AX, AY, AX 60 and AY 120 grades are used in electrical components as reinforcing agents with plastics

Geology

Fred G. Hewett, Mine Geologist,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

The Cassiar orebody is located in a sill-like serpentinite body intrusive into the Devonian-Mississippian sedimentary rocks of northern British Columbia. Subsequent intrusion of the Cassiar batholith formed the McDame synclinorium and produced chrysotile asbestos.

Present information indicates an orebody of approximately 16,277,000 tons amenable to open-pit mining, but continued diamond drilling will be needed to accurately define total tonnage and grade. A program has been developed at the mine to provide the required information on a daily basis, and work is continuing on more effective dollar-value prediction methods.

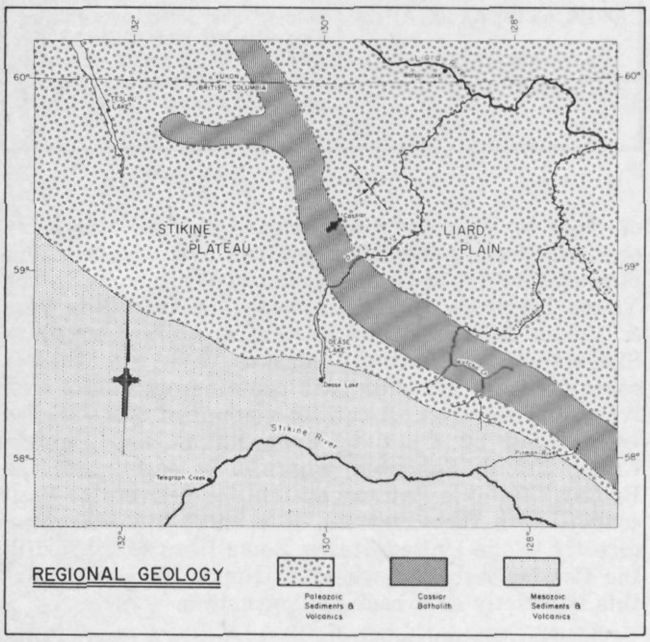
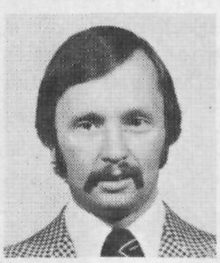


FIGURE 1 — Plan of regional geology.



Fred G. Hewett was born in New Denver, B.C., and graduated from the University of British Columbia with a B.Sc. in geology. His working background has been evenly divided between exploration and mining geology. He was employed by Conwest Exploration Company Limited as a junior geologist in the Northwest Territories and Ontario during 1966. In 1967, he joined Placer Development Limited as a technician and underground diamond drilling coordinator at Craigmont Mines Limited in Merritt, B.C., and subsequently transferred to Canex Placer Limited as a field geologist in 1969. He returned to Craigmont in 1972 as the exploration geologist, based in Kamloops.

In 1973, he joined Cassiar Asbestos Corporation Limited in his current position as mine geologist in charge of the Cassiar Mine geology department. He is a member of the CIM and a Fellow of the Geological Association of Canada.

Keywords: Cassiar Mine, Geology, Mine geology, McDame synclinorium, Sylvester Group, Structural geology, Drilling, Adit exploration, Grade control, Reserve estimation, Computers.

Geological Setting

REGIONAL

THE CASSIAR MINE is located approximately in the middle of a northerly trending belt of rocks which forms the spine of the Cassiar Mountains. This belt is approximately 45 miles wide, being bounded on the east by the Liard Plain and on the west by the Stikine Plateau (Fig. 1).

The Cassiar asbestos deposit occurs within the Sylvester Group — a thick eugeosynclinal assemblage of volcanic and sedimentary rocks of late Devonian to early Mississippian age. This assemblage forms a major part of the McDame synclinorium, a large package of predominately sedimentary rocks laid down from Precambrian to Mississippian time. Regional rock types are described in Table 1. The syncline was formed during the upheaval caused by the implacement of the Cassiar batholith in Jurassic-Cretaceous time. Occurring chiefly along the axis of the syncline, but also occasionally at other stratigraphic horizons, lies a string of ultramafic bodies known as the McDame Intrusives. These Mississippian bodies intrude the Sylvester Group intermittently for 70 miles and, although most have been converted to serpentine and contain chrysotile asbestos, so far only one — the Cassiar deposit — has proven economically viable.

LOCAL

The Cassiar orebody occurs in a sill-like body of serpentinite which intruded the west limb of the

TABLE 1 — Regional Rock Types

Age	Name	Description
Jurassic/Cretaceous	Cassiar Intrusions	Quartz-monzonite, granodiorite, etc.
Mississippian	McDame Intrusives	Serpentine and related rocks.
Devonian-Mississippian	Sylvester Group	Interbedded argillites, quartzites and volcanics.
M. to U. Devonian	McDame Group	Platy limestones, black dolomites and shales.
Silurian and (?) Devonian	Sandpile Group	Interbedded quartzites, sandstones, dolomites and minor phyllites.
M. to U. Cambrian	Kechika Group	Black to brown platy shales; often pyritic.
L. Cambrian	Atan Group	Interbedded marbles, dolomites, argillites and quartzites.
Late Precambrian	Good Hope Group	Coarsely crystalline limestone and marble.

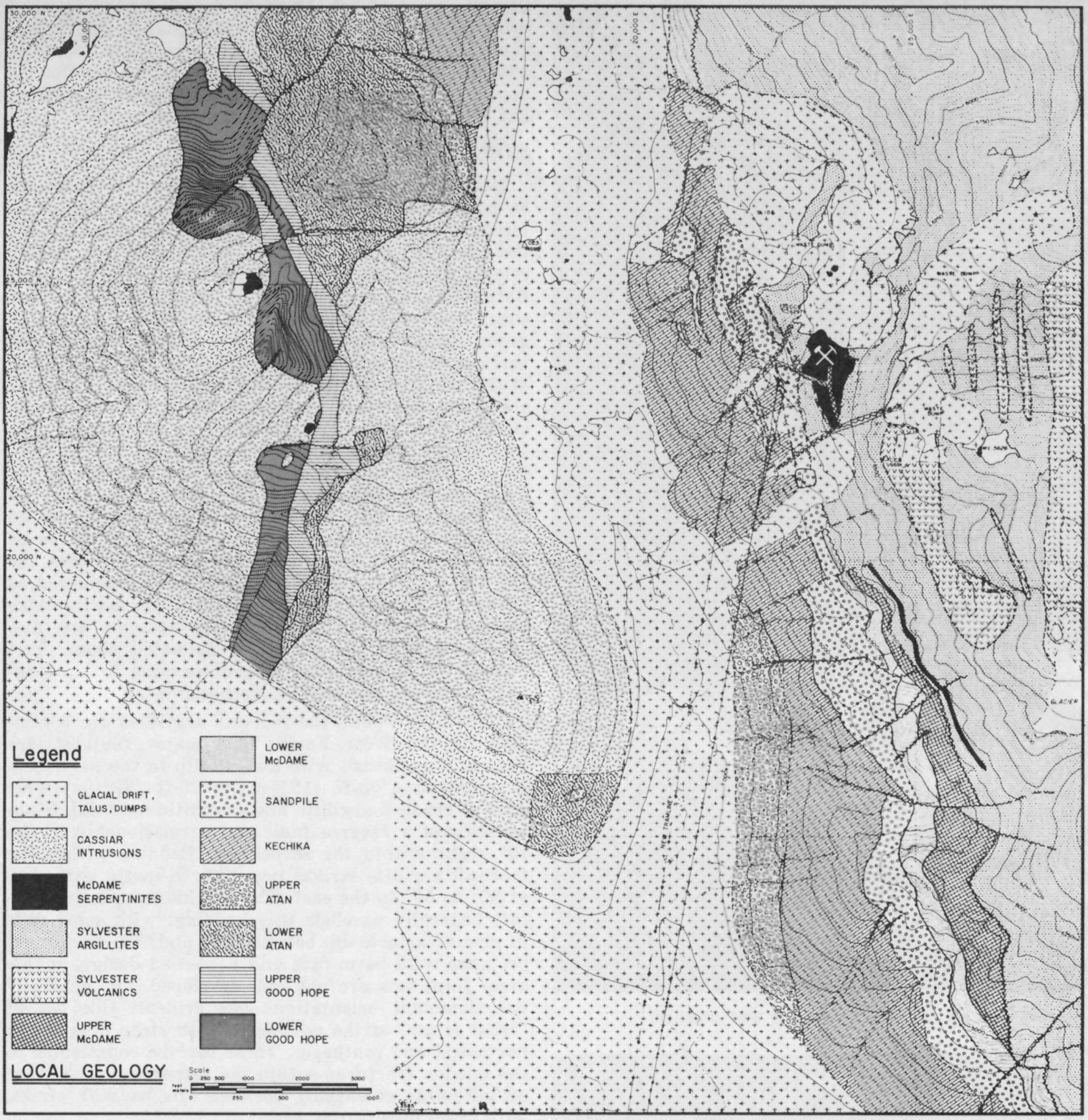


FIGURE 2 — Plan of local geology.

McDame syncline near the base of the Sylvester Group (Fig. 2). The orebody strikes approximately north-south, with a dip of 30-45 degrees to the east. Approximate present surface dimensions are 700 ft (213 m) by 1500 ft (457 m). The northern limit has been partially eroded by glacial action, leaving a large cirque filled with serpentine and argillaceous talus and debris. The host body consists of blocky, locally slickensided, light to dark green serpentine containing numerous veinlets of chrysotile asbestos (Fig. 3). In some areas of lesser serpentinization, bastites of serpentine pseudomorphous after rhombic orthopyroxene are evident. Magnetite is fairly abundant, occurring in microscopic veinlets and larger veins throughout the serpentine. Disseminated magnetite is conspicuously absent. Other minerals associated with the serpentine emplacement include: picrolite, magnesite, nemalite, brucite, tremolite and antigorite.

Locally, the Sylvester rocks comprise argillite, argillaceous quartzite, volcanics (greenstones) and graphitic schist. The contact between these rocks and the serpentine is conspicuously marked on the footwall by a zone of broken and incompetent argillite and graphitic schists. In general, the footwall argillites are grey-brown to black, fine grained and graphitic.

Wall-building characteristics are poor, and this has resulted in an inter-ramp footwall slope design at 37 degrees. On the hanging wall, the contact is much less conspicuous and consists of a zone of indurated argillite locally referred to as the "alteration zone". This zone is composed of a zoisite-quartz-tremolite hornfels with local irregular bodies of nephrite jade

Rotary drill rig overlooking the pit and crusher.

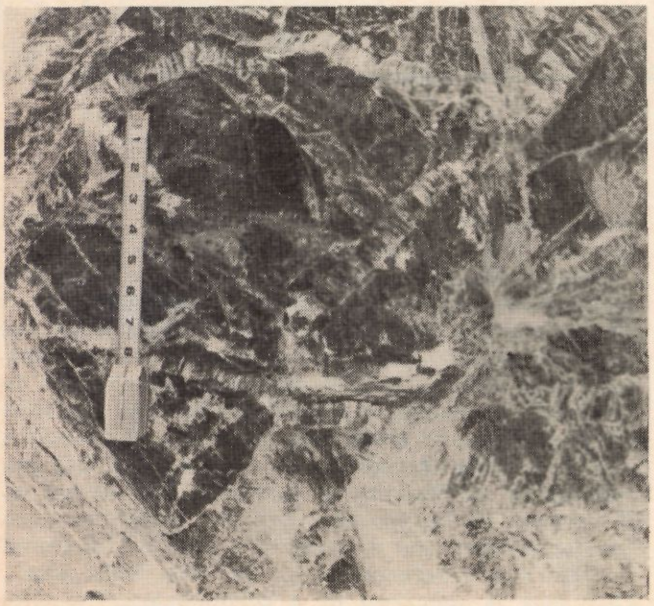


FIGURE 3 — Photograph of ore face — 5780 Bench.

and uvarovite garnet. As the hanging wall is traversed away from the contact, a gradual change from competent grey-brown argillite to grey-green volcanic and altered volcanic flows in the upper portion is apparent. This evidence of eugeosynclinal deposition has resulted in good "wall-forming" material.

STRUCTURAL

Detailed structure within the serpentine body can be considered complex. Numerous seemingly random joints, shear zones and vein systems occur throughout the orebody. These seem to be a result of tension release and cooling during emplacement of the Cassiar Batholith and the formation of serpentine. Faults and shear zones can be generalized into two groups. One group strikes east-west, with dips of ±20 degrees from vertical. This group is characterized by the "70° shear" at the southern end of the main orebody (Fig. 4). The other group strikes approximately north-south, with either a very steep western dip or an eastern dip near 45 degrees. This group is characterized by the "45° shear", which runs through the central portion of the main orebody and acts as the western boundary of the southern lobe. Faulting has been both pre-ore and post-ore, as evidenced by abrupt termination of fibre at the 70° shear and sheared fibre within the 45° shear. Joints and vein systems are considered as random in dip, although they seem to persist in strike between 90°and 180°. Some vein systems are also associated with the 45° shear as noted above. Two sets of joints and veins are sub-parallel to hanging-wall joint sets and will be discussed later.

Structures within the footwall argillites are dominated by the West Fault. This major fault strikes generally southeast, with a steep dip to the northeast. It occurs as a 50-ft (15.2-m) to 75-ft (22.9-m) wide zone of crushed argillite and graphitic material. Studies suggest a reverse fault and a relationship to the 70° shear within the serpentine. Bedding within the footwall argillite strikes near north-south, with dips of 30° to 50° to the east. The argillite-serpentine contact generally parellels this bedding, with some rolls to give a variable dip between 20° and 70°. Care must be taken with berm-face and haul-road design in this area. Joint sets are not well developed, but generally two dominant orientations are evident. One set of "cross joints" strike northeast, with steep dips to the northwest and southeast. These may be comparable to the tension fractures within the serpentine. The other set of joints essentially parallels the bedding strike, with steep dips to the west.

Structures within the hanging-wall argillites are generally simple, with three dominant sets of joints and related faults. Two joint sets strike slightly north of east, with steep dips to the north and south; the north-dipping set is the most continuous. Major faults throughout the hanging wall parallel, and occur along, these joint sets; they offset the argillite-serpentine contact in many places. A third joint set strikes slightly east of south, with steep dips to the west. The combination of three joint systems produces wedge failures that are generally less than one bench height. These systems sub-parallel two joint sets in the serpentine, as previously discussed. Evidence is therefore reinforced for much post-implacement fracturing and movement. Bedding within the hanging-wall argillites generally strikes 45 degrees, with fairly shallow (25°-40°) dips to the southeast. The strike of the argillite-serpentine contact varies from north to northeast, with dips slightly steeper than the argillite bedding (30°-45°). As the hanging wall is traversed, some

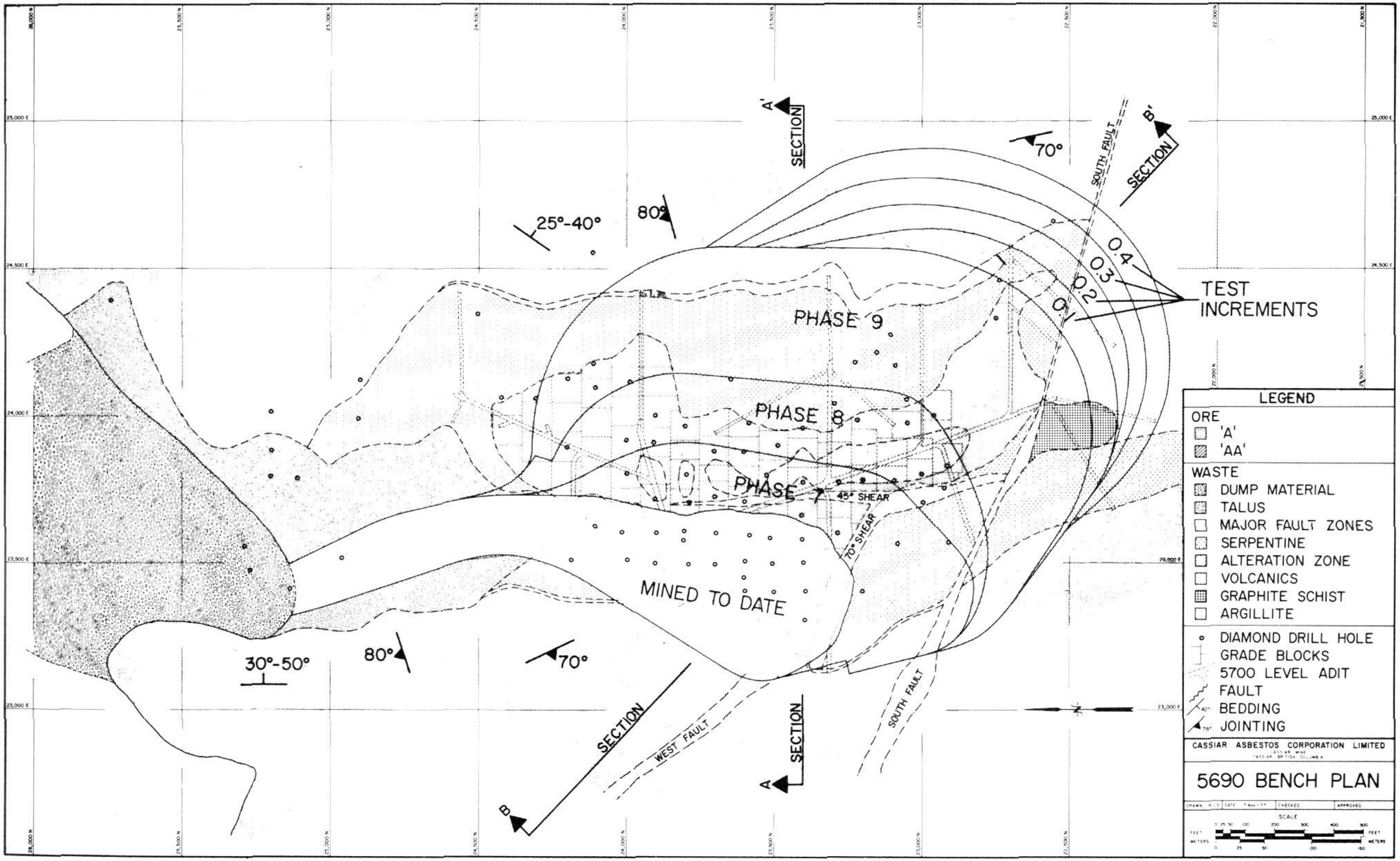


FIGURE 4 — Bench plan — 5690.

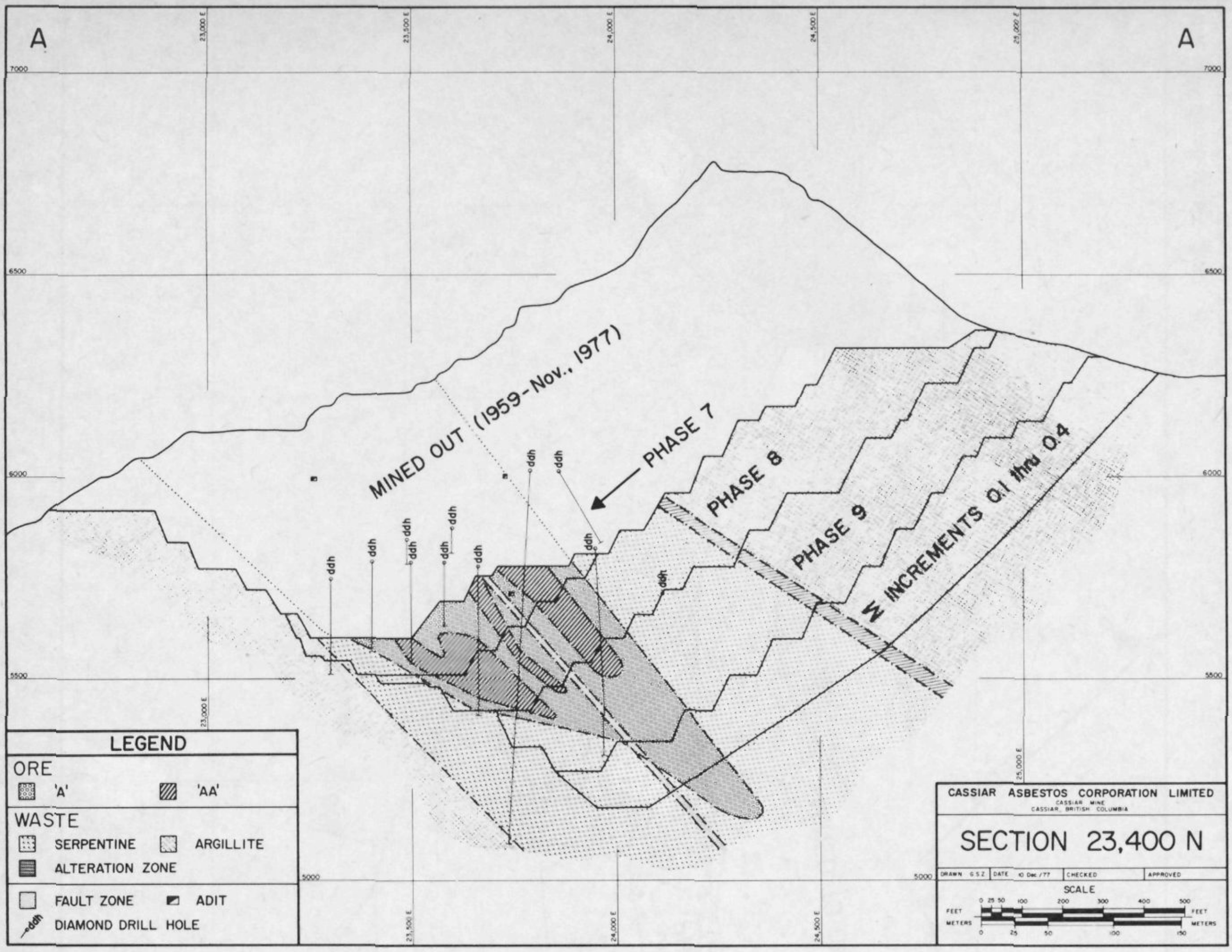


FIGURE 5 — Cross section: 23,400 N.

volcanic flows are encountered, until, in the "south peak" area of the mine, the wall rock is predominently volcanic. These rocks are generally structureless, with the general flow direction sub-parallel to the serpentine emplacement. A major fault system (striking east-west and essentially vertical), sub-parallel to the west fault and 70° shear, occurs in the southern portion of the pit. This system, known as the south fault, can be considered as an important area of future exploration due to its apparent southern termination of the ore and its genetic relationship with other major systems.

Geological mapping of all fresh faces is conducted throughout the pit to provide the geological base for pit planning and slope analysis.

Mine Development

SURFACE DIAMOND DRILLING

An annual program of HQ diamond drilling was initiated in 1971 within the Cassiar pit to provide the on-going grade and fibre characteristics needed for mine planning, and to explore the fibre zone currently beneath the open-pit design limits. Preliminary diamond drilling was attempted on the main orebody in 1951, but was largely unsuccessful due to poor core recovery. Diamond drilling was also carried out in 1961 and 1963 in the cirque to the north of the orebody to determine if any fibre-bearing structures followed the serpentine band. Pit waste dumps are now located in this area. Total footage drilled to date is nearly 39,000 ft (11,887 m) (Fig. 5).

Diamond drilling at Cassiar presents problems. Some are unique to serpentine, whereas many are common to the mining industry. Drilling within the orebody is attempted on 100-ft (30.5-m) grid centres where possible. The nature of the serpentine host rock, chrysotile fibre veinlets and shear zones make core recovery difficult. However, much success has been achieved since the initial 1951 drilling using HQ wireline equipment, with reduction to NQ where necessary. Deviation of planned hole direction in the serpentine is minimal.

Drilling from the hanging-wall argillites into the serpentine presents other problems. Due to the dominant hanging-wall bedding, all drill holes deviate to the north. This has resulted in holes being laid out to compensate for the expected deviation, which naturally precludes vertical holes. Down-the-hole methods in the argillites will be investigated in 1978 to obtain a vertical hole, with coring starting at the serpentine. Drilling in 1978 and subsequent years will be carried out in the pit to obtain further information for future mining and from the peak to determine the extent of the fibre zone at its southern end (Fig. 6). By the fall of 1980, a final decision must be made on the final design, which will influence both this and succeeding designs. Each design has a maximum waste yardage to be weighed against former ore. However, if by drilling, the ore is shown to be appreciably reduced (automatically increasing waste yardage), then the

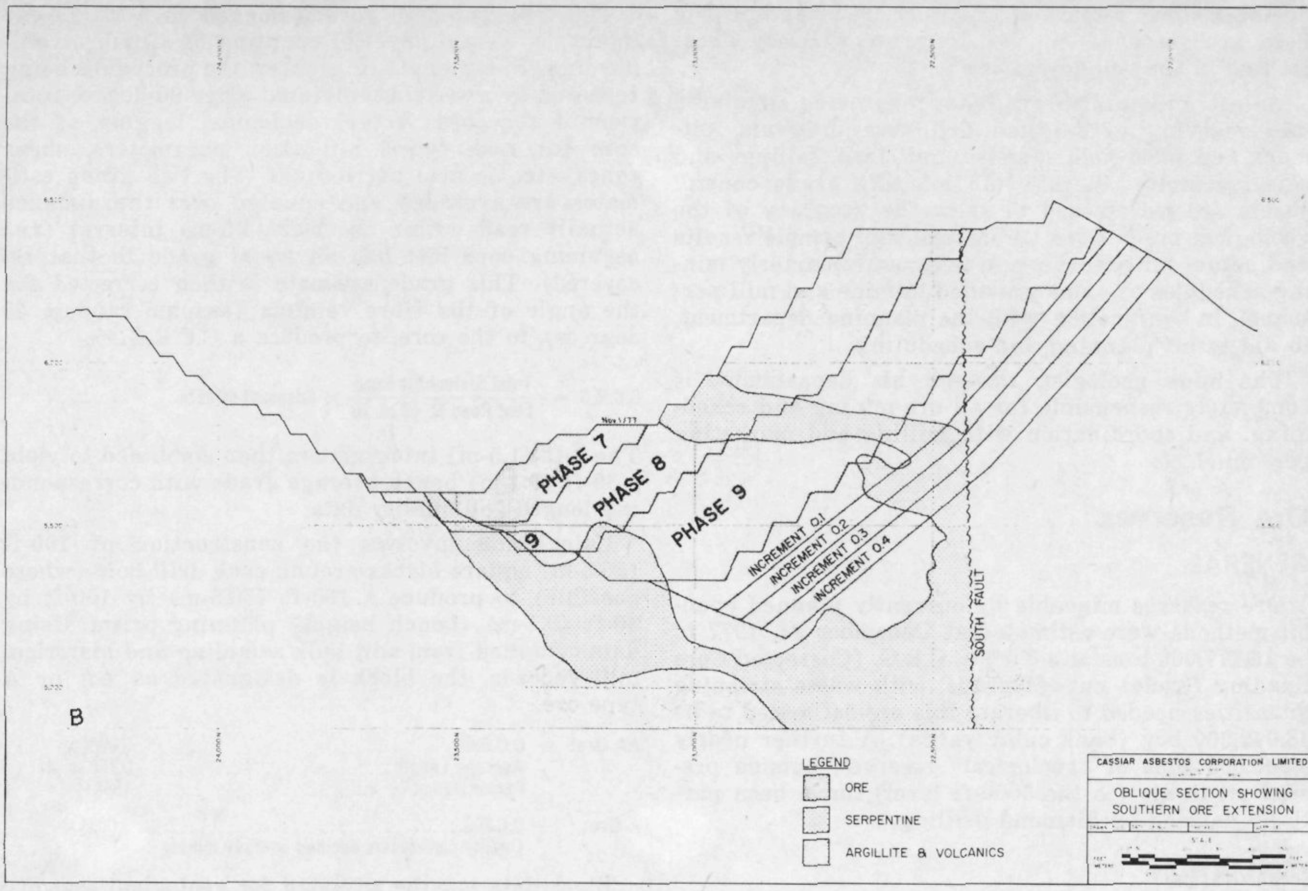


FIGURE 6 — Oblique section showing southern ore extension.

design must be revised to ensure a continuous ore supply. Comparable arguments apply to possible preceding designs and/or increment trials.

All drill holes are surveyed using the Sperry-Sun single-shot device. Where magnetic deflections are suspected, the Sperry-Sun Surwell Gyro survey is used for exact hole location.

No water under pressure has been encountered by diamond drilling, although groundwater is naturally found in the holes. This lack of head and slow recharge has indicated that there is no perched water table (or very local ones) within the pit. A plan of pit water flows and drill-hole water levels is maintained during the summer months to aid in groundwater studies. Diamond drill core is stored in a central plant-site facility for colour photography (Fig. 7), geological and fibre logging, and possible test milling.

ADIT EXPLORATION

Exploration adits were collared in 1952 at the 6000-ft (1829-m) elevation and in 1958 at the 5700-ft (1737-m) elevation. These adits traversed the orebody and provided valuable bulk sample data as to grade and dollar value, as well as providing geological mapping sites. Some limited underground diamond drilling also was done. Mining has since removed the 6000-ft (1829-m) adit and portions of the 5700-ft (1737-m) adit.

Grade Control

Daily grade control comprises face mapping and examination of all blast-hole drill cuttings within the pit. Blast plans are prepared showing the various

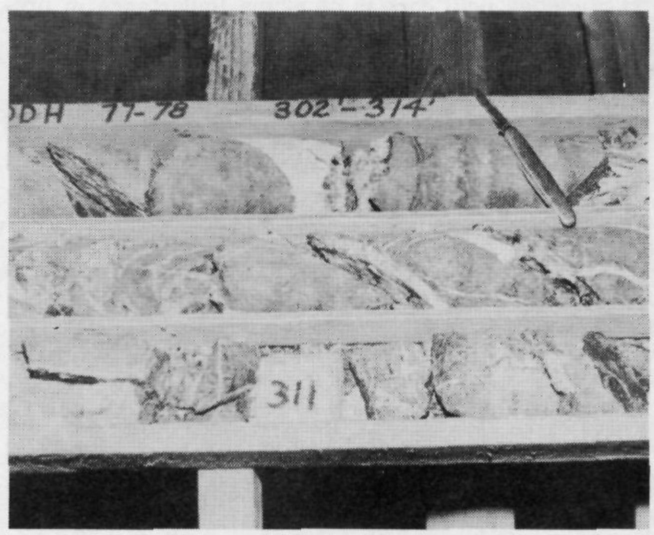


FIGURE 7 — Photograph of HQ diamond drill core.

lithologies present in each blast, as well as fibre grade and quality (dusty, wet) estimates of serpentine holes. As well, daily mucking sheets are maintained in the mine shifter's office showing the grade and quality of the ore available to be mined and the planned mining parameters. Plans of the mill's outside ore storage areas and dry rock storage are maintained in the mill shifter's office to provide information for blending and quality control. Daily examination of mine ore delivery, mill production, mill storage areas and laboratory assay results yields a comprehensive picture of the daily ore situation. The data are recorded on the daily ore delivery sheet located in the geology office.

A test mill is also available to provide simulated mill recovery of selected drill-core intervals, pit-muck and blast-hole samples, mill feed, tailings and reject samples. Weekly and monthly grade control sheets are maintained to show the accuracy of the geological predictions versus test mill sample results and actual mill recovery. Weekly and quarterly mining schedules are also provided to mine and mill personnel, in conjunction with the planning department, to aid in pit planning and scheduling.

The mine geologist, through his department, is completely responsible for all ore mining and scheduling, and coordination with milling and marketing personnel.

Ore Reserves

GENERAL

Ore reserves mineable by currently planned open-pit methods were estimated at December 31, 1977 to be 16,277,000 tons at a 3.0% C.C.R.G. (Corrected Core Reading Grade) cut-off grade, with waste stripping quantities needed to liberate this ore estimated to be 48,042,000 bcy (bank cubic yards). A further nearly 8,000,000 tons of "geological" reserves, outside present pit limits (to the 5000-ft level), have been partially defined by diamond drilling.

ESTIMATION

Ore reserve estimation is carried out using the data obtained from adit mapping, blast-hole logging, pit mapping and diamond drill information. Naturally, diamond drill core is of prime importance in projecting fibre zones.

The diamond drill core is logged in 5-ft (1.5-m) intervals, with a physical counting of all fibre veinlets of 1/16-in. length or greater, the procedure being repeated by a second individual after 90-degree rotation of the core. Actual geological logging of the core for rock types, alteration parameters, shear zones, etc., is also carried out. The two grade estimates are averaged and equated over the distance actually read within the 5-ft (1.5-m) interval (i.e., assuming core lost has an equal grade to that recovered). This grade estimate is then corrected for the angle of the fibre veinlets (assume random 45 degrees) to the core, to produce a C.C.R.G.

$$\text{C.C.R.G.} = \frac{\text{Total Sixteenths Read}}{\text{Feet Read} \times 12 \times 16} \times \text{Cosecant of } 45°$$

The 5-ft (1.5-m) intervals are then combined to yield a 30-ft (9.1-m) bench average grade with corresponding length and quality data.

Calculation involves the construction of 100-ft (30.5-m) square blocks around each drill hole (where possible) to produce a 100-ft (30.5-m) by 100-ft by 30-ft (9.1-m) (bench height) planning prism. Using data compiled from adit bulk sampling and historical mill records, the block is designated as AA or A type ore.

AA Ore:	— C.C.R.G.	8.00% +
	Average Length:	0.150 in. +
	Percentage of ½ in.	15% +
A Ore:	— C.C.R.G.:	3.00% +
	Quality parameters decided on daily mining	

Block data may be adjusted for geological reasons, such as contact areas, etc. The data for the block are then assigned block numbers and fed into the 9825A/9871A Hewlett Packard mini-computer. Print-outs from the computer yield a C.C.R.G. grade calculation, length distribution data, ore type (AA and A) and

ORE DEPOSIT MODEL FILES-PHASE 0 (PAH version July 1977) EXAMPLE ONLY

LEVEL	BLOCK	BLOCK AREA	CCRG PRCNT	AVG. LENGTH INCHES	1/2 + INCH PRCNT	3/8 INCH PRCNT	5/16 INCH PRCNT	1/4 INCH PRCNT	3/16 INCH PRCNT	1/8 INCH PRCNT	1/16 INCH PRCNT	AA ORE AREA
5690	1	1.00	2.90	0.104	3	5	8	2	14	31	7	0.00
	2	0.64	2.08	0.089	0	6	4	6	15	28	41	0.00
	3	1.00	2.09	0.100	2	3	6	8	12	30	39	0.00
	4	1.00	2.80	0.111	7	6	7	7	13	23	37	0.00
	5	1.07	3.51	0.121	12	9	9	6	15	17	32	0.00
	6	1.07	7.49	0.123	8	2	8	15	15	25	27	0.13
	7	0.50	8.39	0.136	9	6	7	16	16	24	22	0.00
	10	1.01	6.85	0.115	4	6	5	12	16	29	28	0.00
	11	0.96	8.07	0.132	7	8	5	14	16	27	23	0.65
	12	1.05	9.28	0.148	10	10	6	17	17	23	17	0.36
	13	0.44	3.99	0.122	5	8	4	13	17	27	26	0.00
	14	0.30	2.14	0.108	0	4	0	16	14	36	30	0.00
	15	0.92	3.60	0.109	5	8	3	12	14	29	29	0.00
	16	0.66	7.19	0.134	11	9	6	12	13	25	24	0.00
	17	1.00	10.77	0.158	17	11	9	13	12	21	17	0.48
	18	0.97	7.65	0.158	16	7	9	16	17	31	15	0.09
	19	0.56	5.81	0.164	20	12	7	16	10	20	15	0.00
	20	0.55	8.41	0.139	12	11	6	11	13	27	20	0.00
	21	0.88	6.72	0.134	7	9	8	14	15	26	21	0.00
	22	1.00	10.67	0.148	16	8	6	13	15	24	18	0.13
	23	0.53	9.78	0.142	13	7	7	13	15	25	20	0.38
	24	1.14	8.89	0.135	11	6	8	14	15	24	22	0.21
	25	0.48	5.54	0.127	7	6	6	16	15	27	23	0.28
	26	1.05	5.91	0.137	10	8	7	14	15	26	20	0.00
	27	1.02	10.72	0.138	7	11	5	13	16	32	16	0.01

FIGURE 8 — Computer print-out — 5690 Bench data.

tonnage, and estimated product results for each block, bench, phase or life of mine as desired.

Correlation

A tonnage reconciliation of the reserves is carried out monthly, as noted in the planning paper. The grade reconciliation is somewhat more difficult, with the calculated C.C.R.G. serving as a base for all other grade estimations. Mill recovery, and therefore Recoverable Mine Grade (R.M.G.), may change due to economics, mill circuitry, etc. The changing relationship between C.C.R.G. and R.M.G. is called the Grade Factor (G.F.) and is defined as:

$$\frac{\text{R.M.G. (historical)}}{\text{C.C.R.G. (historical)}} = \text{G.F. (historical)}$$

The **historical** grade factor can therefore be used in conjunction with the C.C.R.G. to produce a **predicted** R.M.G. for any period:

$$\text{C.C.R.G. (calculated)} \times \text{G.F. (historical)} = \text{R.M.G. (predicted)}$$

This would provide an estimate of the actual fibre recoverable from the reserves for the life of the mine, or for any mining period, assuming constant mill efficiency. The most critical value, however, needed for an economic evaluation is the **value** of that recoverable fibre (see planning paper). This fibre value (F.V.) is currently obtained from the Fibre Value Prediction Graph, which relates R.M.G. (%) to F.V. ($).

A more definitive system, based on correlation of C.C.R.G. length distribution with mill product distribution, is currently being evolved. This system yields an estimate of products contained in any phase or mining increment. Application of appropriate fibre prices and summation gives an estimate of rock value ($/ton) and total increment value ($).

References

(1) Gabrielse, H., G.S.C. Memoir 319, McDame Map Area, Cassiar District, B.C., 1963.
(2) Plumb, W. N., The Cassiar Asbestos Deposit, unpublished report, 1968.
(3) Piteau, Gadsby MacLeod, Ltd., Hangingwall and Footwall Slope Analysis, unpublished reports, 1974/76.
(4) Pincock, Allen and Holt, Inc., Planning Reviews, unpublished reports, 1975/77.
(5) Bowen, N. L, and Tuttle, O. F., The System MgO-SiO_2, Bulletin of the Geological Society of America, Vol. 60, 1949.
(6) Wyllie, P. J., Ultramafic and Related Rocks, J. Wiley and Sons, Inc., 1967.

"THE CASSIAR STORY"

Mine Planning

G. Scott Zimmer, Mine Engineer,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

Mine planning at Cassiar consists of an annual cycle of projection, control, evaluation and revision following completion of annual diamond drilling during the summer months. The inherent difficulty in evaluating fibre grade and distribution, combined with a high-wall phase or push-back mining system, requires continuous modification to designs and schedules based on a cumulative evaluation of equipment performance and prediction versus production. To expedite the planning sequence, a mini-computer system has been introduced which allows emphasis to be placed on optimizing designs, exempting personnel from manual calculations and tabulations.

Introduction

SINCE 1971, Cassiar has been carrying out an extensive diamond drilling program to further define the limits of the orebody in terms of tonnage and grade. Evaluation of drilling data indicated sufficient ore of suitable quality to warrant an ongoing expansion of operations and facilities. In order to provide a base for sound decision making, Cassiar's mine planning procedures have undergone a substantial change since 1973 with the review of old procedures and the introduction of a mini-computer.

Mine planning at Cassiar can be summarized as an annual cycle consisting of ore and waste projection, control, evaluation and revision. This entails the revision of the ore reserve figures after the evaluation of the annual drilling program, a sub-division of reserves into mining increments, engineering control and direction of mining operations, a reconciliation and evaluation of production with initial projections and, finally, a modification of reserves based on the production evaluation in preparation for use in conjunction with new drilling data to begin the annual cycle again.

G. Scott Zimmer was born in the U.S.A. and educated at Michigan Technological University from 1962 until 1966, majoring in geological engineering. He joined United Keno Hill Mines Limited in 1967 as senior exploration geologist. From 1969 until 1970, he was employed as a staff geologist in the consulting firm of R. G. Hilker Limited in Whitehorse. In 1970, he joined Conwest Exploration Company Limited as a staff geologist, progressing to resident geologist in charge of the Whitehorse office. He joined Cassiar Asbestos Corporation Limited in 1972 as mine geologist and became mine engineer in 1973. He is a Member of the CIM and a Fellow of the Geological Association of Canada.

Keywords: Cassiar Mine, Mine planning, Drilling, Sampling, Pit design, Reserve calculations, Scheduling, Computers.

There are three fundamental problems to consider in medium- to long-range planning. Two of the difficulties are fibre related; the third is due to a mining method commitment.

The first fibre-related problem is the difficulty of developing a rapid and consistent method of determining fibre content to an absolute or standard base. A new test mill system is currently being installed, but until the system is operational, all "assay" data used in mine planning are based on C.C.R.G. (Corrected Core Reading Grade). The relationship of C.C.R.G. with R.M.G. (Recoverable Mine Grade) is entirely dependent on mill efficiencies.

The second fibre-related difficulty is in assigning a rock value ($/ton) to future ore for evaluation purposes. Cassiar produces different fibre grades with values ranging from $320 per ton to $3,620 per ton. It is theoretically possible to produce nearly all of these products from any ore grade. Accordingly, derivation of rock value is again a function of mill efficiency and relies heavily on historical product distribution and market demands.

The third constraint is a commitment to a push-back or phase mining concept due to an ever-increasing hanging-wall depth [currently 1000 ft (304.8 m) vertical and increasing to at least 1600 ft (488 m) over the mine life], which imposes severe restrictions on ore availability and flexibility, requiring critical ore and waste scheduling, as well as constant evaluation to ensure ore continuity between phases.

Reserve Calculations and Pit Design

Mine planning procedures have been simplified as much as possible in order to produce a general flow sheet (Fig. 1). The foundation for planning consists of a master set of bench plans covering the entire orebody at mining bench intervals: 45 ft (13.7 m) in waste and 30 ft (9.1 m) in ore. These plans carry lithology, macro-structure, a fibre to no-fibre interpolated (to 3.0% C.C.R.G.) contact and high-grade zones within the "ore" limits. The fibre-bearing zone on each bench within the 3.0% C.C.R.G. contour is sub-divided into blocks, nominally 100 ft (30.5 m) by 100 ft (30.5 m), based on drill-hole intersections. Computer input of block areas, C.C.R.G. grade and C.C.R.G. length distribution for each bench produces a master grade-block model for the fibre-bearing zone which is independent of economic cut-off grade, mining limits and topography. Details of master model development are covered under "Computer Applications". The up-dated master model provides the base for economic evaluation and mining projection of the orebody.

Prior to establishing pit limits, a costing exercise is carried out to define the economic cut-off grade (currently established at 3.0% C.C.R.G.). This exercise can be complicated due to the probability that much of the low-grade material below a given cut-off grade would necessarily be mined as waste due to access considerations.

Also prior to establishing or revising pit limits, structural geology must be evaluated to derive an optimum slope design for various zones of the proposed pit. It should be noted that one of the advantages of phase mining is that intermediate phases may be treated as trial slopes so that design parameters can

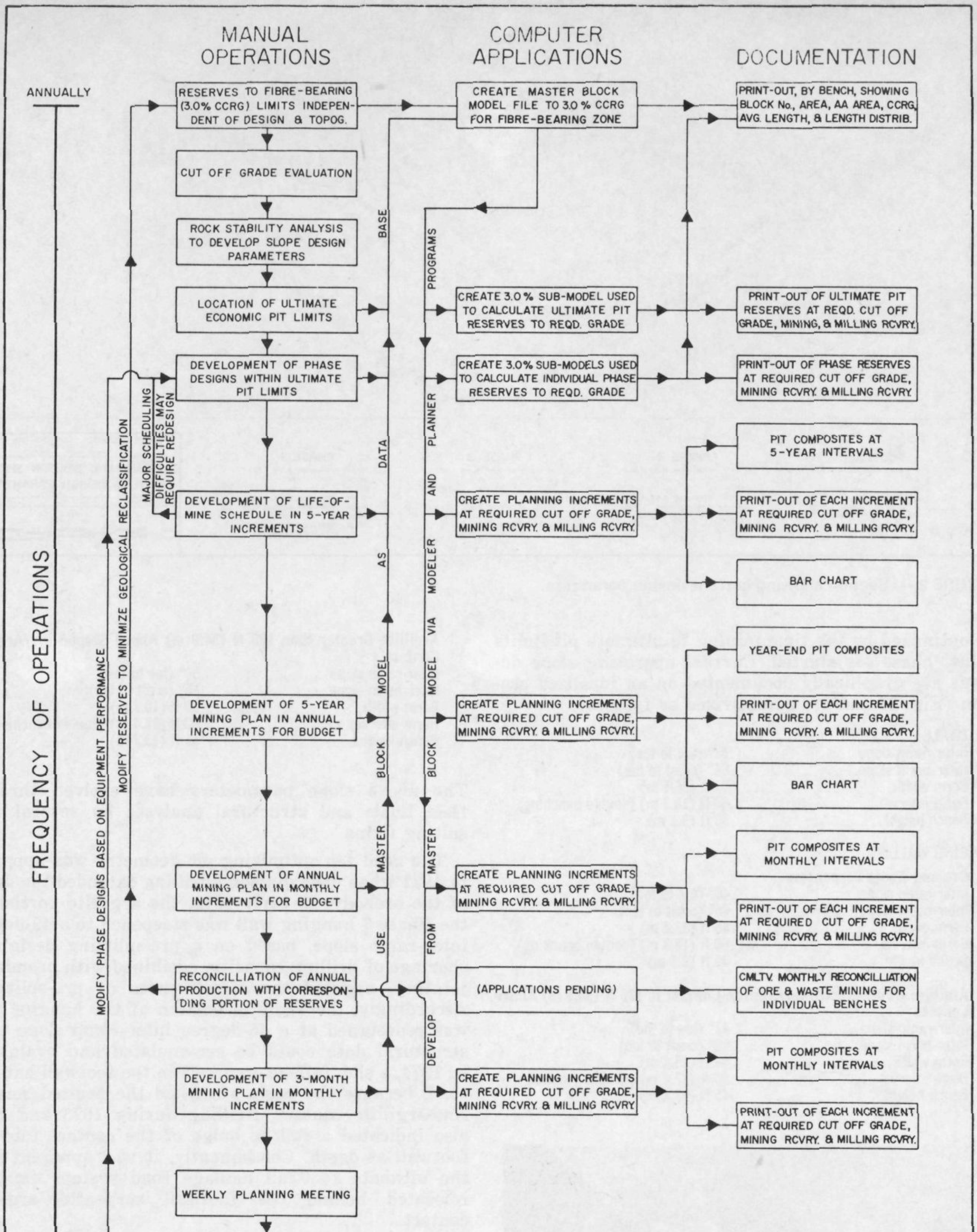


FIGURE 1 — Mine planning flowsheet at Cassiar.

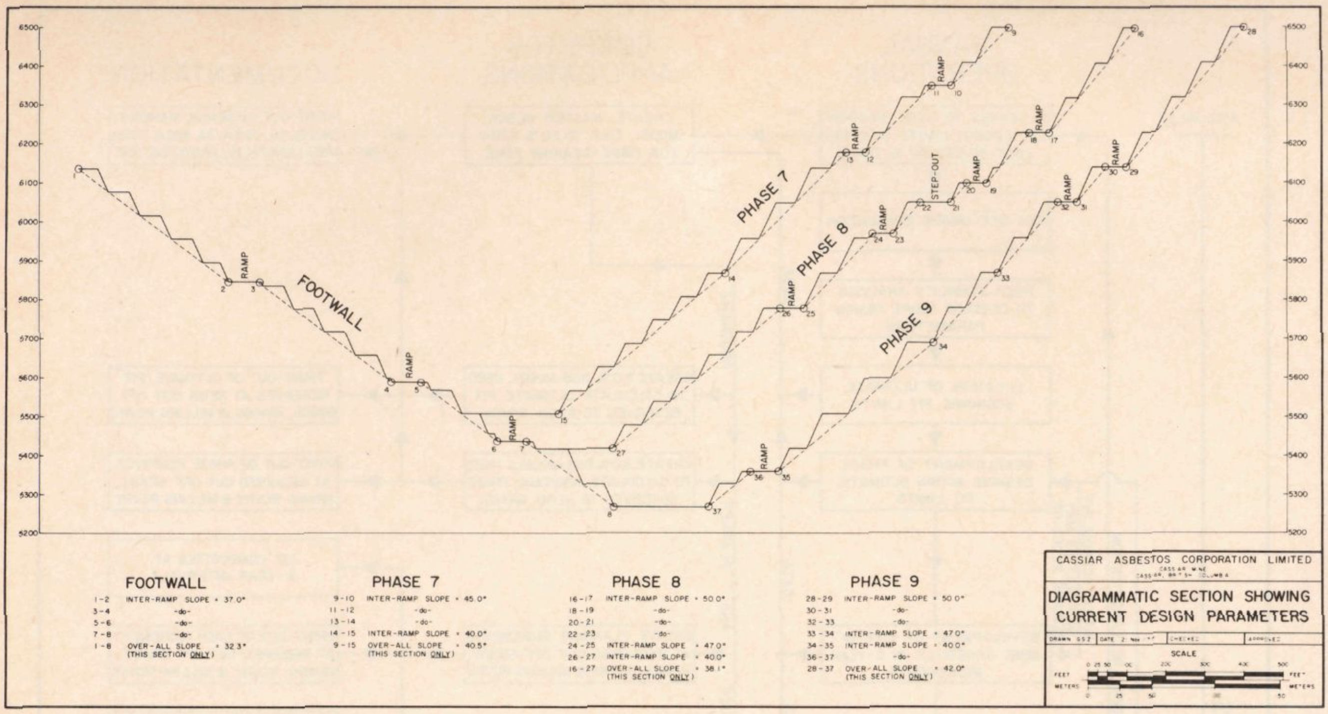


FIGURE 2 — Section showing current design paramters.

be optimized by the time mining to ultimate pit limits (final phase) is started. Current operating slope designs are graphically documented on an idealized section (Fig. 2) and are summarized as follows:

FOOTWALL:

Inter-ramp slope..................37° (toe to toe)
Inter-berm slope..................66° (crest to toe)
Berm width......................53 ft (16.1 m)
Berm interval.....................60 ft (18.3 m) (double benching)
Bench height.....................30 ft (9.1 m)

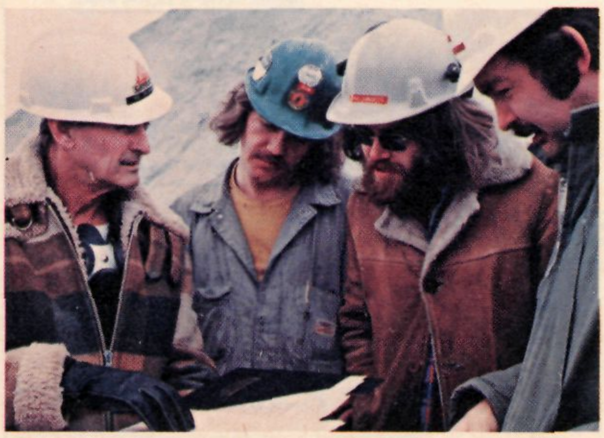
HANGING WALL:

Ore and Waste Serpentine

Inter-ramp slope..................40° (toe to toe)
Inter-berm slope..................66° (crest to toe)
Berm width......................45 ft (13.7 m)
Berm interval.....................60 ft (18.3 m) (double benching)
Bench height.....................30 ft (9.1 m)

Argillite from Serpentine/Argillite Contact to 180 ft (54.9 m) Above Contact

Inter-ramp slope..................47° (toe to toe)
Inter-berm slope..................66° (crest to toe)
Berm width......................44 ft (13.4 m)
Berm interval.....................90 ft (27.4 m) (double benching)
Bench height.....................45 ft (13.7 m)



Cassiar personnel map out strategy.

Argillite Greater than 180 ft (54.9 m) Above Serpentine/Argillite Contact

Inter-ramp slope..................50° (toe to toe)
Inter-berm slope..................66° (crest to toe)
Berm width......................35 ft (10.7 m)
Berm interval.....................90 ft (27.4 m) (double benching)
Bench height.....................45 ft (13.7 m)

The above slope parameters have evolved through field trials and structural analyses by several consulting firms.

The need for optimizing pit geometry was apparent in 1971 when deep diamond drilling extended the depth of the orebody. In early 1972, the argillite portion of the Phase 7 hanging wall was steepened to a 49-degree inter-ramp slope, based on a pre-splitting design. A shortage of drilling capacity, combined with prominent jointing, negated the effectiveness of pre-splitting. Accordingly, the argillite portion of the hanging wall was redesigned at a 45-degree inter-ramp slope until structural data could be accumulated and evaluated. In 1974, a slab failure occurred on the footwall haulage road, because the ramp straddled the faulted serpentine-argillite contact. Drilling during 1973 and 1974 also indicated a roll or bulge of the contact into the footwall at depth. Consequently, it was apparent that the ultimate footwall haulage road system must be relocated behind the footwall serpentine/argillite contact.

Initial evaluation indicated that a 47-degree inter-ramp was feasible and a contractor was brought in to start the excavation in the fall of 1974. However, mining revealed a more complex sructural system than anticipated and an abundance of highly unstable graphitic material was encountered which eventually required the redesign to the present inter-ramp slope angle at 37 degrees.

As shown, the inter-berm slope for all zones is 66 degrees. This figure was initiated as a function of shovel digging arc to provide a base for experimentation with the objective of evolving the steepest possible inter-berm slopes on a cost/benefit basis, thereby maximizing berm widths. Experimentation with offsetting intermediate drill lines and controlled blasting has indicated that inter-berm slopes to 72 degrees are possible in the upper hanging-wall argillites and that the less competent ore faces will ravel to approximately 53 degrees over the life of a phase.

Once slope parameters have been established, the approximate ultimate pit limits must be defined. The word "approximate" must be emphasized, as additional drilling is required prior to closer definition of the ultimate limits, which is carried out on a plan view by incremental analysis. Using the computer master model file, the ore and waste quantities in these increments are calculated and evaluated, using current costs and fibre prices to calculate a break-even stripping ratio.

With ultimate pit limits established, the volume within is allocated to phases. The phase concept was initiated early in the mine life to avoid the necessity of making a commitment to a massive waste stripping program based on insufficient ore information at depth. Phase mining, combined with topographical relief, imposes serious restrictions.

1. Peak benches are accessible only from the north end (see Figs. 3 and 4), which results in the necessity of maintaining internal ramp systems, which effectively flatten the slope and increase the stripping ratio.

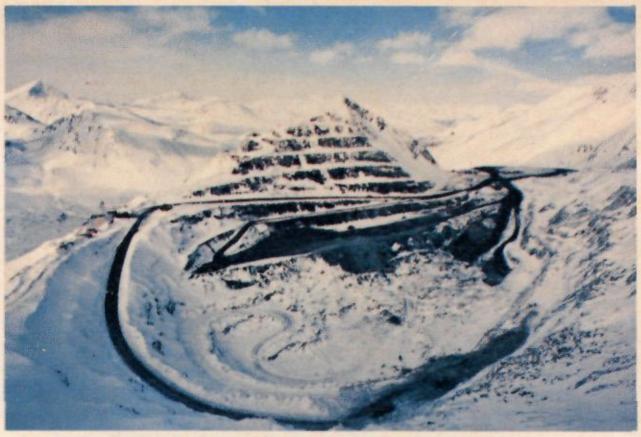
2. Timing, or correlation of ore and waste mining rates, is critical, as a serious shortfall in waste production or overproduction in ore could mean several months without ore between the phases.

3. The waste/ore ratio on the upper ore benches of all phases is high and these benches must be mined with the large shovels. Consequently, the ore on these benches usually cannot be by-passed due to confined working space and must be mined with the large equipment, which results in transportation and storage problems.

4. Bench access restrictions, combined with phase development, yield restricted working widths and result in inefficient equipment utilization. In 1971, when the largest shovel was a 4.5-cu.-yd 1400 P&H, bench widths were designed at 175 ft (53.4 m). Now, the use of the present equipment — 11-cu.-yd 1900 AL shovels — requires a bench width of 300 ft (91.4 m) (Fig. 5).

5. Although the previously described design parameters are functional from a stability standpoint, phase mining unavoidably results in a considerable quantity of waste material filling berms below the working benches. In order to protect ore mining operations below, this material must be removed on a routine basis. The narrow berms resulting from steeper inter-ramp slopes precludes safe effective access for cleaning purposes. Consequently, the internal hanging-wall ramp systems are utilized for clean-up, and step-outs (see 6050 berm, Fig. 4) are developed where the vertical interval between ramps becomes excessive. Stability analysis is currently being carried out to determine the feasibility of triple benching in selected areas to obtain additional berm width.

Current phase widths were developed by trial-and-error methods based on determining the life of an ore phase at a given mining rate and projecting current or anticipated annual equipment waste performance to the ore depletion time, which yields the maximum waste volume in the succeeding phase from the top down to continuous ore. Design trials are then carried out until this volume is reached. It is important to note that the high-wall situation, as indicated in individual phase designs (see Fig. 3), does not occur



The Cassiar open pit in mid winter.

until the final phase nears completion. The mining development sequence requires simultaneous mining of two or three phases (see Fig. 4), so that the overall working slope is effectively much flatter then any individual phase design. Individual phase designs are maintained as accounting and planning reference bases only. In addition to the slope parameters, the following road design criteria are employed:

Footwall Haulage:	
minimum width	70 ft (21.3 m)
grade	8%, with local sections to 10%
duration	life of mine
Hanging-Wall Permanent:	
minimum width	50 ft (15.2 m)
grade	8-10%
duration	life of phase
Hanging-Wall Temporary (muck ramp):	
minimum width	50 ft (15.2 m)
grade	10% for haulage, 12-15% for drilling and blasting access
duration	one or two benches

Scheduling

With completion of phase designs, a life-of-mine schedule, in five-year increments, is "mined" on paper. A mid-bench composite at the current pit configuration is used as an overlay on the ore reserve bench plans and the composite is "mined" bench by bench until the end of the five-year period. Using the master block model, input of bench and block areas will produce an ore-waste-fibre summary for the period at any required cut-off grade, ore recovery and mill recovery. The advantage of the system is that each bench must be examined for haulage access and ore continuity. Waste bottlenecks not apparent in the initial phase designs must be overcome or the phase in question must be redesigned.

Within the time framework indicated in the life-of-mine schedule, a detailed five-year mining plan for the immediate five-year period is developed similarly to the life-of-mine schedule, but at yearly increments. Pit composites at year-end intervals (Fig. 4), along with accompanying computer print-outs, are retained for reference. Following the initial measurements and calculations, a detailed bar graph showing monthly bench position is produced. The five-year plan is reviewed and up-dated every six months with a critical review of the relative positions of ore and peak waste benches.

A detailed yearly mining plan is similarly compiled each September for the following calendar year and is the foundation for the annual budget. Considerable

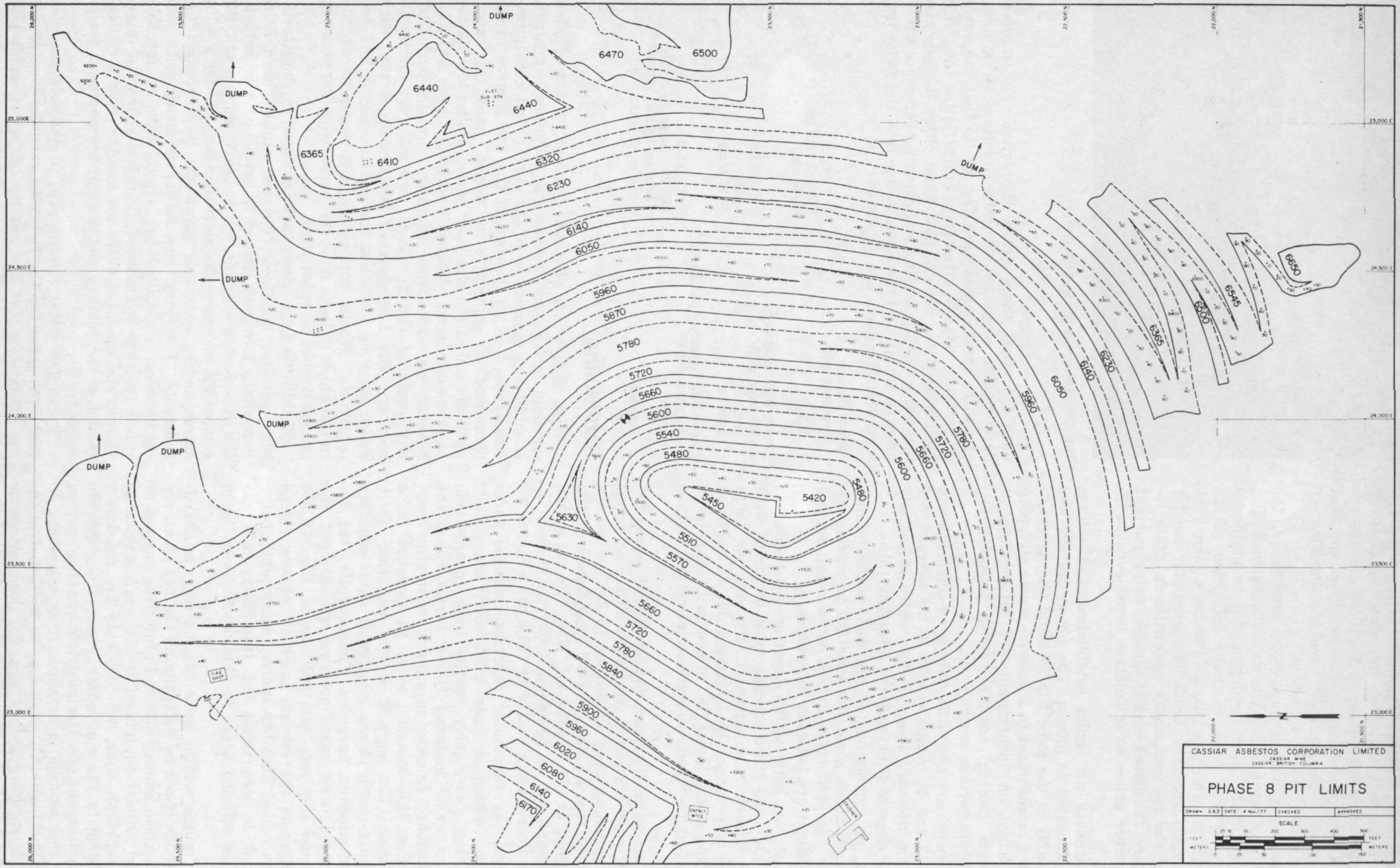


FIGURE 3 — Phase 8 pit limits.

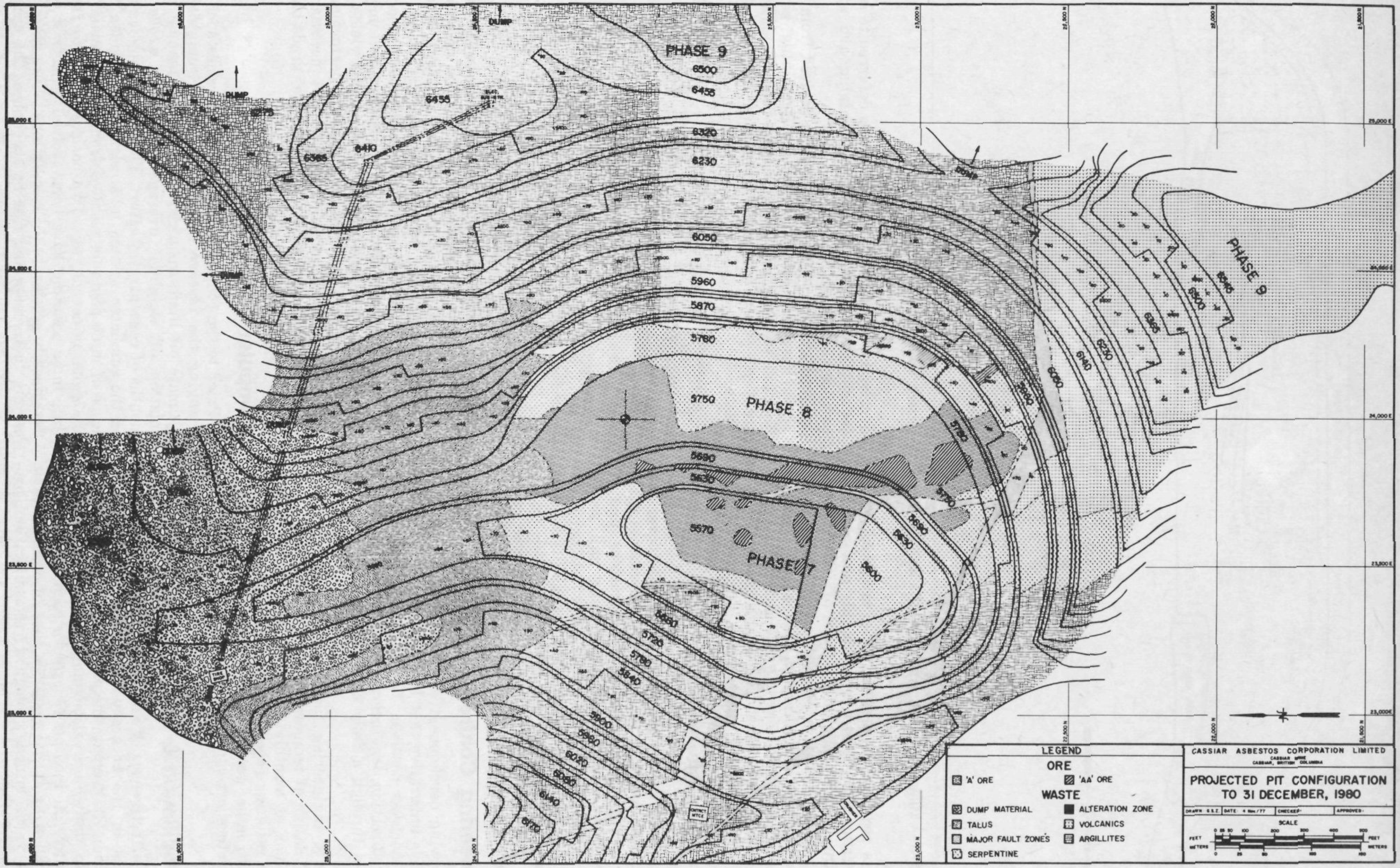


FIGURE 4 — Projected pit configuration to December 31, 1980.

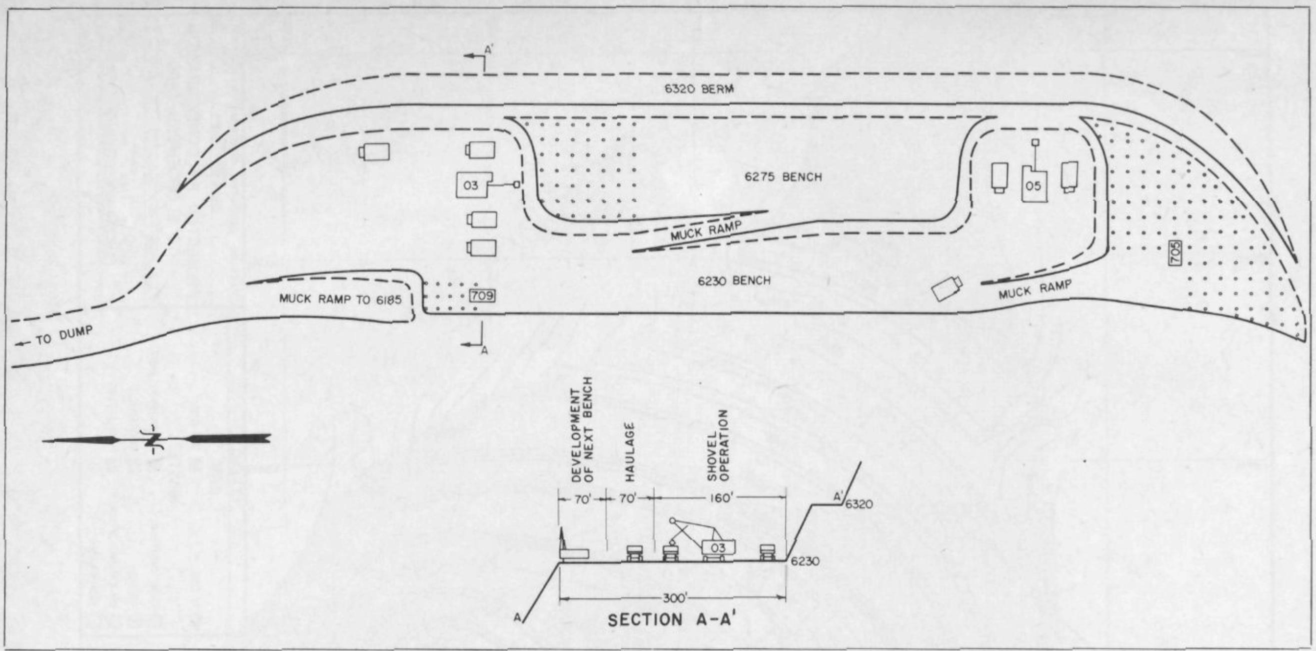


FIGURE 5 — Minimum bench width for efficient operation of current waste stripping equipment.

input is required from sales, mine, mill, geology and engineering in order to optimize this annual mining plan. Due to the preliminary status of fibre distribution projections, it is a prime function of mine planning to maintain the maximum degree of flexibility to compensate for deviations in grade and length of fibre from initial predictions. The annual mining plan is reviewed and revised quarterly. A bar chart for the yearly period is drawn up, with the added provision for comparing actual production with scheduled production.

Short-term mining plans are detailed to cover revisions to sales, mining and milling requirements within the general yearly mining plan parameters, and are reviewed at a monthly meeting of mine, mill, geology and engineering personnel. The presentation consists of a phase composite showing ore availability in terms of grade, ore tonnage and fibre. Proposed ore and waste areas for monthly periods are outlined on the composites and summarized in a computer printout.

Engineering Control

Following development of mining plans and schedules, it is imperative that these schedules are adhered to as much as possible. To supplement mine supervisory direction, engineering and survey control is supplied through a fully equipped field office at the mine site.

On-site engineering functions include blast layout design, design of temporary bench and haulage access, calculation of monthly mining quantities and completion of resulting reports, survey supervision, and liaison with mine, geology and engineering personnel to coordinate short-range objectives and survey requirements.

Reconciliation & Evaluation of Reserves

The effectiveness of an ongoing mine planning system lies in the accuracy of the initial prediction/simulation. To evaluate the prediction/production relationship, daily observations and measurements are incorporated into monthly reports for each ore and waste bench, showing initial reserves, tons or yards broken, tons or yards mined, geological reclassification, production reclassification, tons or yards deferred to subsequent phases, broken balance remaining and solid balance remaining.

Two algebraic equations have been derived from the basic 'Reserves –Mined = Balance Remaining' equation and are used monthly to ensure that calculations are kept within acceptable limits for both ore and waste:

1. $$\left(\begin{array}{l}\text{Unbroken remaining}\\ \text{at end prev. month}\end{array}\right) - \left(\begin{array}{l}\text{Broken}\\ \text{this month}\end{array}\right) + \left(\begin{array}{l}\text{Geol. recl.}\\ \text{this month}\end{array}\right) + \left(\begin{array}{l}\text{Deferred}\\ \text{this month}\end{array}\right) = \left(\begin{array}{l}\text{Unbroken}\\ \text{remaining}\\ \text{at end this month.}\end{array}\right).$$

2. $$\left(\begin{array}{l}\text{Broken bal. at}\\ \text{end prev. month}\end{array}\right) + \left(\begin{array}{l}\text{Broken}\\ \text{this month}\end{array}\right) + \left(\begin{array}{l}\text{Production}\\ \text{recl.}\\ \text{this month}\end{array}\right) - \left(\begin{array}{l}\text{Mined}\\ \text{this}\\ \text{month}\end{array}\right) = \left(\begin{array}{l}\text{Broken bal.}\\ \text{at end}\\ \text{this month}\end{array}\right)$$

Geological reclassification is the variance between the ore reserve ore/waste contact and the as-broken contact and is used to evaluate the accuracy of ore reserve predictions. Mining reclassification is an evaluation of the recovery of material broken as ore.

The requirement that all direct and indirect measurements must satisfy the above equations yields an appraisal of ore reserves and mining performance within acceptable limits. A similar, but necessarily more detailed, exercise to evaluate ore grade-block projections is pending.

Computer Applications

Prior to the computer application, exercises such as changing the reserve cut-off grade or re-appraising ultimate pit limits required the laborious manual calculation of weighted averages involving weeks to months of concentrated effort. The present application of the computer system to accelerate and widen the scope of mine planning resulted from a 1975 review of the mine by consultants. Two basic systems to facilitate mine planning have been evolved. The first was developed to accelerate the calculations of ore reserves, mining increments and mining schedules. The second system was developed to facilitate the accumulation and availability of production statistics for mine units.

The mine planning sequence is based on a master grade-block file for the fibre zone, independent of economic cut-off grade and mining limits. This is accomplished by manually establishing a fibre/no-fibre contact (3.0% C.C.R.G. used to give numerical base) on a series of bench plans covering the entire fibre zone. To provide continuity, all ore-containing benches are set at 30-ft (9.1-m) heights. The fibre-bearing zone within the 3.0% C.C.R.G. contour is manually subdivided into nominal 100-ft (30.5-m) by 100-ft (30.5-m) blocks, terminated by the 3.0% C.C.R.G. contour where applicable, and based on drilling intersections. In areas lacking drilling intersections, larger blocks are used or dummy blocks are set up at the geologist's discretion.

Using the computer Modeler program, the following entries are made for each block on a given bench:

1. Block number
2. Block area
3. High-grade fibre area
4. C.C.R.G.
5. Average fibre length
6. C.C.R.G. length distribution
   - 6.1. — %+½ in.
   - 6.2. — % 6/16 in.
   - 6.3. — % 5/16 in.
   - 6.4. — % 4/16 in.
   - 6.5. — % 3/16 in.
   - 6.6. — % 2/16 in.
   - 6.7. — % 1/16 in.

Following the above entries, opportunity is provided for averaging the grade and distribution data for the current block with the corresponding blocks on the benches immediately above and below, which effectively cuts high and low values. Completion of this exercise creates a master model file for the fibre-bearing zone, which is the foundation of the mine planning system.

After development or re-definition of ultimate pit limits, the Modeler program is again employed to build a sub-model of the ultimate pit, with the addition of total bench area to the block areas in order to determine the total volume (ore plus waste) within the ultimate pit to a 3.0% C.C.R.G. cut-off. With development of phase designs, identical exercises are carried out for each phase. Note that the master model is independent of topography, whereas the ultimate pit and phase sub-models are revised annually to include current topography. The objective of creating sub-models to the 3.0% C.C.R.G. (fibre/no-fibre) cut-off is to allow evaluation at any specified cut-off grade greater than 3.0% C.C.R.G. To obtain print-outs of reserves at various cut-off grades, the Reserve program is used in conjunction with the sub-models. Provision is also included in the Reserve program to vary mining and milling recoveries.

Once phase models have been created and reserves have been calculated, various mine scheduling trials are run using the Planner program in conjunction with the master model file. The scheduling trials are based on developing a time-dependent pit composite by sequentially mining out successive bench plans. Computer entries through the Planner program consist of:

1. Total bench area
2. Block number
3. Block area
4. Block high-grade fibre area

With the above entries, the Planner program extracts grade and distribution data from the master model file for storage and accumulation. The Planner program, like the Reserve program, includes provision for varying cut-off grade, mining recovery and milling recovery. The Planner program has an additional sequence which estimates mill product distribution and tonnages in any increment or plan. This sequence is the result of comparing C.C.R.G. length distribution in areas mined out with mill production during the same period. The results are a forced fit due to the small data base and the exercise is part of an ongoing data accumulation and evaluation program. Due to the preliminary nature of the study, product distributions for periods shorter than one year are suspect until proved otherwise. An Increment Adder program is used to combine individual plans into larger time periods. Monthly plans can be combined to form quarterly or annual increments, annual plans can be combined into five-year increments and five-year plans can be combined to produce a life-of-mine schedule.

Paralleling the computer-assisted planning system, a computer-based production statistics system was developed to upgrade and streamline the manual tabulation of cumulative daily mine production and equipment usage data. The system provides:

1. A base for evaluation and improvement of mine operating costs.
2. A productivity base or index for production units.
3. An index of daily, month-to-date and year-to-date mine production.
4. Equipment performance data used to modify mining plans and schedules.
5. Identification of problem areas during production short-falls.

Input for the system consists of daily entries of equipment number, bench, operating hours per account, ore and waste load count feet drilled, and maintenance hours for each production unit (drills, shovels, loaders, trucks, dozers and graders). Daily entries are made on a shift basis from operators' time cards, mine shift reports and equipment availability reports. Output, through several different programs and employing historical truck factors, consists of the following.

1. Daily Production: Short summary of daily and month-to-date production, with variance from budget.
2. Production Summary: Ore and waste removed from each bench on a daily, month-to-date and year-to-date basis.
3. Daily Shift Summary: On a bench basis, shows footage drilled, ore and waste removal, truck/shovel configurations and equipment operating hours. The summary is used to evaluate individual shift performance and to locate problem areas.
4. Equipment Summary: For each primary production unit, the summary shows operating hours, maintenance hours, standby hours, yards moved or feet drilled, yards or feet per operating hour, mechanical availability, use of availability and effective utilization. All the above are summarized on a daily, month-to-date and year-to-date basis.

A year-to-date 1977 equipment summary is included as Table 1.

For reference and clarification, the following should be noted:

1. Hours are based on an 8-hour shift, a 24-hour day and a 354-day year (365 days less 11 statutory holidays).
2. Maintenance hours include both scheduled and unscheduled repairs and servicing.
3. Standby hours are obtained by difference (8 hours less operating hours less maintenance hours).
4. Bcy (bank cu. yds) includes ore in yards.
5. Mechanical availability is defined as (operating hours

TABLE 1 — Year-to-Date Equipment Summary (November 30, 1977)

DRILLS:

Unit	Oper. Hours	Mtce. Hours	Stdby Hours	Feet Drld.	Feet/ Op. Hr.	Mech. Avail.	Use of Avail.	Eff. Util.
701 40R............	1180.5	4030.0	1917.3	47373	40	48.5	31.1	15.1
705 GD80............	3742.7	1552.2	2529.1	209115	56	80.2	59.7	47.8
707 RR105............	3037.4	3032.5	1754.1	156008	51	61.2	63.4	38.8
709 45R............	982.0	298.0	6544.0	36533	37	96.2	13.0	12.6
Total................	8942.6	8912.7	12744.7	449029	50	70.9	41.2	29.2

SHOVELS:

Unit	Oper. Hours	Mtce. Hours	Stdby Hours	Bcy	Bcy/ Op. Hr.	Mech. Avail.	Use of Avail.	Eff. Util.
2 PH1400 — 4.5 yd...	656.5	664.5	6503.0	110244	168	91.5	9.2	8.4
Total...............	656.5	664.5	6503.0	110244	168	91.5	9.2	8.4
3 PH1900AL — 11 yd..	4965.7	1324.2	1534.1	1962904	395	83.1	76.4	63.5
5 PH1900AL — 11 yd..	4976.5	1332.2	1515.3	1899424	382	83.0	76.6	63.6
Total................	9942.5	2656.4	3049.4	3862328	388	83.0	76.5	63.5

LOADERS:

Unit	Oper. Hours	Mtce. Hours	Stdby Hours	Bch	Bcy/ Op. Hr.	Mech. Avail.	Use of Avail.	Eff. Util.
480 H-400-C — 10 yd..	2410.3	2874.5	1843.2	348869	145	63.3	48.7	30.8
481 Cat 992 — 10 yd..	4854.7	2354.8	614.5	954819	197	69.9	88.8	62.0
483 Cat 992 — 10 yd...	468.5	89.5	138.0	108585	232	98.9	6.1	6.0
Total................	7733.5	5318.8	2595.7	1412273	183	66.0	74.9	49.4

TRUCKS:

Unit	Oper. Hours	Mtce. Hours	Stdby Hours	Ore Tons	Waste Bcy	Total Bcy	Bcy/ Op. Hr.	Mech. Avail.	Use of Avail.	Eff. Util.
43 Wabco 50T.........	561.0	1562.5	5700.5	12162	17433	22408	40	80.0	9.0	7.2
44 Wabco 50T.........	1138.0	1122.5	5563.5	66495	31518	58721	52	85.7	17.0	14.5
45 Wabco 50T.........	2122.5	925.0	4776.5	143471	64202	122895	58	88.2	30.8	27.1
46 Wabco 50T.........	3389.9	1073.5	3360.6	240946	115710	214279	63	86.3	50.2	43.3
47 Wabco 50T.........	3530.4	1438.0	2855.6	301602	98481	221864	63	81.6	55.3	45.1
Total................	10741.8	6121.5	22256.7	764676	327344	640166	60	84.4	32.6	27.5
50 Wabco 75T.........	2066.5	1813.5	3944.0	47982	198400	216575	105	76.8	34.4	26.4
51 Wabco 75T.........	2743.7	975.0	4105.3	126720	251425	299425	109	87.5	40.1	35.1
52 Wabco 75T.........	2284.5	1901.0	3638.5	97218	195250	232075	102	75.7	38.6	29.2
53 Wabco 75T.........	1625.5	3135.0	3063.5	59994	135475	158200	97	59.9	34.7	20.8
54 Wabco 75T.........	1969.2	1564.0	4290.8	36564	183525	197375	100	80.0	31.5	25.2
55 Wabco 75T.........	2160.4	1705.0	3958.6	35046	202175	215450	100	78.2	35.2	27.6
56 Euclid R85.........	3736.1	1727.5	2360.4	660	438125	438375	117	77.9	61.3	47.8
57 Euclid R85.........	3437.0	2139.5	2247.5	2303	419625	420498	122	72.7	60.5	43.9
58 Euclid R85.........	3158.5	2820.5	1845.0	6732	366050	368600	117	64.0	63.1	40.4
59 Euclid R85.........	3289.0	2784.5	1750.5	1056	359400	359800	109	64.4	65.3	42.0
60 Euclid R85.........	3505.7	1715.0	2603.3	1914	414575	415300	118	78.1	57.4	44.8
61 Euclid R85.........	2980.5	2951.0	1892.5	858	381800	382125	128	62.3	61.2	38.1
62 Euclid R85.........		2205.0	2205.0	3300	424750	426000	120	73.5	61.6	45.3
63 Euclid R85.........	3815.0	1996.0	2013.0	3564	426125	427475	112	74.5	65.5	48.8
Total................	40313.6	29304.5	39917.9	423911	4396700	4557273	113	73.2	50.2	36.8

+ standby hours)/(total hours).
6. Use of availability is defined as (operating hours)/(operating hours + standby hours).
7. Effective utilization is defined as (operating hours)/(total hours).

The production statistics system, as currently developed, will be expanded and incorporated into a larger corporate computer system expected to be on line in 1978. In addition to the two basic computer systems described, development is continuing on applications to reserve reconciliation, grade-block reconciliation, C.C.R.G. length distribution correlation with mill products, survey plotting, graphical plotting and rock mechanics.

Summary

The computer-assisted mine planning system developed at Cassiar is the foundation for a sound engineering supplement to mine production. The system is by no means complete and additional development is required, particularly in the area of fibre length distribution.

References

(1) Slope Stability Analysis and Design of the Footwall Slope prepared for Cassiar Asbestos Corporation Limited by Piteau Gadsby MacLeod Limited, 1974.
(2) Slope Stability Analysis and Design of the Hanging-Wall Slope prepared for Cassiar Asbestos Corporation Limited by Piteau Gadsby MacLeod Limited, 1975.
(3) Mine Planning System using HP 9825A Programmable Calculating prepared for Cassiar Asbestos Corporation Limited by Pincock Allen and Holt, Inc., 1976.
(4) Long-Range Mining Plan with Estimate of Fibre Production prepared for Cassiar Asbestos Corporation Limited by Pincock Allen and Holt, Inc., 1977.
(5) Daily Mine Production and Equipment Report System prepared for Cassiar Asbestos Corporation Limited by Pincock Allen and Holt, Inc., 1977.

Mining

Gillyeard J. Leathley, General Mine Superintendent,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

The Cassiar Asbestos mine is situated on McDame Mountain in northern B.C. at a mean elevation of 6000 ft (1830 m), some 6 miles (9.6 km) from the townsite/mill complex. Asbestos ore production averages 1,200,000 tons at 9% R.M.G. (Recoverable Mine Grade) and waste production is 6,000,000 cu. yd annually.

Waste is loaded by P&H 1900 AL 11-cu.-yd shovels and transported by 75-ton and 85-ton end-dump trucks to the waste dumps. The ore is loaded by a 992 Caterpillar 10-cu-yd front-end loader into 50-ton and 75-ton trucks, which transport it to the crushing plant located at the mine. The ore is transported by a 15,370-ft (4685-m) tramline to the mill complex.

Introduction

THE CASSIAR OPEN-PIT MINE is situated at a mean elevation of 6000 ft (1830 m) on McDame Mountain, 6 road miles (9.6 km) from the main mill/plant complex. The access road rises some 2350 ft (716 m) from the plantsite to the mine by means of a series of tight switchbacks. Constant grading is required with a Caterpillar 16G grader to ensure that drainage ditches are kept clear to deal with a heavy spring snow melt and an annual summer rainfall of 12 in. (300 mm). During winter, the grader is equipped with a front snow blade, as well as a rear-mounted hydraulically operated snow wing, and the use of tire chains is standard to deal with 13 ft (400 cm) of snowfall annually (Fig. 1).

Personnel are transported to the mine by a 45-seater manhaul bus, supplemented by 4x4 pickups equipped with 14-seat passenger carry cabs. The pickups allow operators to change over at the equipment location, reducing equipment downtime.

The 1977 production was 1,321,065 tons of ore and 6,874,201 cu. yds of waste rock, which meant a stripping ratio of 1 ton of ore to 5.20 cu. yds of waste. The five-year mining plan, which is updated and revised as sales commitments and mining conditions dictate, envisages approximately the same magnitude of ore for the next 14 years, with waste production continuing at 6,000,000 cu. yds until Phase 9 waste is excavated by the end of 1985.

An increase in waste-rock production, dictated by the introduction of additional milling facilities requiring an increase in asbestos ore, as well as an increasing stripping ratio, led to the introduction of a 1900 AL P&H 11-cu.-yd shovel in October, 1974. An additional 1900 AL P&H 11-cu.-yd shovel was introduced in October, 1975. This necessitated pit electrification, which was completed in October, 1974, and the introduction of a fleet of 85-ton Euclid trucks.

The following figures give some indication of the progress of the mine since production commenced in 1954:

	Ore Tons	Waste Cubic Yards
1954 — 1960	2,266,761	2,977,458
1960 — 1970	7,907,277	17,217,172
1970 — 1975	5,376,551	13,786,669
1975 — December 1977	2,244,616	13,600,575
TOTAL REMOVED	17,795,205	47,581,874
REMAINING TO END OF PLANNED PIT LIFE:	16,277,000	48,042,000

The phase mining plan dictates meticulous planning of ore and waste scheduling to ensure a continuous supply of ore to the mill, especially during the interphase period. *Phase mining configurations are shown* in the paper dealing with the planning operation.

The assistant mine superintendent and the general foreman - mine, work very closely with the planning and geology departments to ensure that long-range plans and monthly schedules are adhered to. It is the responsibility of this group to ensure that adequate ore supplies are available to blend low- and high-grade areas or supply ore from a high-grade, long-fibre area if a spinning-grade fibre run is required in the mill. A planning meeting to relay this information to the mine shift foreman is held every two weeks to allow discussion and examination of the developed plans. Mill requirements regarding the type and/or blending of ore required are marked daily on a location plan in the mine shift foreman's office at the mine. If the mine foreman must deviate from this plan, he is required to inform the mill shift foreman immediately to allow the mill to relocate the ore in the dry rock storage to a more suitable area. No designated ore can be discarded unless agreed to by the geology department. A monthly reconciliation of ore predicted by the geologist, and typed by him from production drilling and actual ore mined, is carried out to ensure that a minimum amount of ore is discarded. Unfor-

Gillyeard J. Leathley was born and educated in Scotland. He graduated from Glasgow's Royal College of Science and Technology, with a Higher National Certificate in mining and mine surveying. He joined the National Coal Board (Scotland) in 1952 and worked as a mine surveyor and work study engineer in underground coal mines. In 1962, he joined Anglo American Corporation, Zambia, as a learner mine official and was later assistant manager, Open Pit Division of AMCO, responsible for open-pit coal and copper mines. Returning to Scotland in 1967, he joined a rock excavation company in the United Kingdom and served as a director of the company until 1971. At that time, he became mine technical superintendent of Guyana Bauxite in South America, progressing to general mine superintendent in 1972. In this position, he was responsible for the operation of six nationally owned bauxite mines producing 5,000,000 tons of bauxite annually, with the removal of 30,000,000 cu. yds of overburden. Experience in the operation of bucketwheel excavators, walking draglines and rail mining was gained at this time.

In 1975, Mr. Leathley joined Cassiar Asbestos Corporation Limited as chief engineer. He is now general mine superintendent responsible for the complete operation of *the engineering and mines departments. He is a member* of the CIM.

Keywords: Cassiar mine, Asbestos, Open-pit mining, Drilling, Blasting, Lauding, Haulage, Mining methods, Dewatering, Crushing.

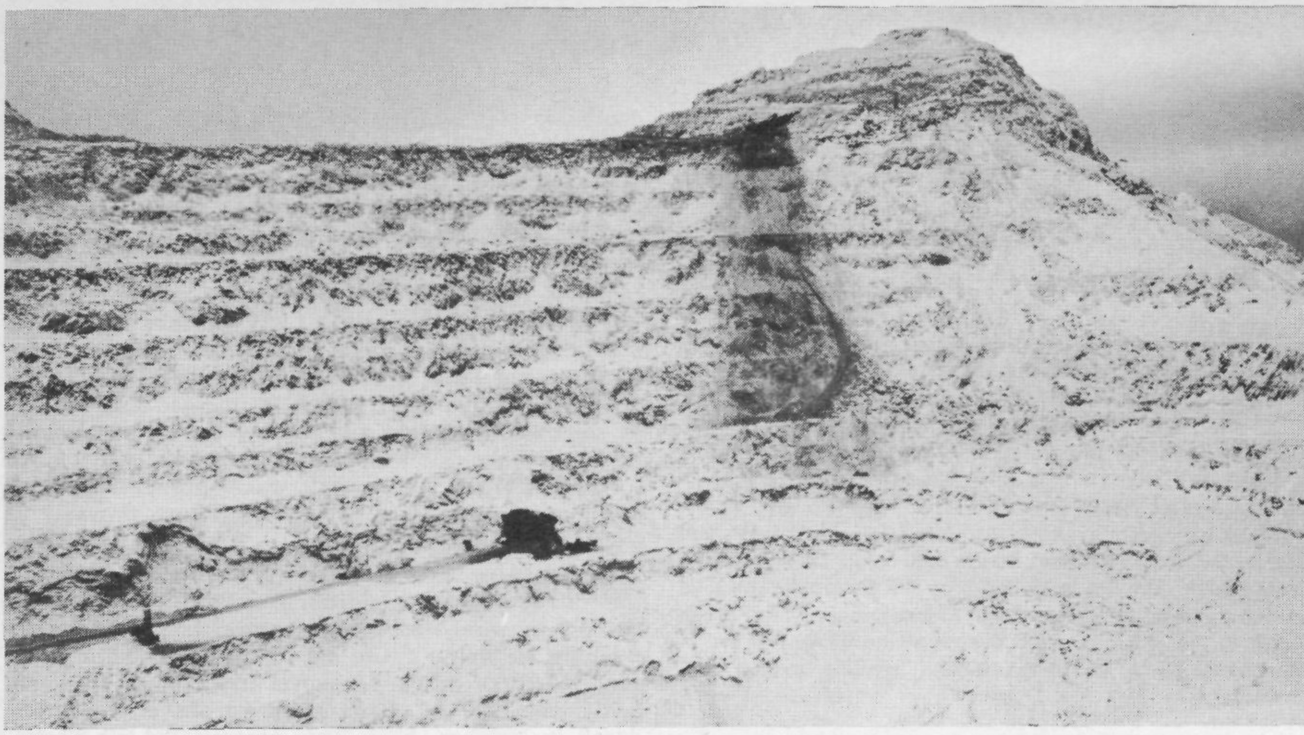


FIGURE 1 — Winter-time operations at the Cassiar pit.

tunately, the pit configuration allows very limited stockpiling of low-grade ore, with the result that maximum in-pit blending must be utilized.

Because a hot change is practiced on the equipment at the mine and the operators report on shift at the mine dry, situated at the mill/plant complex, the on-going-offcoming shift foremen have only a few minutes to discuss problem areas. To alleviate this as much as possible, a detailed log is written out by the general foreman, prior to the afternoon shift reporting. The general foreman normally also verbally instructs the ongoing foreman. The offgoing foreman will telephone the ongoing foreman at the mine dry, detailing the situation regarding equipment and problems at the end of his shift.

The pickups supplied to supervisory personnel are equipped with a two-way radio system, enabling them to maintain contact with all loading equipment, drills, the crusher, the pit maintenance shop and the mill/plant complex.

A shift report, recording equipment hours, truck loads and equipment availability, is filled in by each mine shift foreman. An equipment availability report compiled by the maintenance department at the mine is countersigned by the mine shift foreman. Equipment operators' time cards, the shift report and the availability report are scrutinized daily by the mine clerk to ensure that equipment operating hours and loads are compatible from the various information sources. These are then entered into a Hewlett Packard desk computer. This information is invaluable for compiling availability and performance statistics on equipment for future planning and can be quickly recalled at any time.

Total mine labour force is 160 persons, including mine production supervision. At the present time, the waste crews work six days on, followed by two rest days, the ore crew works five days on, with two rest days. Supervisors rotate with their own crews of approximately 30 persons. The drill and blast foreman and the crushing plant foreman work dayshift, Monday to Friday.

High labour turnover necessitates the employment of two training foremen to ensure that an adequate supply of trained operators are available for production requirements.

Drilling

Drilling was originally carried out using Ingersoll-Rand Drill Masters, with 5-in. or 6½-in. down-the-hole hammers, on ore and waste benches. The major waste drilling was carried out using a Bucyrus-Erie 30R rotary drill with 6¾-in. bits. A Bucyrus-Erie 40R rotary drill, using 9-in. bits, was introduced in 1964. In 1970, the Bucyrus-Erie 30R was retired and an additional Bucyrus-Erie 40R was introduced. Both BE 40R drills were diesel-electric powered.

A new 24-kv powerline to the pit was installed in October, 1974, and the following drills are now used for primary waste drilling:

1 Gardner-Denver GD80 electirc	54-ft single-pass drill steel, 9-7/8-in hole
1 Bucyrus-Erie 45R electric	32½-ft drill steel, 9-7/8-in. hole

The single-pass GD80 drill is equipped with a GD SJWG 1750-cfm compressor, derated to 1400 cfm at 55 psi; the BE 45R has an Allis-Chalmers 17-L 1310-cfm compressor at 40 psi. Because both units are equipped with oversized compressors, no difficulty has been encountered in clearing drill cuttings in 55-ft-deep holes, even in fractured rock. The drill pipe used on the GD80 is 8⅝-in. diameter, single-pass, 54-ft (16.5-m) long with 5½-in. A.P.I. thread ,driven from the drill deck, using hydraulic rotary drive, employing two wedges which fit into machined slots on the drill pipe. The BE 45R has standard 8⅝-in.-diameter pipe, using 6-in. B.E.C.O. thread. A swivel-mount shock sub is used on the BE 45R to reduce shock loading on the mast and rotation drive assembly. The roller stabilizers used on all drills are the tungsten-carbide studded roller type and are not normally repaired. Hole depth recorders are provided on both electric units.

Records of drill bit life are kept to ensure that adequate life is being obtained from the tricone rotary tungsten-carbide bits. The grade of bit used is changed, depending on the rock formation being drilled (serpentine, footwall argillites, hanging-wall argillites/jade and volcanic formations). Bearing failure is the major reason for a bit being discarded — not loss of gauge or button failure — as the material being drilled is not highly abrasive. Some attempts have been made to replace bearings, without a great deal of success.

Dust suppression is available at all times of the year for both operator and equipment intake air. The BE 40R drills were originally equipped with Rotoclone dust collectors, but these were removed and a water dust suppression system installed.

The GD 80 drill was introduced in February, 1975. It has a fully pressurized operator's cab and machine house, and was originally equipped with a pulse air-type bag collector. High air bailing velocity exiting from the drill-hole collar, coupled with drill holes that contained water, combined to render it ineffective due to dust particles freezing and blinding the filter bags in the baghouse. Water obtained from the hot water waste supply at the powerplant is trucked to the drills at the mine and kept heated in 600-gallon tanks fitted onto the drills. Water lines are insulated and wrapped with electric heat tape, and a blow-down valve is fitted on each drill allowing the operator to purge the water lines at shift change or prior to switching off power to the drills.

Drilling patterns are varied depending on the rock competency, the depth being drilled and the fragmentation required for loading. All drill patterns are laid out in accordance with the drill and blast foreman's instructions, with the assistance of the survey crew, who report to the short-term planning engineer.

A first line of drill holes, adjacent to the back line of the previous blast, is laid out by instrument. This is normally 20 ft (6.09 m) to 25 ft (7.6 m) from the previous blast holes, depending on the bench height and backbreak. The drill line delineating the back line of the new blast is also laid out by instrument and the intermediate holes are tape measured and marked for drilling with the appropriate total depth of hole required, including subgrade drilling.

Various patterns have been tried, including square and staggered patterns, with different burden and spacing ratios. However, a 22-ft (6.7-m) by 22-ft square pattern appears to give consistent fragmentation results when drilling 45-ft (13.7-m) deep benches. This is reduced to 18 ft (5.5 m) by 18 ft on a 30-ft (9.1 m) deep bench and further reduced if drilling ramps. The drilling of a subgrade of 5 ft (1.5 m) is practised in waste rock on the hanging wall; in the softer argillite/serpentine rocks, a 3-ft (0.91-m) subgrade is drilled. No sub-grade is normally required in the asbestos ore. In winter, the drill holes are covered with wooden discs or plastic bags filled with snow to prevent snow from entering the drill hole and causing a loss in depth. Redrilling of holes represents about 10% of total drilling and is due to broken ground, collars of holes collapsing or ingress of water.

Secondary drilling is carried out by using a Gardner-Denver Airtrac equipped with a PR123 percussion/rotary hammer, drilling a 2½-in. hole and blasting with 2-in. by 16-in. Powerfrac. Secondary drilling on grade level is not significant except in the higher pit elevations, where the volcanic and jade intrusions are extremely difficult to blast effectively. Large oversized blocks of ore and waste are drilled with a Schram drill mounted on a rubber-tired tractor, utilizing 1¼-in. chisel-pointed drill steel.

TABLE 1 — Equipment Details

Unit	Manufacturer	Details	Productivity
Drills			
1	Gardner-Denver	GD80 — 9⅞ in. hole	45 ft/hr
1	Robbins	RR10S — 9 in. hole	40 ft/hr
1	Bucyrus-Erie	45R — 9⅞ in. hole	45 ft/hr
Shovels			
2	P & H	1900 AL — 11-yd	400 cu. yds/hr
Trucks			
5	Wabco	Haulpak — 50-ton	
6	Wabco	Haulpak — 75-ton	
8	Euclid	R85 — 85-ton	
Loaders			
2	Caterpillar	992B — 10-yd	220 cu. yds/hr
Tractors			
1	Caterpillar	D8K — Ripper	
2	Caterpillar	D8K — Winch	
2	Caterpillar	D9H — Ripper	
1	Caterpillar	825 — RT	
Graders			
2	Caterpillar	16G — snow wings and plow	

Primary Blasting

The blast pattern is influenced by the pit configuration. The wider the bench, the more efficiently the blast can be designed to combine maximum yardage broken with good fragmentation and minimize the amount of moving that must be done by the electric shovels and drills. By ensuring that the drill pattern is correctly laid out, the holes are drilled to the required depth and are charged properly with the explosive; choke blasts of up to 400,000 cu. yds have been blasted with excellent results.

All explosives are loaded in accordance with the drill and blast foreman's instructions and loading of holes is carried out by only one team of blasters to ensure consistent results, with a high degree of safety.

Ammonium nitrate is delivered to the mine by means of a bulk carrier, offloaded at the storage area into two 105-ton storage silos by the use of a low-pressure blower (7 psi), and then gravity loaded into the Amerind Mackissic 10-ton or 15-ton ANFO truck units for transportation to the open pit, 3 miles (4.8 km) away.

Laboratory tests are carried out at regular intervals to ensure that consistent quantities of diesel fuel are being metered to obtain the required 5.7% diesel fuel addition to the ammonium nitrate prill.

Maximum use is made of the ammonium nitrate fuel oil mixture (ANFO), in spite of water being encountered in the holes. In the summer, wet holes can

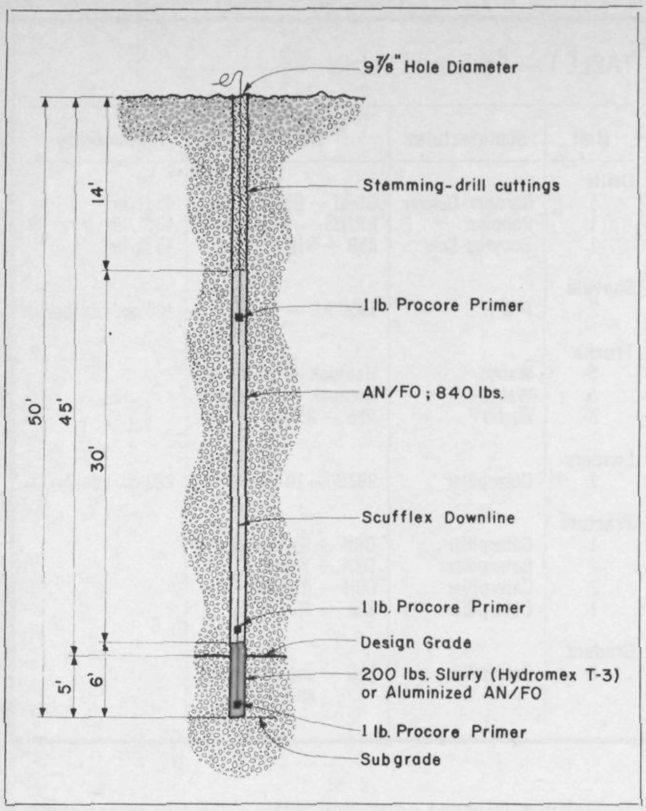


FIGURE 2 — Cross section of a typical blasthole.

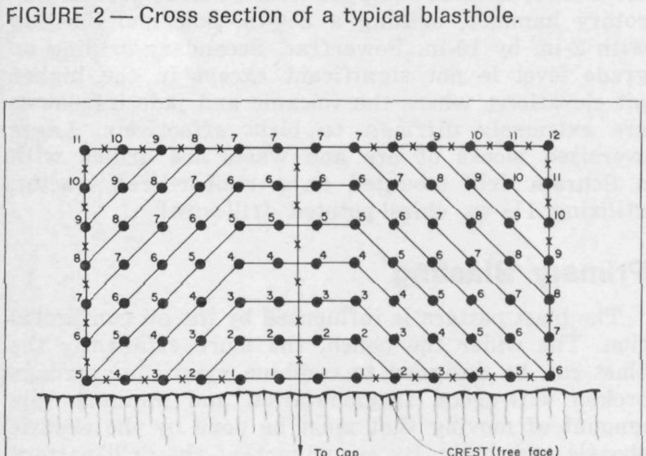


FIGURE 3 — Typical blast initiation.

constitute 40% of all holes loaded. Borehole dewatering is carried out using a Canadian Industries Limited borehole dewatering unit, powered by a hydraulic pump, driving a Prosser Hymergible pump rated at 65 gallons per minute at 150-ft (45.7-m) head. The unit is mounted on a 4x4 pickup (two similar units are available). When using ANFO in wet holes, the hole is pumped dry, and a double-wall, 9-in.-diameter, .004-mill.-thick polyethylene liner is inserted. A base charge of either packaged slurry Hydromex T3 or aluminized ANFO is inserted; the charge is 150 to 200 lb, depending on the material to be blasted. The hole is then column-charged with ANFO, inserting three 1-lb procore primers (Fig. 2). If it is found to be impossible to dewater a drill hole successfully, packaged slurry is substituted, using basically the same charging configuration as shown in Figure 2. No allowance is made for the difference in relative weight or bulk strength of the Hydromex packaged slurry. ANFO comprises 85% of the explosives consumed on site, with excellent fragmentation results.

The charging ratio is varied, depending on the type of rock being blasted. However, an average powder factor to ensure good fragmentation is of the order of 1.3 lb per cu. yd. Reinforced primacord is used, employing a single downhole line and single trunk lines connecting the holes. The delays used are the 15- or 25-m.s. primacord type, depending on whether the blast has a free face or a choke blast. Initiation is in a "V" pattern, as shown (Fig. 3). High-speed filming of blasts on site have confirmed that the hole spacing, charge patterns, blasting ratios and delays being used are in accordance with the computerized program developed by the explosives manufacturers, Canadian Industries Limited.

The charging ratio, although appearing slightly high, has ensured good fragmentation of muck piles and high loading rates with minimum stress on the shovel, and has produced suitable material for front-end loader applications. This also produces a sized material, ensuring that a smooth dump area and bench level can be maintained, thus reducing tire wear.

Wall control between catchments is of major concern, with controlled blasting or perimeter blasting having been practised at various times. The present practice is to survey in the drill line delineating the catchment crest; this line is normally offset 15 ft (4.6 m) away from the designed crest line. Spacing of holes is reduced to 15 ft (4.6 m) on this line and no subdrilling is carried out directly above a crest or catchment. The next production line of drill holes will be set out parallel to this line and 15 ft (4.6 m) distant, but with the normal 22-ft (6.7-m) spacing being used on the blast pattern.

The holes on the drill line are charged with 250 lb of 8-in.-diameter packaged slurry and partially back stemmed with drill cuttings; the next row of blast holes are charged according to the standard hole charge for that particular blast pattern. Where possible, the drill-line holes are detonated in groups, using 15-m.s. delays. Experiments continue using different hole spacings, the possible use of ANFO in cardboard tubes of 6-in. diameter and the use of primadets.

Scaling of walls is carried out by using a complete track pad from an old shovel, attached to a Caterpillar D8 or D9 tractor blade by means of a swivel which allows the track to turn and prevent it from riding up the face when in use. A ship's anchor chain of 4-in.-diameter links, having a tire secured to the trailing end, has also been used in a similar manner with success (Fig. 4).

Truck Loading

Waste-rock loading is carried out using two 1900 AL P&H 11-cu.-yd electric shovels, power being supplied from one of two 3000-KVA sub-stations, located in the pit, depending on shovel location. Power transmission from the main diesel power plant, located 6 miles (9.6 km) away, is by 24-kv pole line. Power to the shovel is carried by No. 2 A.W.G.S.H.G. 5-kv trailing cable at 4160 volts. The cable is lifted off the ground at all pot head connectors to ensure that it is visible during winter snow as well as to keep the pot heads dry during the spring snow melting period.

Shovel maintenance, which is scheduled at 8-12 hours per week during daylight hours, coupled with preplanning major overhauls, has ensured good mechanical availability for a northern location; it is currently running at 81% since the inception of the shovels. Major spare items have been purchased recently to ensure that components can be sent for overhaul without reducing the mechanical availability of the machine. Hoist cables with socketed ends, and also a bail pin unit without sheaves, have been installed to reduce the time required to change broken hoist ropes.

Shovels and other major equipment units are equipped with radios to ensure contact with the mine shift foreman or the mine garage in the event that re pairs are required. The high standard of radio maintenance has assisted in ensuring excellent control of operations and minimizing downtime on equipment.

A Caterpillar 824 rubber-tired tractor is used for cleaning up rock spillage around the shovels, moving cables during blasting operations and generally assisting in material movement. To maintain an acceptable grade elevation, Caterpillar D9G tractors are equipped with single-shank rippers. The digging limits with reference to the toe or crest positions are marked by stakes in the field and a daily reference survey is carried out by the surveyors to ensure that strict adherence to the line and level is maintained.

To supplement the waste rock production units, a Caterpillar 992 10-cu.-yd front-end loader is used during the periods when the shovels are not available. Special emphasis is also placed on keeping the catchments clean, using the 992 front-end loaders (Fig. 5), to ensure that equipment or personnel working below the waste-rock excavation levels can work with safety. The 992 front-end loader was selected on the basis that it can load 50-ton, 75-ton and 85-ton trucks with ease, thus complementing the shovels.

When using the Caterpillar 992 front-end loader, a dramatic increase in production has been achieved when a D8 or D9 tractor is deployed in pushing rock to the loader. The additional cost of using the tractor is more than compensated for by the increased production. A tractor is always used with the loader during catchment cleaning operations.

Shovel production is running at 400 cu. yds per hour. However, when mining the less-competent serpentine rock associated with the orebody, production of 540 cu. yds per hour is not uncommon.

All support equipment is Caterpillar supplied and maintained on a contract basis by Finning Tractor and Equipment Co. Ltd., Vancouver. A spare parts depot is located at the plant complex and a repair facility has been built in the pit.

Haulage

The haulage fleet employed on waste rock consists of eight R85 Euclids of 85-ton capacity, powered by Detroit Diesel 16V 92T 820-hp series engines with DP8961 Allison transmissions, as well as six Wabco 75-ton-capacity trucks powered by VTA 1710-12V 700-hp Cummins engines with Allison DP8961 transmissions.

The 14 trucks are assigned to hanging-wall waste production. Normally, nine trucks are required to service the two 1900 AL P&H shovels and the Caterpillar 992 front-end loader. Trucks are double-side loaded by the shovels whenever possible. Studies carried out indicated that a 40% increase in production can be obtained as compared to single-side loading. The shovel or drill trailing cables are protected at truck crossing points by inserting the cable in an Irathene elastomeric crossover pad containing a slot for the cable. A strip insert is placed in the slot to protect the cable, and the pad is covered by fine rock to protect the pad from damage during cold weather (Fig. 6).

Haul ramps are designed 70 ft (21 m) wide at an 8-10% grade and are graded by a Caterpillar 16G grader, using serrated ice blades in winter. Grading on the dumps is maintained by D8 tractors with the assistance of the grader. A dump person is employed when trucks are dumping over the edge. All dump areas are lighted at night by portable light units; the

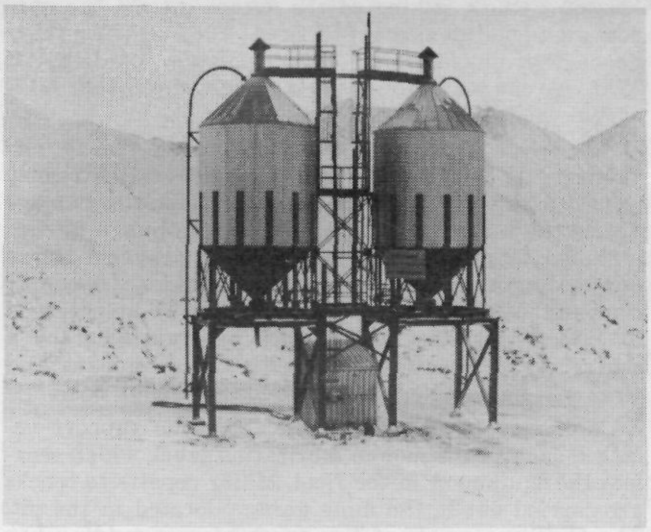


FIGURE 4—A. N. prill storage silos—total capacity 210 tons.

FIGURE 5—Caterpillar 922 front-end loader cleaning catchments to avoid rock spills onto the ore faces below.

slumping and cracking of the dumps dictate the need for good maintenance practices. Haul lengths vary depending on location, with averages of 2500 ft (762 m) one way, uphill hauls comprising about 50% of the production time.

Tire wear is a major concern during spring and summer, as the combination of water and extremely sharp-edged rock fragments cause serious tire damage unless the roads are well graded and drainage ditches installed. A revised tire maintenance program begun one year ago with the assistance of a major tire supply company is already paying dividends. It is proposed to extend this tire maintenance program whereby a contract will be entered into with a tire company and tires will be supplied on a unit-cost-per-hour basis.

Chains are used on graders and some support vehicles, such as the mobile lubrication truck which services the equipment in the field. Major repairs to trucks are carried out in the mine garage, located in the pit. The facility consists of five truck bays, a welding bay and a steam bay.

Ore Mining

Mining of ore is carried out 260 days per year, three shifts per day, with two days of scheduled maintenance each week on the crushing plant and tramline. The daily production rate is 5000 tons, with an average recoverable mine grade (R.M.G.) of 9%.

Drilling of the ore and serpentine host rock is performed with a diesel-powered Robbins R1OS, mounted on an International TD25 tractor, utilizing a 9-in.-diameter hole and capable of drilling 60 ft (18.2 m) if required. Prior to drilling, the area is levelled and prepared for the surveyors to mark out the required drill pattern. Patterns vary from 22 ft (6.7 m) by 22 ft in the summer to 18 ft (5.5 m) by 18 ft in the winter months on a 30-ft. (9.1-m) deep bench. The top 3 to 4 feet of the ore bench are saturated with water during the summer, therefore, once freezing conditions prevail, this portion freezes and is very difficult to fragment during blasting. On occasion, holes 8 ft (2.4 m) deep are drilled between each row of production blast holes, charged with 25 lb of explosives and blasted with the main blast to try to alleviate the problem of large blocks of ore being produced after blasting. This blocky ore is extremely difficult to fragment in the jaw crusher. To help reduce orebody saturation caused by water, ditches and culverts are installed in the summer to control and direct water to the dewatering sump.

FIGURE 6 — Double-side shovel loading after a typical blast.

The ore mining sequence is dictated by the sales commitments, and a three-month mining plan is drawn up each month and revised as required. Blending of ore in the pit is practised under the supervision of the mine geologist, with all information required being recorded in the mine foreman's office.

Contamination of ore by foreign objects cannot be tolerated and great care is taken to ensure cleanliness, both on the ore bench and during the ore crushing cycle.

No subgrade drilling is practised, as the ore is relatively easy to blast at a normal charging ratio of 0.4 lb per ton. Whenever possible, ANFO is used, with excellent results. Drill holes are not normally pumped in the asbestos ore, as no polyethylene liners can be used. A special explosive, "Cilgel", has a package covering that conflagrates during the blast. All blast initiations are carried out using a capped fuse attached to primacord in preference to electric blasting, thus preventing ore contamination by lead wires. After the blast, the bench is levelled to ensure that during winter any heavy snowfall can be cleaned off the ore prior to sending it to the crushing plant. Fragmentation of the asbestos ore is controlled to ensure maximum rejection at the mill concentrator, thus ensuring that no unnecessary rock is dried or milled. A 30% rock rejection is the scheduled target at the concentrator.

Loading of the ore is carried out with a Caterpillar 992 front-end loader, equipped with a 10-cu.-yd bucket and beadless tires (linked tracks similar to tractor pads), which have proved effective in reducing tire spin, thus reducing rubber contamination in the ore. Whenever possible, serpentine waste rock is made available to enable the loader to load ore and waste. As the loader is radio equipped, the crusher operator can direct the ore trucks to waste and inform the loader operator when the next requires ore.

The ore haulage fleet consists of five 50-ton Wabco trucks, equipped with VTA 1710 12V 635-hp Cummins engines and Allison 6061 transmissions. Normally, two trucks are running during the ore cycle. If waste is available close by, an additional truck is added to the fleet. Truck counts are taken of the ore hauled to the crusher; a survey check is made and the ore is weighed at the mill. This has established that the 50-ton Wabco trucks average 44 tons per load. Haul roads are kept at an 8-10% gradient whenever possible. Salt and sand are used during winter months and crushed gravel is spread on the main footwall haul road to ensure good drainage.

Dewatering

Pumping is required to ensure that the ore is kept as dry as possible (so that less fuel will be required for ore-drying) and to ensure good access to the ore during the spring melting and the summer rain periods. Snow is normally cleaned from the catchments and the pit area in April to minimize the inflow of water from the snow melt. A truck equipped with a 4000-gallon tank is used to haul water out of the pit until such time as the electric submersible pumps and the 8-in.-diameter pipeline can be installed (in May) without freezing problems. A 2250B Flygt pump, rated at 700 gallons per minute, with a combined static and friction head of 215 ft (65.5 m), is installed in the sump at the pit bottom (5570-ft elevation on Phase 7). The pump is installed on a base made from 6-in. 'I' beams to ensure that the suction inlet is clear of the sump bottom, as the long asbestos fibre tends to block

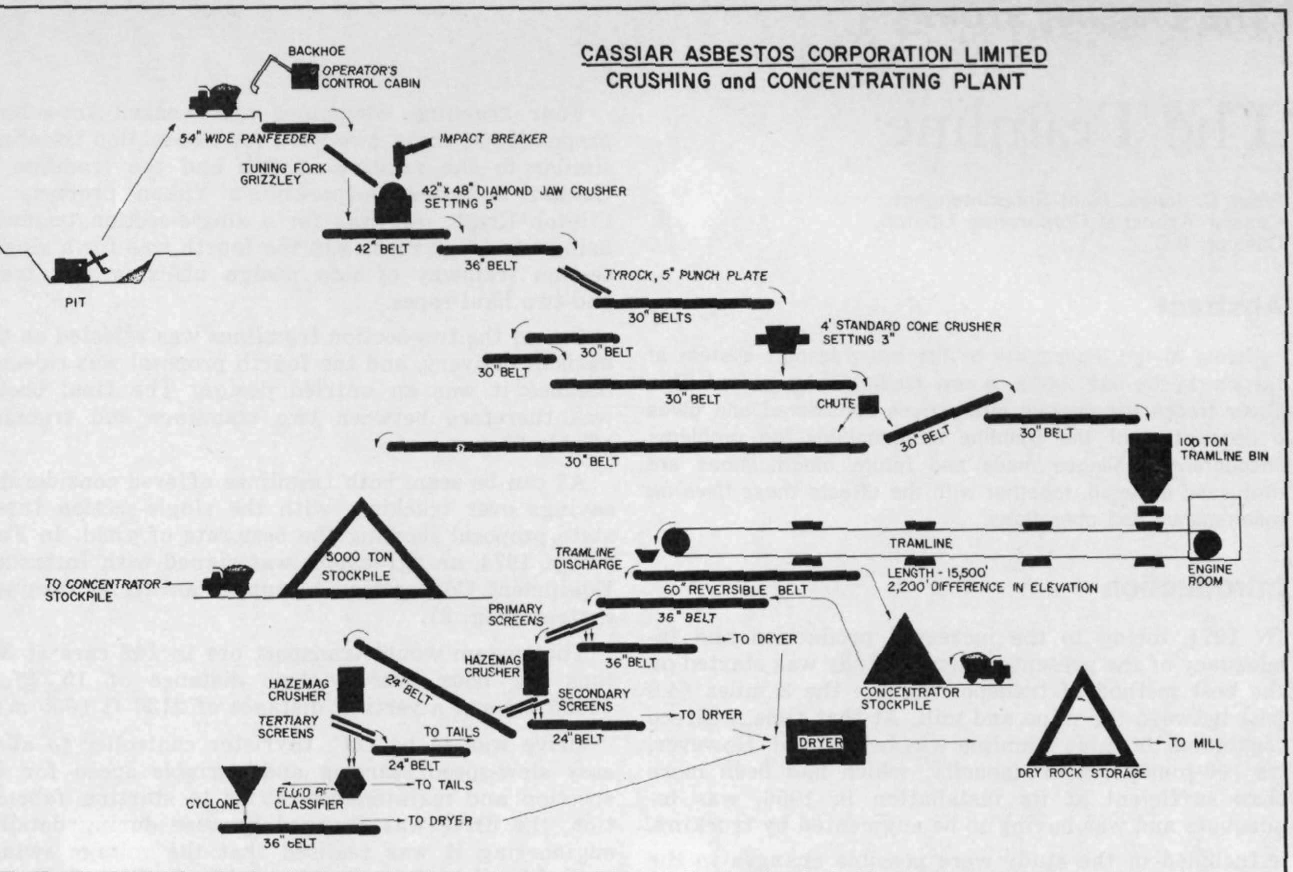


FIGURE 7 — Cassiar's crushing and concentrating plant flowsheet.

the inlet openings of the pump if left on the sump bottom, with subsequent overheating of the pump, which will destroy the seals.

The 8-in.-diameter pipeline is 2500 ft (762 m) in length, fitted with a check valve to protect the pump from water backflow pressure and discharges at the northwest corner of the pit. Pit configuration prohibits pump discharges at any other location for safety reasons regarding dump lubrication.

Crushing

Crushing is carried out at the pit, the trucks dumping onto a variable-speed 54-in.-wide plate feeder, discharging onto a tuning-fork grizzly, with the oversize +6-in. going into a 42- by 48-in. Diamond jaw crusher, set at a 5-in.-opening. All crushing operations (Fig. 7) are controlled from a cabin located above the 54-in. plate feeder. The operator can visually observe and control the trucks dumping, and can also observe the feed discharging into the jaw crusher by means of a 30-in.-diameter convex mirror. T.V. camera monitors have been installed in the operator's cabin, which allows the monitoring of strategic locations in the crushing cycle. Indicator lights display equipment running and ammeters connected to conveyor drive motors show overload conditions. The conveyors are equipped with zero speed switches, which should trip the motor in the event of belt slippage.

The flowsheet (Fig. 7) depicts the complete crushing operation, which is controlled by a crew of four persons, including the tramline operator. Radio communications and a paging system are available in the crushing plant, tramline loading station and tramline discharge terminal. Crushed product delivered to the tramline for transportation to the mill has a nominal size of –3-in. after having passed through the 4-ft standard Symons cone crusher.

Dust lifts are installed at all major conveyor transfer points, the air being supplied by two 8000-cfm Wheelabrator baghouses, operating at 2-in. W/G, discharging back onto the conveyor system. The primary crushing plant walls have recently been covered with 20-gauge galvanized sheeting, which reduces the adherence of dust to the inside walls and also enhances the inside appearance with an increased lighting effect. A Hoffman 4000-cfm vacuum cleaning system operating at 11.5-in. W/G has been in operation for 2 years and is extremely useful in collecting dust or spillage up to a maximum fraction size of half an inch. A primary separator has been installed in the primary crushing area. Discharge is through airlocks and the collected material is discarded to the waste dump, due to high contamination by foreign objects. A piping system provides 2-in.-diameter inlets at strategically located points to allow use of the vacuum system in all areas of the primary and secondary crushing plants.

Heating the crushing plant is essential in winter. It is accomplished by oil-fired hot air in conjunction with propane radiant heaters. Transfer points are heated with a propane blow torch fitted securely inside an 8-in.-diameter pipe.

The flowsheet depicts the route of the crushed asbestos ore, which is conducted from the cone crusher/Tyrock screens to the tramline loading station by two 30-in.-wide belts. An alternate route is available for the ore in the event of a major period of downtime experienced on the tramline. This consists of a bypass chute onto a 30-in.-wide belt dumping on a stockpile capable of stocking 5000 tons. The ore can then be transported to the wet rock stockpile at the mill by truck.

The Tramline

Peter C. Jones, Plant Superintendent,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

Owing to the inadequacy of the ore transport system at Cassiar in the late 1960's, a new facility was required. This paper traces the various alternatives considered and gives a description of the tramline as installed. The problems encountered, changes made and future modifications are discussed in detail, together with the effects these have on maintenance and operations.

Introduction

IN 1971, owing to the increased production and inadequacy of the present system, a study was started on the best method of transporting ore the 3 miles (4.8 km) between the mine and mill. At that time, a Breco continuous bi-cable tramline was being used. However, its 100-tons-per-hour capacity, which had been more than sufficient at its installation in 1956, was inadequate and was having to be augmented by trucking.

Included in the study were possible changes in the crushing and concentration plant, which were then situated at the mine site (Fig. 1).

Three basic proposals were considered, together with their variations: a new tramline, a conveyor and trucking. For each proposal and variation, estimates were made of capital and operating costs and estimated rate of yield over a 16-year life (Table 1).

After further study, it was decided that the design parameters should be for an ore mining rate of 1,200,000 tons per year, with the concentrator at the mill. This would allow for a steady drier feed, resulting in increased drier efficiency. On this basis, a closer cost study was undertaken.

The conveyor-belt option was rejected on the basis that its capital cost was too high and the amount of ore to be moved would be too low to take advantage of this method of high-volume delivery. The local topography also ruled against a conveyor.

Peter C. Jones was born in England in 1947, and received his early education in England and later in Wales. In 1968, he graduated in electrical engineering from Rugby College of Engineering. His early career was as a student engineer in nuclear power plant construction. From 1969 to 1972, he worked as assistant engineer and section engineer for Anglo American Corporation in Zambia in both underground and open-pit operations. Since coming to Canada in 1972, he has worked for Newmont Mining as electrical superintendent and plant superintendent at the Granduc operation and later as plant superintendent at the Similkameen copper mine.

Mr. Jones is currently plant superintendent at Cassiar — a position he has held since joining the company in late 1976.

Keywords: Cassiar Mine, Asbestos, Transportation, Tramlines, Loading, Ropes, Wire ropes, Cables.

Four tramline companies were asked to submit proposals. Of these, two were for two-section tramlines similar to the existing system and the tramline at Cassiar Asbestos Corporation's Yukon property at Clinton Creek, one was for a single-section tramway using two track ropes and the fourth was for a single-section tramway of new design utilizing two track and two haul ropes.

One of the two-section tramlines was rejected on the basis of delivery, and the fourth proposal was rejected because it was an untried design. The final choice was therefore between two tramlines and trucking (Table 2).

As can be seen, both tramlines offered considerable savings over trucking, with the single-section Interstate proposal showing the best rate of yield. In February, 1974, an agreement was signed with Interstate Equipment Corporation to supply an aerial tramway system (Fig. 2).

The system would transport ore in 142 cars at 300 tons per hour over a slope distance of 15,370 ft (4685 m) and a vertical distance of 2130 ft (650 m).

Drive was to be D.C. thyrister controller to allow easy slow-speed starting and variable speed for inspection and maintenance. Prior to starting fabrication, the drive was changed because during detailed engineering it was realised that the voltage swings caused by shovels in the pit combined with this drive/regenerative application would lead to a frequent blowing of the thyrister protection fuses or the thyristers themselves. The choice then was between a more expensive Ward-Leonard control system or an induction motor drive with a separate pony motor for slow-speed operation. As commissioned, the drive consisted of two 300-hp 550-volt 900-rpm induction motors driving dual-input shafts of a Falk gear reducer, one motor being designed to start on reduced voltage. The driveshaft of the gear reducer drives a 188-in.-diameter drive wheel at 12.45 rpm, giving a tramline speed of 600 fpm. Spring-loaded grips on the drive wheel grip the tramline haul rope, thus transmitting the drive during starting and the required braking during full-loaded running. Emergency and service braking are provided by two brake bands acting on the drum — similar to many shaft hoists. Tension is kept in the haul rope by using a weighted travelling truck-type constant tension unit at the discharge. This moves a total of 20 ft (6.1 m), giving 40 ft (12.2 m) of take-up.

A 75-hp pony motor was fitted for 100-fpm running and initial starting. This suffered from several disadvantages. For maintenance, the fixed-speed operation led to extended time for greasing, as the cars moved at a slow pace and full start-up procedures had to be followed after every stop. Worse still, after prolonged shutdown of the system in sub-zero weather the pony system would not provide sufficient torque to move the tramline. On at least one occasion, the tramline had to be towed with a 'Cat' and the buckets manually filled until the tramline had sufficient inertia to be started.

These problems eventually led to the design and installation of an hydraulic pony motor drive in 1977. In this system, two hydraulic motors power shaft extensions on the motor via air-operated clutches. The hydraulic system brings the tramline up to about 180 ft/min. (24.4 m/m), at which time the main motors are started and the clutches disengage the hydraulic

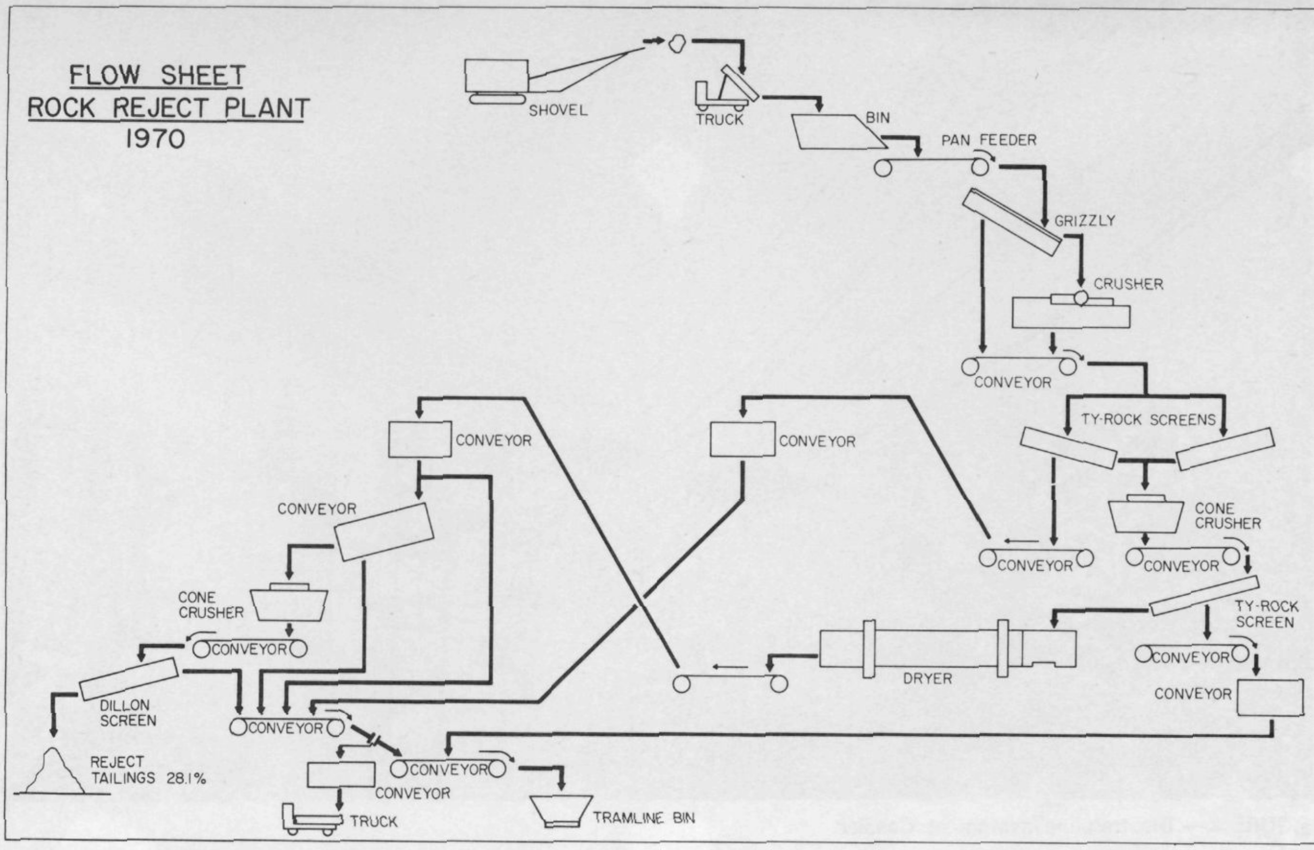


FIGURE 1 — Flowsheet of the rock plant, 1970.

TABLE 1 — Comparison of Delivery Methods at Cassiar

	SYSTEM	1 PRESENT	2 T-L	3	4 TRUCKS	5 T-L	6 CONV.	7 TRUCKS
	Location of Crusher	Mine	Mine		Mill	Mine	Mine	Mill
	Location of Rock Reject	Mine	Mill	Mill	Mill	Mill	Mill	Mill
Unit Costs (Dollars)	Loading Bay	0.21	0.21		0.175	0.17	0.17	0.17
	Haul to Prim. Crusher	0.25	0.25			0.144	0.144	
	Primary Crushing	0.67	0.15		0.28	0.14	0.14	0.26
	Concentration		0.18			0.17	0.17	
	Delivery to Mill Area	0.887	0.30		0.54	0.27	0.135	0.54
	Dryer Saving (Deduct)							
	TOTAL	2.017	0.09		0.995	0.894	0.759	0.97
Dollars	Cost per Year	2,420,000	1,310,000		1,197,000	1,340,000	1,140,000	1,140,000
	Savings per Year		1,110,000		1,223,000	1,680,000	1,880,000	1,560,000
	Power Savings		60,000			75,000	37,500	
	Total Savings per Year		1,170,000		1,223,000	1,755,000	1,917,500	1,560,000
	After-Tax Savings							
Capital	For Delivery System		2,660,000		1,833,000	2,910,000	4,750,000	1,064,000
	For Conc. Plant		2,000,000		2,500,000	2,200,000	2,200,000	2,700,000
	For Dryer					450,000	450,000	450,000
	Tailings Disposal (1700 ft)		340,000		340,000	340,000	340,000	340,000
	TOTAL							
Rate of Yield (16-Yr Life)	Capital/Savings		7.1		6.36	5.6	6.28	5.93
	Rate of Yield		18.6%		20.7%	23.4%	21%	22.2%

Notes: 1. At the 1500,000 ton per year rate, assume costs for the present system would be the same per ton. Cost per year = $3,020,000
2. Assume $0.07 per ton to stabilize the rock reject pile.

FIGURE 2 — The tramline system at Cassiar.

system from the normal drive. The high torque capabilities of this system have overcome start-up problems and the infinitely variable speed between 0 and 180 fpm (forward or reverse) has cut greasing time from 16 hours to 8.

Loading

The tramline loading is fully automatic (Fig. 3). Ore is transported from the crushing plant by conveyor and fed into a 100-ton-capacity bin. The bin is heated to prevent freezing of the ore, and kept empty to avoid freezing whenever the tramline is shut down. From the bin apron, feeders feed two counter-weighted air-operated 40-cu.-ft (1.12-m^3) loading hoppers. The feeder is interlocked so that the apron feeder is shut off when the hopper reaches a pre-set weight.

TABLE 2 — Financial Comparison of Delivery Systems

Case	1	2	3	4
Mine Rate Per Year	1,200,000	1,200,000	1,200,000	1,200,000
Delivery Rate Per Year	900,000	1,200,000	1,200,000	1,200,000
Method	**Present**	**Trucks**	**T. Line A** (Riblet)	**T. Line B** (Interstate)
Unit Cost ($)	1.38	0.79	0.44	0.31
Cost per Year ($)	1,242,000	948,000	528,000	372,000
Savings per Year ($)		294,000	714,000	870,000
After-Tax Savings ($)		176,000	458,000	552,000
Capital Required ($)		3,223,000	3,973,000	4,313,000
Ratio — Capital Exp. to Income after Tax		18.27	8.66	7.81
Rate of Yield — %		6.2	15.8	17.5

*Costs in 1973 ($)

As the car comes into the loading area, it passes an infra-red photo cell which activates a timer set to open the loader hopper as the car passes below. The control system allows loading to be done automatically from either one or both hoppers alternately or manually.

A system of belts below the loading area removes any spillage.

Cars

The 142, 40-cu.-ft (1.12-m^3) tramcars are each mounted on two axles, with a two-wheel bogie at each end (Fig. 4). The initial wheels were a mixture of cast and fabricated units mounted on roller bearings. Below each truck are two brackets for attachment of the 2¼-in. open socket fitted to each end of the 214-ft (65.2-m) long, 2¼-in. (6x25 Langs lay) haulage rope. The socket is attached below the car with a steel pin through a bushing.

The first modification made to the cars was the addition of a three-piece ¼-in.-thick polyethylene lining to help prevent the ore sticking during the alternately freezing and thawing weather conditions. This sticking has been further alleviated by installing a four-nozzle automatic blowpipe at the discharge terminal. The combination of these two have been effective and have reduced spillage of ore from the cars.

After only a short period of running, the pins holding the haul-rope sockets to the cars were found to be spalling badly, preventing free movement of the socket. Several different designs of pin were used, but the only effective method of stopping the problem was to grease the pins at three-week intervals. Cassiar has recently modified a few tramcars and fitted bearings for the pin to rotate in; however, at this time we cannot say whether this will be successful.

A third area of modification on the tramcars is the

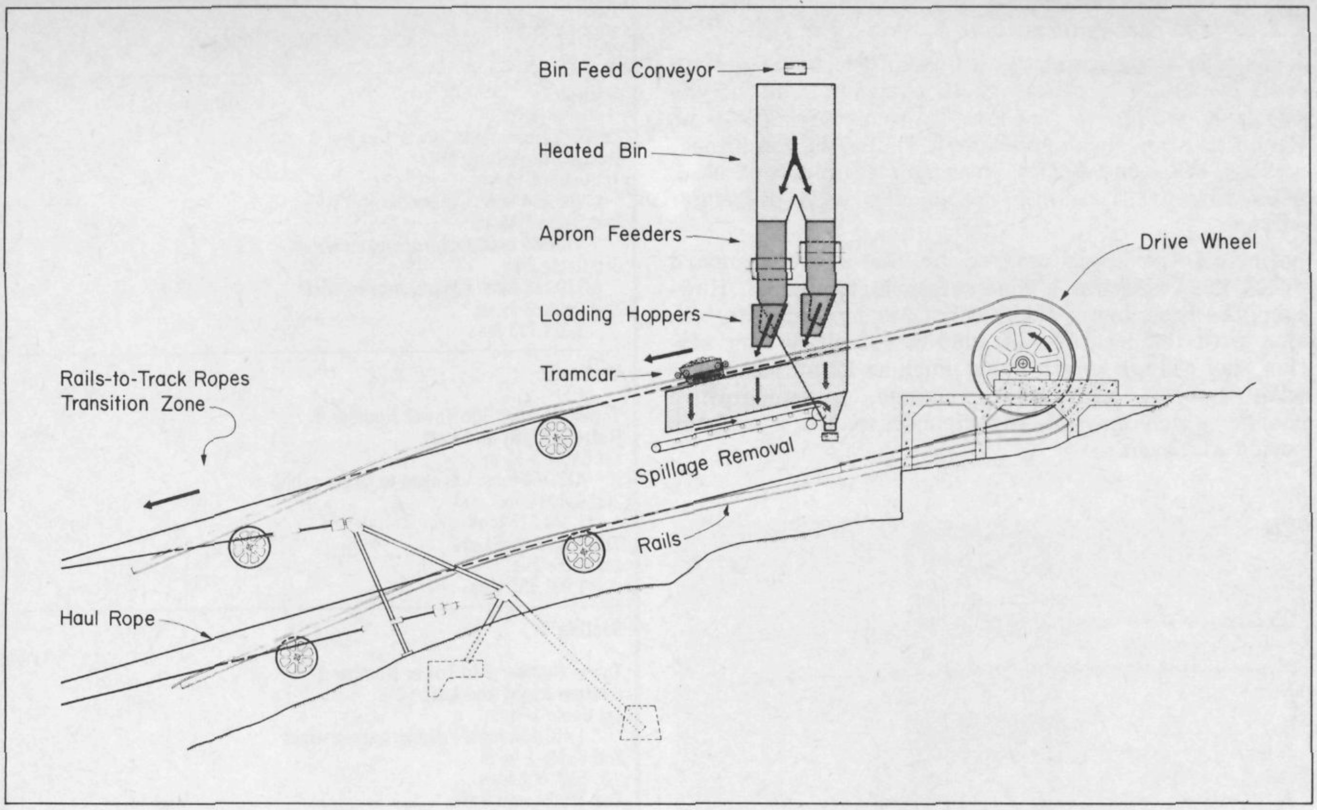


FIGURE 3 — Tramline loading terminal.

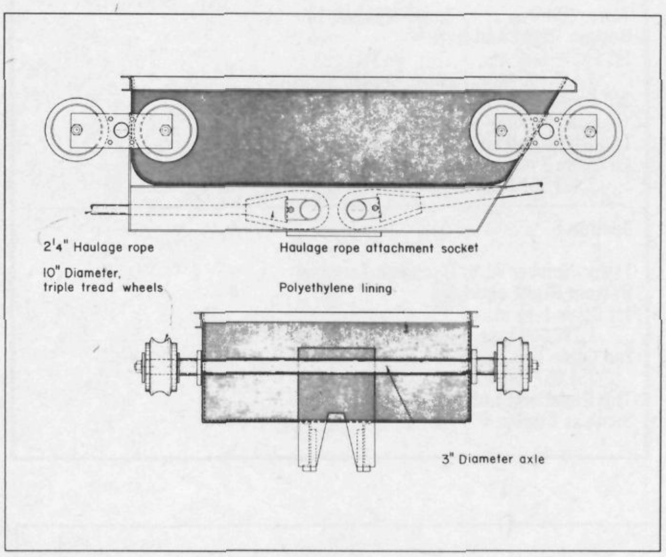


FIGURE 4 — Cross sections of a 40-cu.-ft tramcar.

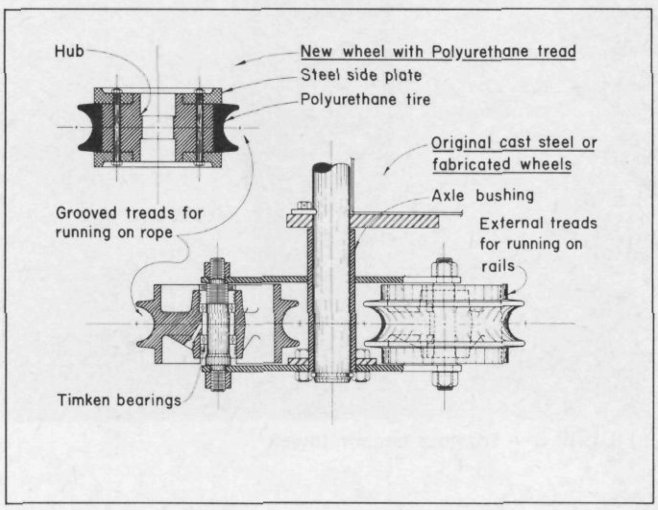


FIGURE 5 — The tramcar wheel assembly.

wheels. Experience to date indicates the useful life of the original wheels (Fig. 5) to be about 2.5-3 x 10^6 tons. At this stage, the wheels would either have to be replaced with new units or built up and machined. Because of this, and in an effort to reduce track rope wear, Cassiar has, over the last six months, been experimenting with lined wheels.

The first type were made by machining out the rope tread of the original wheels and moulding in a polyurethene liner. Later, a split wheel of steel and aluminum was designed which would take a removable 'tire'. It is also fitted with a cartridge-type bearing for ease of maintenance. This design has allowed the company to easily change tire materials in order to test for the best compound. Present estimates show a life expectancy of about 1 x 10^6 tons per tire.

Track Ropes and Towers

The tramline is divided into five rope sections, between anchor points: from the load terminal to Tower 3 is 2500 ft (762 m); Tower 3 to Tower 8 is 4000 ft (1219 m); Tower 8 to Tower 11 is 2500 ft (762 m); Tower 11 to Tower 14 is 2500 ft (762 m); and Tower 14 to the discharge is 4000 ft (1219 m). The original installation was fitted with 2-in. locked-coil ropes for the top (loaded) ropes and 1⅝-in. ropes on the return side. The ropes are held at either end in the anchor towers and also fastened by clamps into the saddles of the pivoted-base intermediate towers (Fig. 6).

To fully understand the problems encountered with track ropes, it is necessary to consider some of the problems unique to Cassiar. There are currently no Canadian standards applicable to industrial tramlines; instead, the standard for passenger tramlines is used, neglecting areas dealing specifically with passenger service.

One of the areas covered by the above standard (CSA Z98) calls for a rope safety factor of 3.3. However, the rope, being fixed at both ends, alters its tension with temperature. A 4000-ft (1219-m) long section may change length by as much as 12 in. (30.5 cm) with a 28°C temperature change, a temperature change which may be experienced within a 24-hour period at Cassiar.

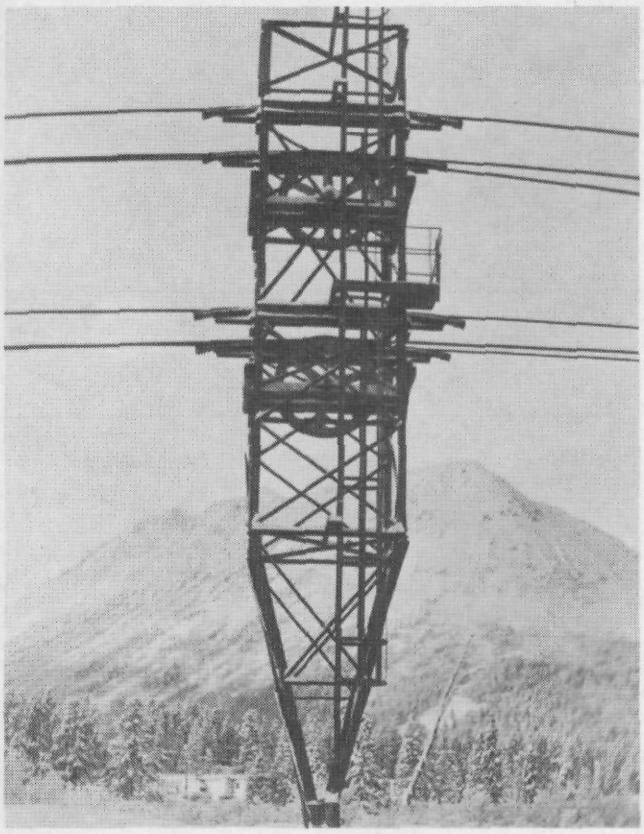


FIGURE 6 — Tramcar anchor tower.

TABLE 3 — Track Cables Tonnage Report

Section 1

Loading Terminal to Tower Number 3
Bottom Right and Left
1st Cable 1-5/8 in.
196,454 tons. Failure, broken wires
2nd Cable 1-5/8 in.
413,764 tons. Failure, broken wires
3rd Cable 2 in.
710,218 tons. Failure, broken wires
2nd Cable 2-1/2 in.
1,221,793 tons

Section 2

Tower Number 3 to Tower Number 8
Bottom Right and Left
1st Cable 1-5/8 in.
413,764 tons. Changed to larger cable.
2nd Cable 2 in.
1,344,217 tons
Top Right and Left
1st Cable 2 in.
1,931,557 tons

Section 3

Tower Number 8 to Tower Number 11
Bottom Right and Left
1st Cable 1-5/8 in.
1,401,355 tons. Failure, broken wires
2nd Cable 1-5/8 in.
530, 602 tons
Top Right and Left
1st Cable 2 in.
1,931,957 tons

Section 4

Tower Number 11 to Tower Number 14
Bottom Right and Left
1st Cable 1-5/8 in.
1,621,600 tons. Failure, broken wires
2nd Cable 2 in.
310,357 tons
Top Right and Left
1st Cable 2 in.
1,931,957 tons

Section 5

Tower Number 11 to Discharge Terminal
Bottom Right and Left
1st Cable 1-5/8 in.
1,180,000 tons. Failure, broken wires
2nd Cable 1-5/8 in.
751,957 tons
Top Right and Left
Same as Section 4

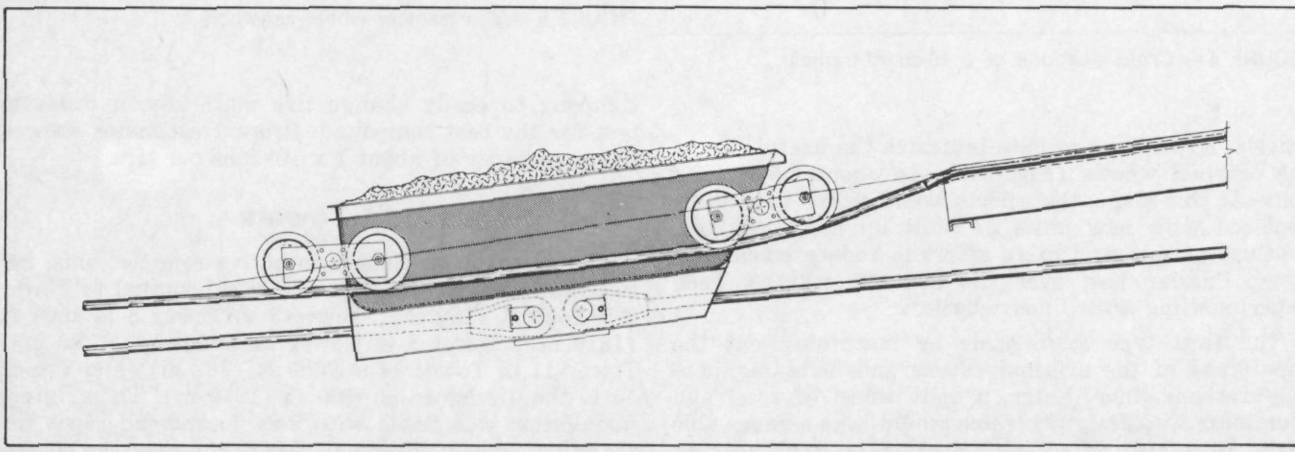


FIGURE 7 — Approach of car to tower.

On a hot day, the rope sag means that cars approaching a tower climb steeply as they enter the saddle and descend similarly at the exit. This will produce very high loadings on the rope at this point (Fig. 7). Conversely, if the rope is tensioned sufficiently to overcome this, a drop in temperature can increase the rope tension past the yield point of the rope.

Because of these problems and the necessity to stay within the 3.3 safety factor, rope sizes are being increased to 2.28 in. (58 mm) and 2 in. (51 mm) on the top and bottom respectively (Fig. 7).

Approach and exit angles, wear on the rope as the car comes out of the tower, rail wear within the tower and terminals, and methods of solving these problems are open to many theories. In an attempt to gather as much information as possible, we have used a high-speed movie of the tramline, slowed down to show the rope and car movement. We are also embarking on a test, together with help from Wire Rope Industries Limited and Noranda Research, to place an instrumented tramcar in operation which will record car wheel loadings at points along the route. It is hoped that this information will help the company to design saddle and tower configurations to increase the present short rope life (Table 3) to at least 6,000,000 tons.

Apart from the increased rope diameters, we have already tried or are in the design stages of other modifications to better the original configurations.

On two towers on the top sections of the tramline, 20-ft (6.1-m) extension pivoting rails were fitted to help support the car entering the tower and relieve the rope of the strain. However, experience is showing that rather than alleviate the problem of rope strand breakages these are just being moved another 20 ft (6.1 m) farther along the line. On two towers we are now experimenting with a longer saddle of larger radius, to give a more gradual change in rope-saddle-rope transition.

Although these new saddle designs have only been installed for a few months, they appear to be having the desired effect, and more will shortly be fitted (Fig. 8).

As mentioned earlier in the text, increases in ambient temperature can cause sagging. This, in fact, caused a derailment on the 1060-ft (323-m) long top span of the tramline, when the line became unstable. A tie-down system was then installed to increase the stability, but on hot days it is still sometimes necessary to shut down the tramline due to sway in this section.

The permanent solution will be to install an additional tower to reduce the length of the span. This has now been designed and will be installed during the summer of 1978. While the extra tower was being designed, alternative designs were also produced for the other towers in the top section, changing the profile to a more gentle one and embodying the new saddle design (Fig. 9). If put into practice this change should further extend rope life.

Discharge Terminal

The discharge terminal is an enclosed steel structure providing a steel framework for the 10-ft-diameter urethane-lined floating tail sheave and the discharge hopper (Fig. 10). The hopper is designed to receive ore from any location of the tail sheave. Below the hopper is a 60-in. reversible belt which directs the ore either via belt to the concentrator or via a radial stacking conveyor onto a wet-rock stockpile. An ultrasonic level detector within the hopper signals the bin high-low levels.

Electrical

As described under the drive system, two 300-hp 575-volt induction motors drive through a Falk reduction gear box. Under full load, the motors regenerate, about 400 kw/hr. The hydraulic pony drive is powered by two 100-hp motors, the variable-rate pumps being controlled by a 'Moog'-type controller.

Safety systems on the tramline, consisting of overspeed, underspeed and axle limit switches, all originate shutdown and braking through an undervoltage relay. Speed read-outs are provided by an 'Airpax' system.

Now in the design stages is a derailment alarm and

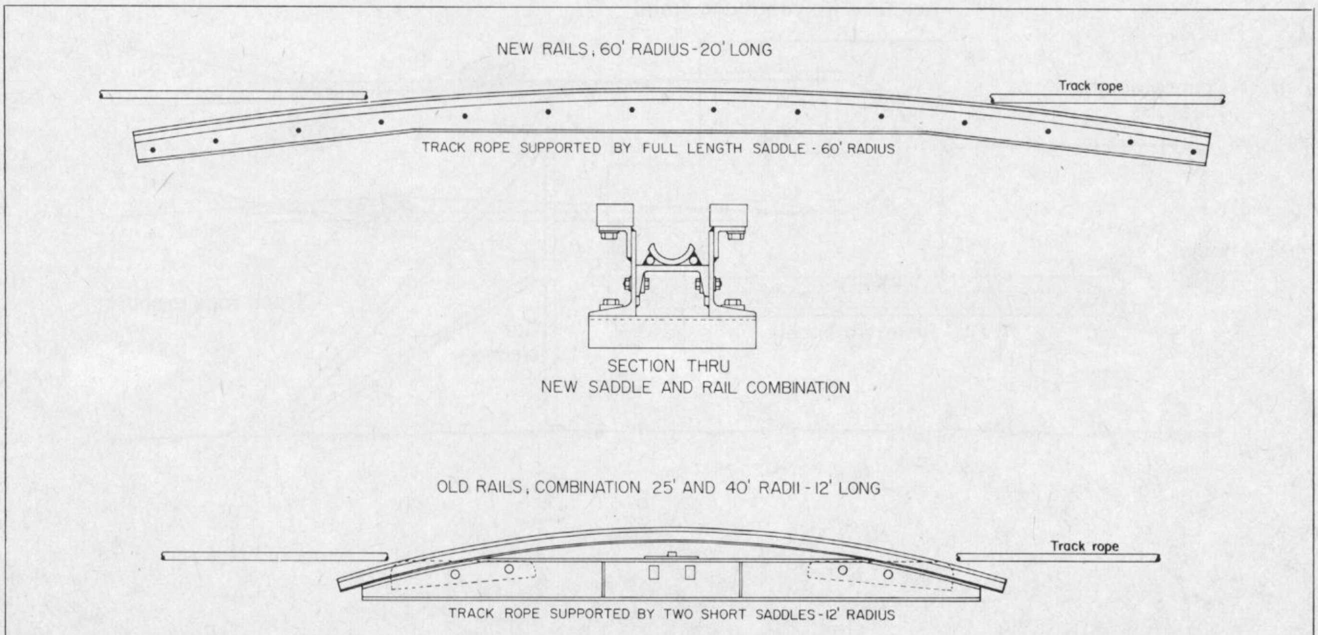


FIGURE 8 — Old and new tower saddles and rails.

shutdown system to be fitted to all towers, because at present a derailment could destroy several towers before the underspeed relay initiated a shutdown.

Maintenance

As may be expected from reading this paper, maintenance requirements are extremely heavy. A crew of 11 men, with a foreman and general foreman, look after the mechanical maintenance requirements of the crusher and tramline. The tramline works on a five-day, three-shift basis, with two days normally scheduled for maintenance. In addition to these two days, additional time is scheduled as required for major work.

During the time the tramline is operational, the track ropes are lubricated by a special oiler car which is permanently fastened in the string of tramcars. The car wheel operates a pump to apply lubricant to the

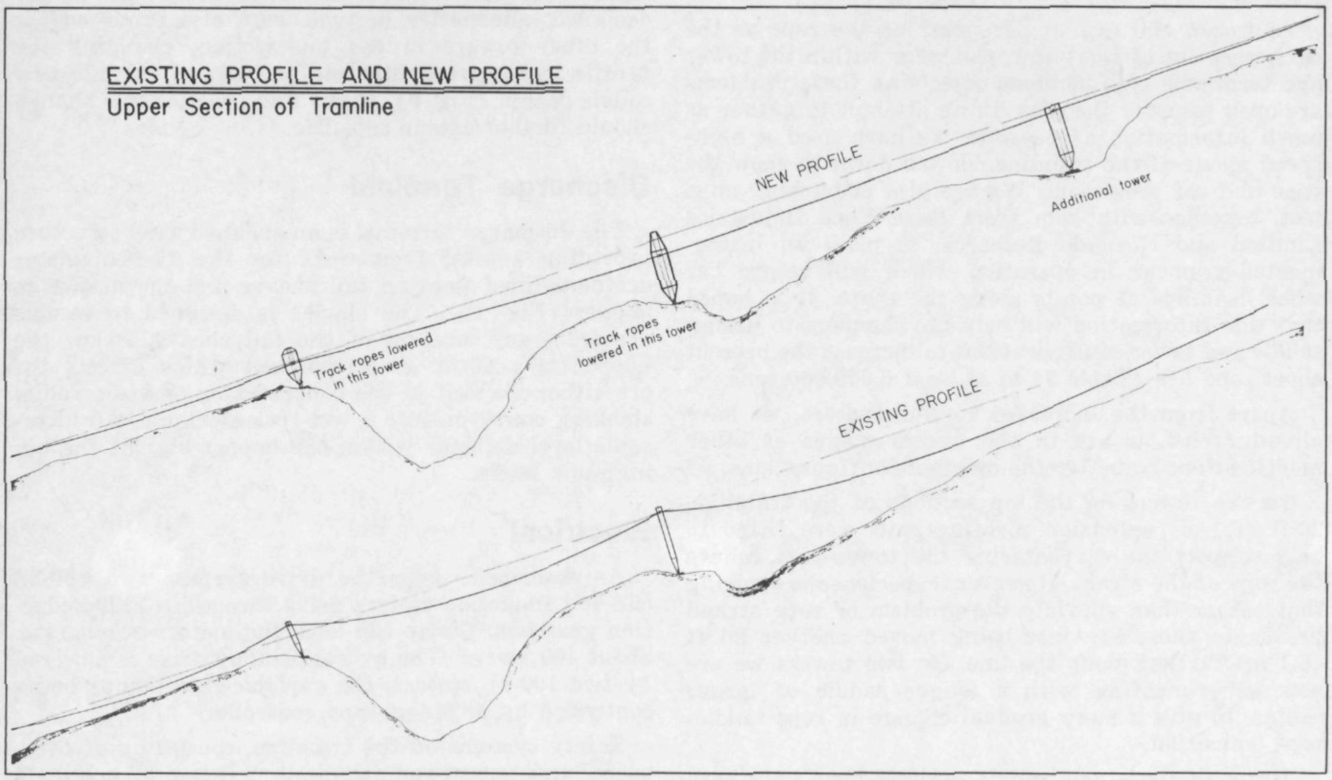


FIGURE 9 — Existing profile and new profile for the upper section of the tramline.

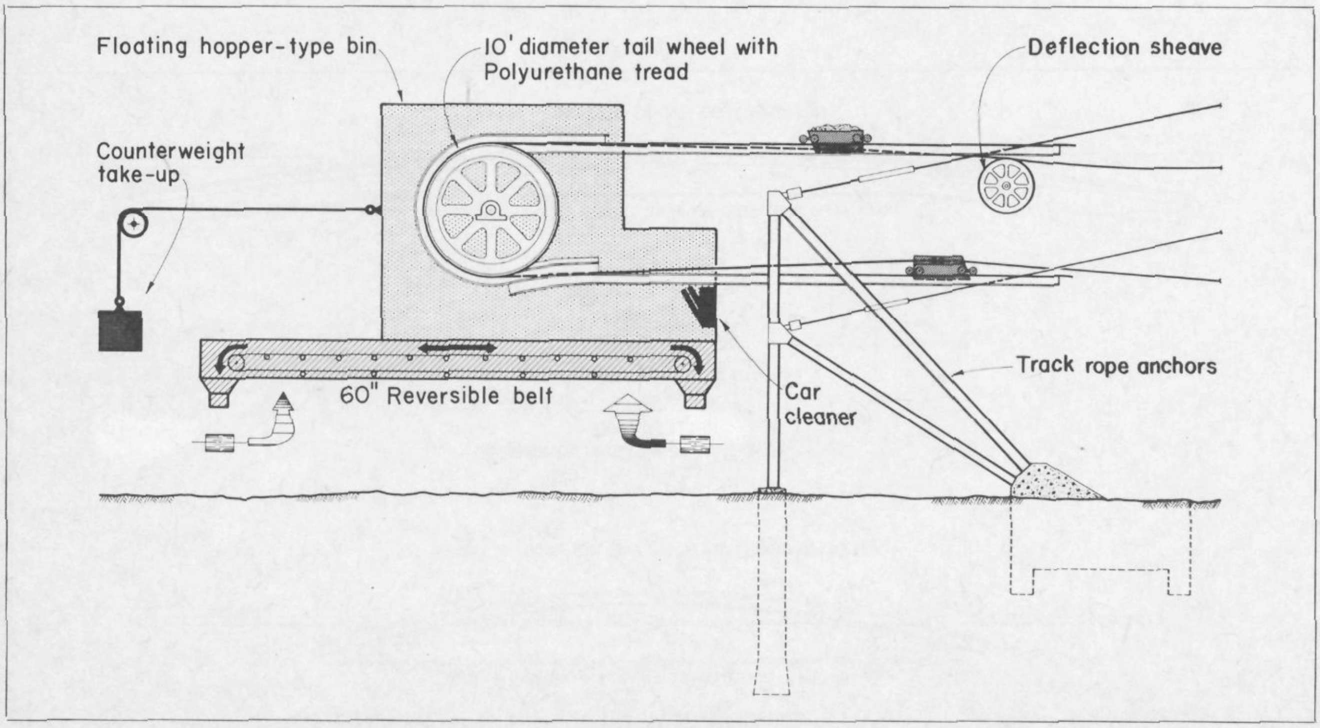


FIGURE 10 — Tramline discharge terminal.

track ropes. Haul-rope lubrication is provided by drip feeders at the loading and discharge terminals. Periodic tightening of bolts in structures and constant checking of the track ropes is also achieved during the running cycle.

Perhaps the most time-consuming area of maintenance is that concerned with the track and haulage ropes. As soon as a broken wire is spotted on a track rope, a metal cuff must be fitted before the wire comes out of lock. As may be expected from studying the entry and exit of the car from a tower, the majority of the damage occurs within 20 to 30 ft (6.1 m to 9.15 m) of the tower. To relieve constant wear on the rope in one area, a program of rope slipping is carried out on a scheduled basis. Generally the rope is slipped in its direction of travel, but this is sometimes changed in order to 'bury' the damage within the saddle area. Rope slipping and retensioning now takes about 1½ days/rope, as the rope may be moved 10 ft (3 m) to 12 ft (3.5 m), at 6 inches (15.25 cm) at a time. Tensioning itself is a time-consuming item; this is done at least twice a year, in fall and spring. The tension is adjusted by means of hydraulic jacks to a pre-set value. (Table 4).

However, because of the friction between rope and saddle, the rope must first be broken free, often with the use of a chain hoist. Getting the rope to the right tension and properly balanced with its partner can be a long and frustrating exercise. Although grease fittings are designed into the new type of saddles to help reduce friction, it is very obvious that we must design something to speed up the slipping and tensioning process.

Once a rope develops too many broken strands in an area, it must be replaced. The method employed for replacing the bottom ropes is to first suspend the cars from the top ropes using load binders and chains. A winch, the new rope and the empty drum are then positioned at one end, with a second winch at the other end. The old rope is pulled out and wound directly onto the empty drum. At the same time, a rope from the winch at the fore-end is attached to the old rope and pulled through the towers. Once the old rope is moved out of the way, the end of the new rope is attached to the winch line and pulled back into place. The initial tension of 80,000 lb is put into the new rope by using the winch and an 8:1 block and tackle. Final tensioning is done using hydraulic jacks.

Changing the top ropes requires a different technique. On the first attempt, the cars were pulled back from the section being changed. However, the haul ropes became so tangled that it was later decided to remove all the cars from that section and store them on the ground. This obviously is a time-consuming job. On the next rope change, we plan to install a dummy rope to support the cars during the change. How successful this will be we can only guess — having not yet tried it.

Haul-rope inspection is also done visually, with a close inspection every three weeks when the cars and pins are greased. The replacement program being used for haul ropes is to have the ends cut off, resocketed and tested at about 1.5×10^6 tons and to replace the whole rope at about 3×10^6 tons. As we have only carried 2×10^6 tons on the tramline to date, it is impossible to say if this is the right formula. It is, however, recommended by the manufacturer of the ropes and other operations.

Up until now, all the in-situ rope tests have been visual, but we hope to try a tramcar-mounted electromagnetic unit in the near future. If successful, this would indicate any broken strands inside the rope as well as on the surface.

Other maintenance procedures, such as shimming tower rails to maintain a good rail rope transition or changing the grip units on the drive wheel, are still being done on an as-required basis until we have sufficient information to indicate the best timing. In fact, with the number of modifications now being incorporated, it will probably be at least two more years before we will be able to formulate a reliable scheduled maintenance program.

The Cassiar tramline, with typical mountain scenery in the background.

TABLE 4 — Chart of Tensions for Ropes

	Tension in Spring (Tension in Kips)			Tension in Fall (Tension in Kips)	
Temp. (Deg. Of)	2-¼-in. Top Track	2-in. Bottom Track	Temp. (Deg. Of)	2-¼-in. Top Track	2-in. Bottom Track
−20	190	145	−40	190	145
0	180	138	−20	180	138
20	170	131	0	170	131
40	160	124	20	160	124
60	150	117	40	150	117

Milling

David C. Cook, General Mill Superintendent,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

The milling complex, situated just east of the Cassiar townsite, receives raw ore from the tramline and processes it into 105,000 tons per year of milled asbestos fibre. The ore is passed through a concentrator, where a barren rock fraction is removed. The concentrated ore, representing 70% of the feed, is dried, stored and then milled into nine separate fibre classifications. The finished product is packaged into 100-lb bags and loaded into trucks in one-ton units. New air-handling facilities were added in 1977 to provide additional air for environmental control.

Introduction

THE ASBESTOS ORE, delivered by the tramline at the rate of 300 tons per hour, is received through the discharge terminal into the concentrator building (Figs. 1 and 2) where the processing of the ore into the finished product begins. Milling consists of concentration, drying, storing and the actual separation of the fibre from the rock by means of screening, aspiration and fiberization. The aspirated fibre is cleaned and graded by means of horizontal gyratory screens and rotary dedusters (paddle trommels).

The mill produces nine direct-milling grades, classified according to length and dust content, five of them being produced concurrently. Special fibres, which are dictated by customer preference, number over twenty, and are variations of the nine basic grades. After classification, the fibre is packaged in 100-lb woven polyethylene bags, sewn, palletized and strapped into one-ton packages. The strapped pallets are stored for shipment by truck to Whitehorse, Y.T., or Fort Nelson, B.C.

The mill complex is comprised of several interconnected, but distinct, structures (Fig. 3). The original mill building, with the associated dryer and dry-rock storage structure, was built in 1954 and produced two grades of fibre. It had a daily capacity of 150 tons of ore, which was trucked directly from the mine. The mill was gradually expanded until 1970, when a 132- by 128-ft addition was built. It was then producing a total of eight fibre grades, four concurrently, and handling 3000 tons of ore per day.

The 1970 laminated wood mill building was designed to:

(1) obtain improvements in packaging and material handling;
(2) improve the production of spinning and cement grades;
(3) allow the production of an additional fibre grade;
(4) improve environmental conditions.

The mill air building was built in 1977 to improve environmental conditions.

Milling Principles

Recovery of asbestos fibre is based on the physical differences between the fibre and the host serpentine rock. The fibre and rock are chemically alike and, in the rock state as mined, of equal specific gravity. Only by making use of the fibrous nature of the asbestos, through the process of fiberization to change its apparent specific gravity, can it be effectively separated from the host rock.

The value of the fibre is primarily dependent upon length (Fig. 4), so the milling process is designed to separate the fibre from the rock as early as possible in the circuit to prevent fibre degradation.

In order to maintain fibre quality, care must be taken through the mining and milling operation to preserve fibre length and to create a minimum amount of damage and dust impregnation in the fibre. Long spinning fibre is the highest quality and is released early in the milling process; therefore, the ore must be treated carefully until the aspiration from the initial mill screens. This is achieved by minimal handling, gentle crushing, and drying at controlled temperatures and moisture content.

As all the separation is physical, contamination by any fibrous materials, such as wood, paper or other light material, which could be aspirated with the fibre, has to be eliminated at the source. The ore delivered to the mill is passed over large-mesh screens to remove the longer fibre before any fiberization. All fibre is aspirated through hoods to cyclone collectors. The cyclones make use of centrifugal force to separate the fibre from the aspirated air. The fibre is discharged through rotary valves to horizontal cleaning screens, where the fine rocks are dropped out. Fibre is further cleaned and classified with the selection of screen meshes on horizontal gyratory and horizontal cylindrical screens equipped with counter-rotating beaters. These trommel dedusters are under negative pressure and the fine dust, released by the action of the beaters, is removed to the dust system.

Final aspiration is through air classifiers in which the last traces of fine rock particles still remaining in the fibre are removed.

Fibre grade is controlled by ore selection in the pit, blending from storage, and circuit and screen mesh changes.

David C. Cook was born in Newcastle, New Brunswick, attended Mount Allison University and graduated from Queen's University, Kingston, Ontario, in mining engineering. He worked at Sigma Gold Mines Ltd. and Algoma Ore Properties. In 1951, he joined Canadian Johns-Manville Co. Ltd. as mining engineer and worked in various positions at Asbestos, Quebec, and Matheson, Ontario. He joined Cassiar in 1963 as assistant mill superintendent, then served as mine manager with Indusmin Limited at Nephton, Ontario. In 1973, he joined United Asbestos as mine manager.

Mr. Cook returned to Cassiar in 1977 as special project manager and is now general mill superintendent. He is a CIM Member and a member of the Association of Professional Engineers of Ontario.

Keywords: Cassiar Mine, Asbestos, Milling, Concentrators, Dryers, Ventilation, Tailings disposal.

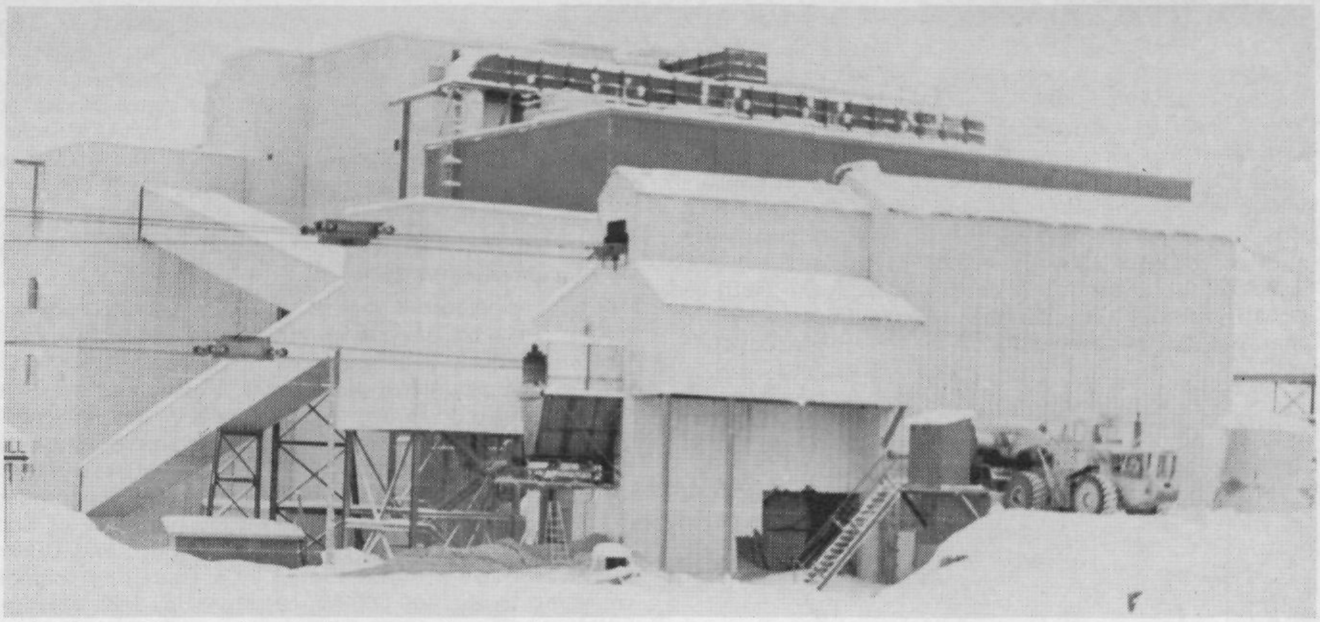


FIGURE 1 — The tramline discharge terminal at the concentrator.

In all stages of milling, dust control is a major consideration, with the use of enclosures and maintenance of negative pressures within the enclosures through the dust collection system.

All processes — drying, ventilation, dust control, aspiration, conveying, separation and fibre cleaning — require large volumes of air, a major consideration in an asbestos milling operation.

Concentrator and Dryer Circuit (Fig. 5)

The tramline buckets which deliver the ore from the mine drop it onto a 60-in. reversible feed belt which forms part of the tramline discharge hopper. This belt normally discharges the ore to the 36-in. concentrator feed belt, past a magnetic conveyor, metal detector and weightometer, to a 6- by 12-ft single-deck Tyrock screen fitted with a 1¼-in. by 4-in. opening. The undersize from the Tyrock screen is discharged to a 30-in. belt feeding the dryer circuit. The oversize is sent to an APK60 Hazemag impact crusher, dis-

FIGURE 2 — The Cassiar mill buildings.

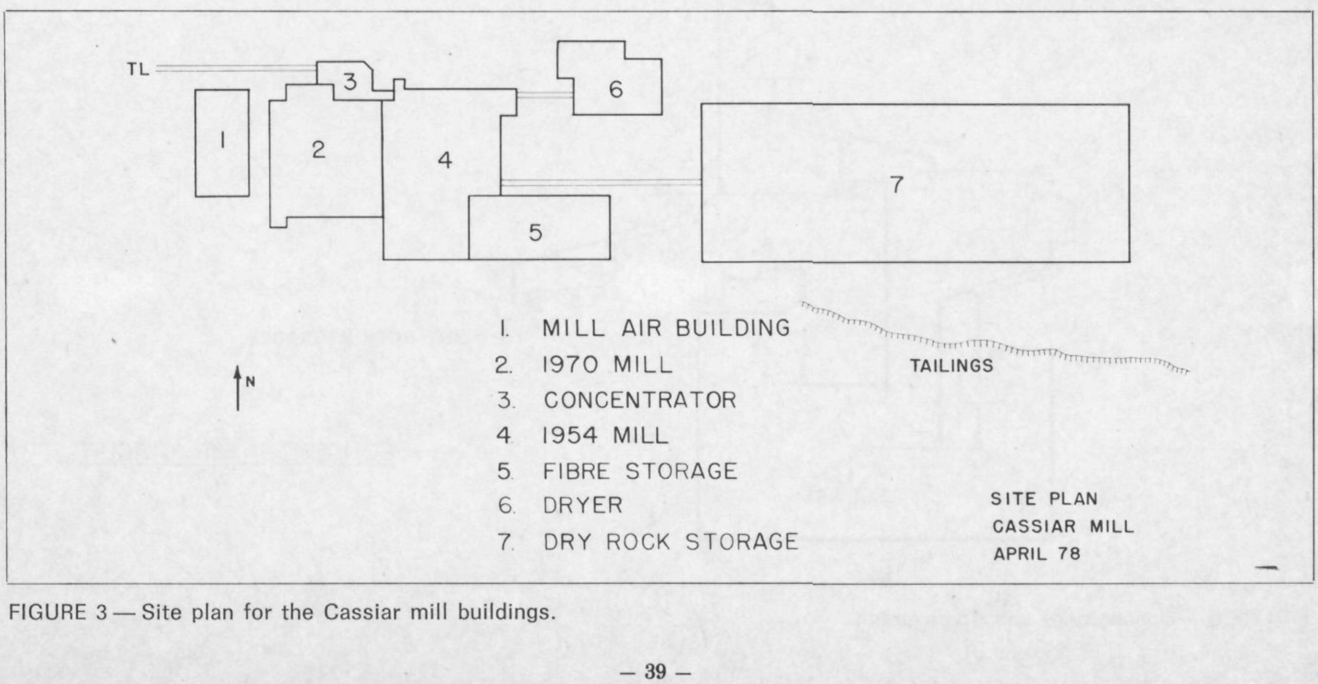


FIGURE 3 — Site plan for the Cassiar mill buildings.

charging to a second 6-ft by 12-ft Tyrock screen fitted with a 1-in. slotted punched plate deck. The undersize of the secondary screens goes to the dryer circuit; the oversize, consisting of primarily barren rock, is transported to an APK50 Hazemag which discharges to a third Tyrock screen. The undersize is passed over three push-pull-type fluid-bed classifiers in parallel, which remove the fibre released by the Hazemags. The middle size of the tertiary screen may, by means of a splitter, be sent directly either to tailings or, if it contains sufficient fibre, be passed over the fluid-bed classifiers with the undersize. Approximately 30% of the feed to the concentrator is rejected as oversize from the tertiary screen and fluid-bed classifiers. The fibre from the classifiers is aspirated to a 9-ft-diameter cyclone, which discharges to the dryer feed belt. Should any problem occur within the concentrator or dryer circuit, the 60-in. belt feeder, through interlocks, discharges to an outside stockpile by means of a stacker conveyor, thus maintaining tramline delivery.

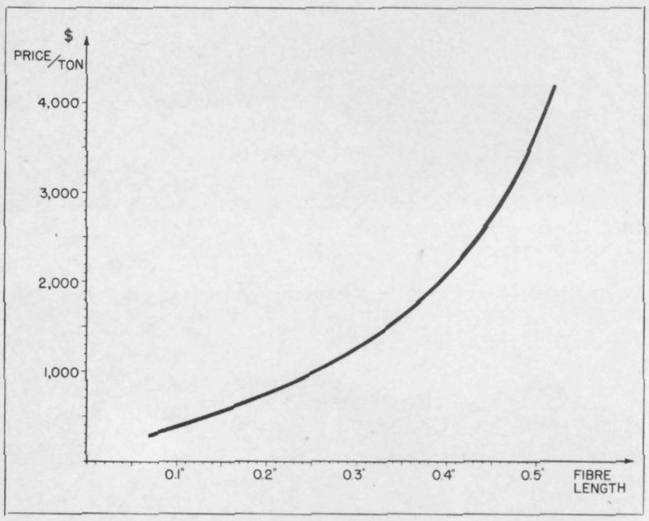


FIGURE 4 — Relationship of fibre length to market value per ton.

The stockpile is reclaimed with a 6-yd front-end loader feeding a grizzly and vibrating feeder to the 36-in. concentrator feed belt.

The concentrated product, representing approximately 70% of the concentrated feed, is passed over two single-deck Dillon screens in parallel, fitted with slotted ¾-in. punched plate decks. The screen oversize is sent directly to conveyors feeding the dry rock storage. The undersize is conveyed to one or more of three oil-fired horizontal rotary dryers. The ore, which is received at up to 12% moisture, is dried to under 2½% moisture at an average oil consumption of one gallon per ton of dryer feed. The dried ore and Dillon screen oversize are distributed by a travelling tripper conveyor to the 100,000-ton-capacity dry-rock storage building. The building is an inverted Bailey truss structure, 480 ft (146 m) long, 150 ft (45.7 m) wide and 74 ft (22.5 m) high. The slope of the trusses is 41 degrees, which conforms to the angle of repose of the ore pile.

The conveying system feeding the dryers and dry-rock storage is part of the original plant and its designed capacity has now been exceeded. It is planned to replace all conveyors between the concentrator and the mill with a new circuit.

Reclaiming from dry-rock storage is accomplished with a 6-yd front-end loader dumping into drawpoints located on the centre-line of the building above the re-

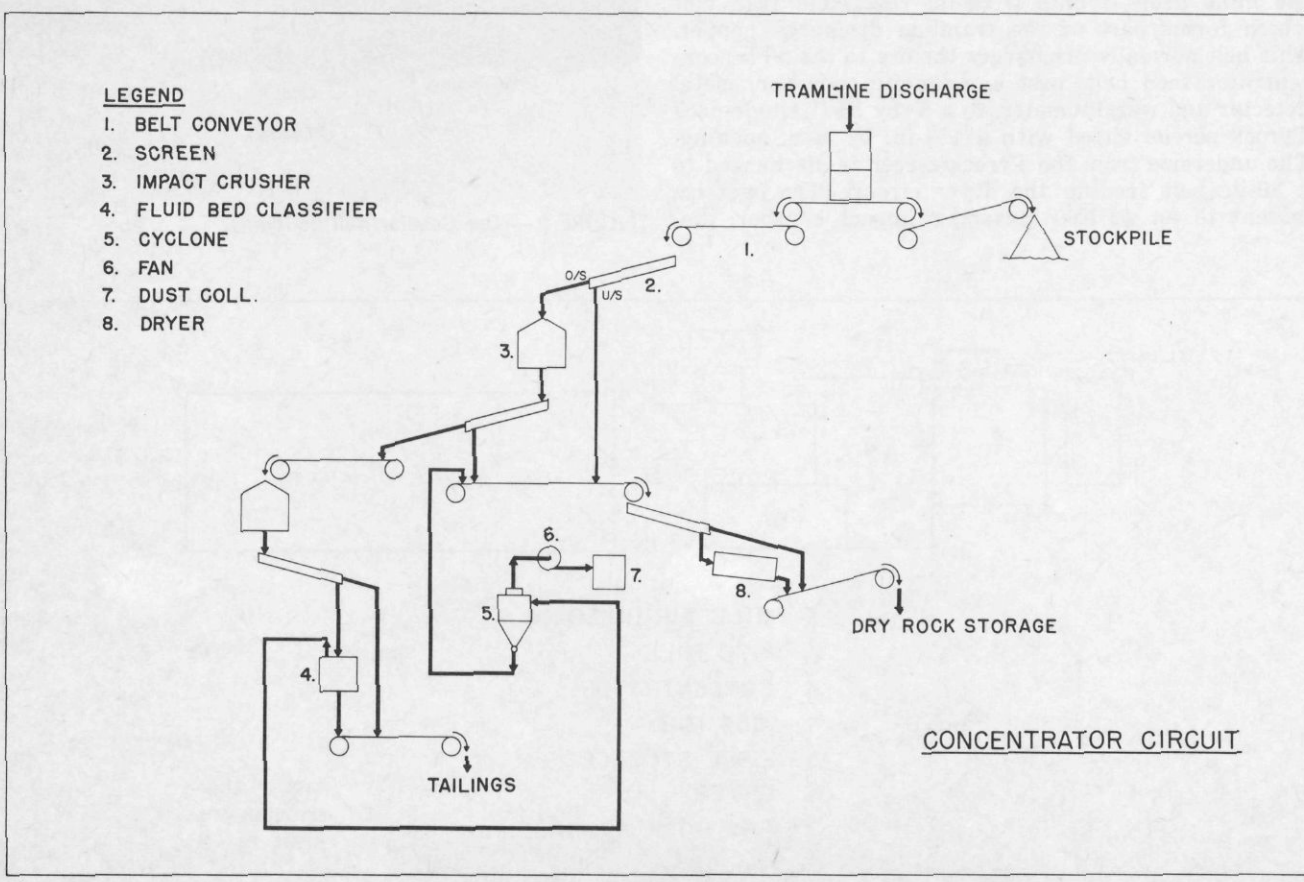


FIGURE 5 — Concentrator and dryer circuit.

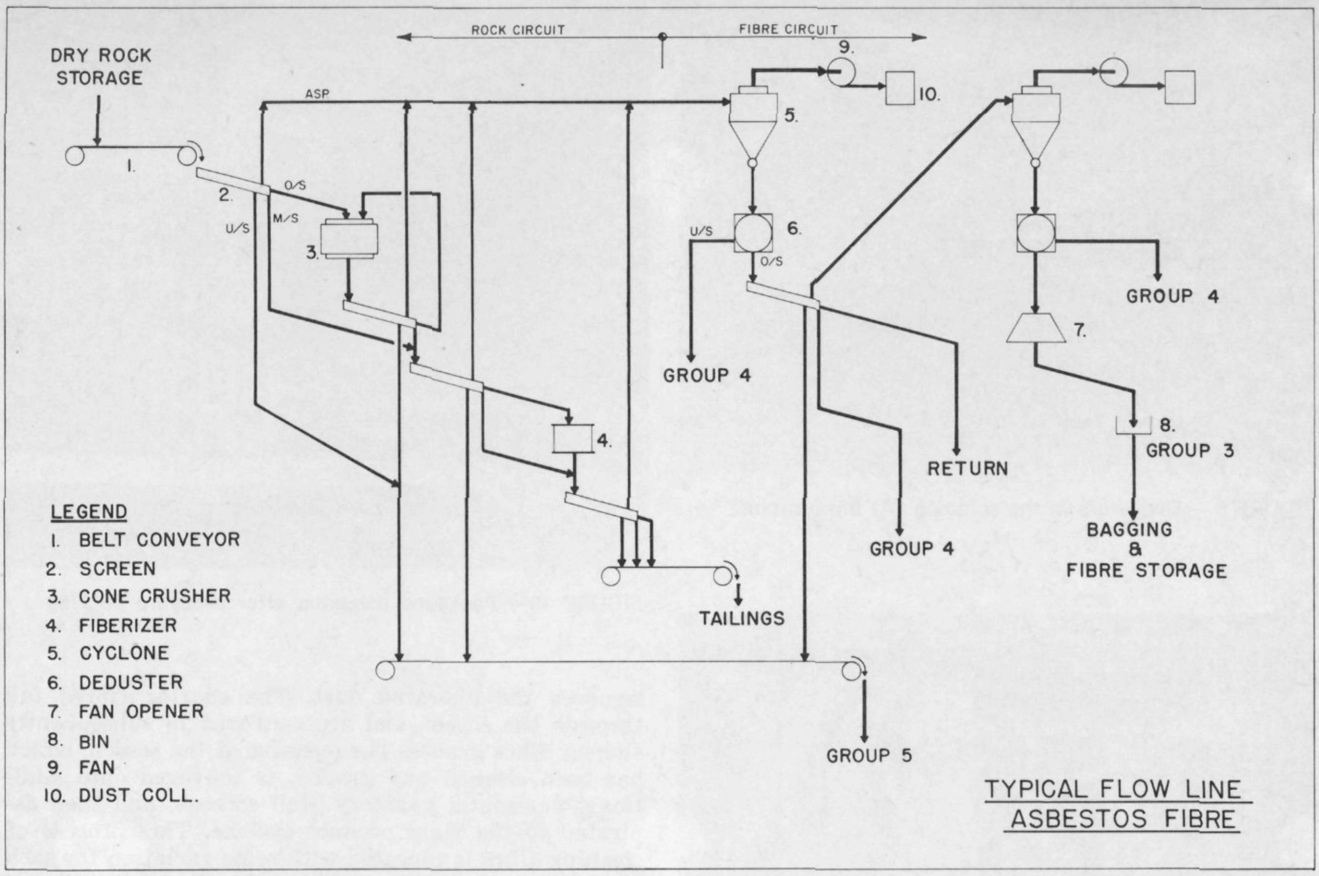


FIGURE 6 — Typical flowsheet for asbestos fibre at Cassiar.

claim tunnel, which runs the full length of the dry-rock storage. These drawpoints discharge onto variable-speed belt feeders, which feed the reclaim belt.

Milling Circuit (Fig. 6)

The reclaim belt from dry-rock storage discharges into two rotary distributors, each of which feed three horizontal gyratory double-deck Hall screens. These primary mill screens are fitted with a two-mesh top deck screen and an 18-mesh screen on the bottom deck. The first fibre is lifted from the top deck at the discharge end by air through an aspirating hood (Fig. 7) and is conveyed through ducting to a 7-ft-diameter cyclone. This first aspiration is designed to pick up all of the released fibre, thus preventing damage by further crushing. To ensure aspiration of all the fibre, a certain amount of rock is necessarily lifted at the same time and this rock is eliminated in subsequent screening and aspiration stages.

The cyclone collector is designed to separate the fibre from the air by means of centrifugal action. The fibre drops through the cone of the cyclone and is discharged through a six-bladed rotary airlock. The air and dust are vented through the central core of the cyclone to a fan which discharges to a fabric dust filter. The cyclone is not designed to be highly efficient, as it also acts as a fibre cleaner, removing some of the finer dust, thus improving the fibre quality during the air-conveying cycle.

The oversize rock from the primary screens is crushed in two 3-ft short-head cone crushers in parallel. The cone crusher product is elevated to four additional screens in closed circuit with the crushers. These screens are also fitted with a two-mesh screen panel

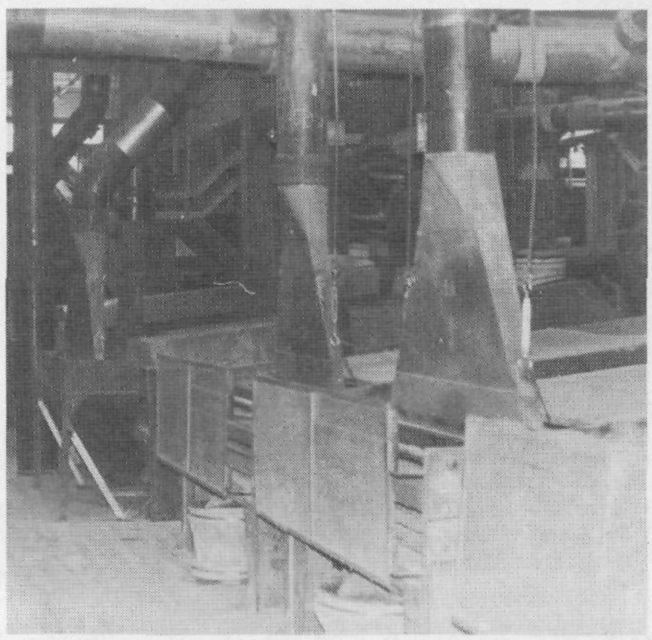


FIGURE 7 — Primary mill screens with aspirating hoods.

and the aspirated fibre from these screens is conveyed to similar cyclones in the initial fibre circuit. The fibre lifted from the two-mesh screen decks is directed to the group three or spinning fibre cleaning circuit.

The middle size of both the primary and secondary screens discharges onto six screens fitted with a top screen deck of four mesh. Aspiration from these

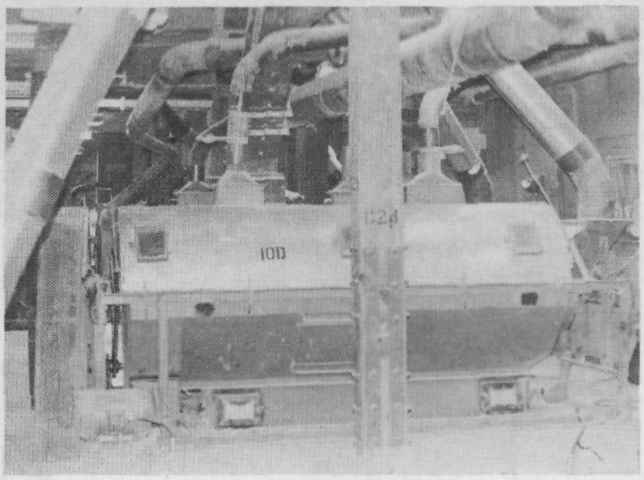


FIGURE 8 — Dedusters in the spinning (A) fibre circuit.

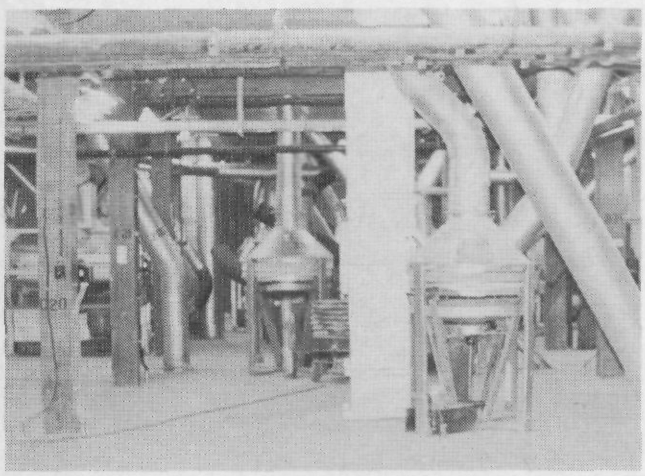


FIGURE 9 — Bauer air classifiers in the cement (AK) fibre circuit.

screens is the first lift into the cement fibre grade. All screens in the rock circuit are double decked, with aspiration from the top deck. The oversize from the third set of screens after aspiration goes to three vertical impact mills called 'fiberizers'. These mills receive rock by gravity, and hammers attached to a vertical shaft throw the rock against serrated liners of manganese steel. The impact releases fibre from the host rock, which is aspirated from the screen receiving the fiberizer discharge. The process of screening, aspiration and fiberization is carried on through two additional stages in the primary rock circuit on progressively finer mesh screens.

The aspirated fibre from the rock circuit is discharged from the cyclone to cleaning screens. A typical flowline would have the cyclone discharging to a series of rotary paddle trommels, commonly known as dedusters. These machines consist of a horizontal cylindrical screen, 36 inches in diameter and approximately 9 ft long, in a dust-tight enclosure and under negative pressure.

The fibre is introduced into one end of the horizontal revolving screen (Fig. 8) and is subject to the action of paddle arms rotating in the opposite direction. The paddles beat the fibre against the screen and dust aspiration from the top of the enclosure

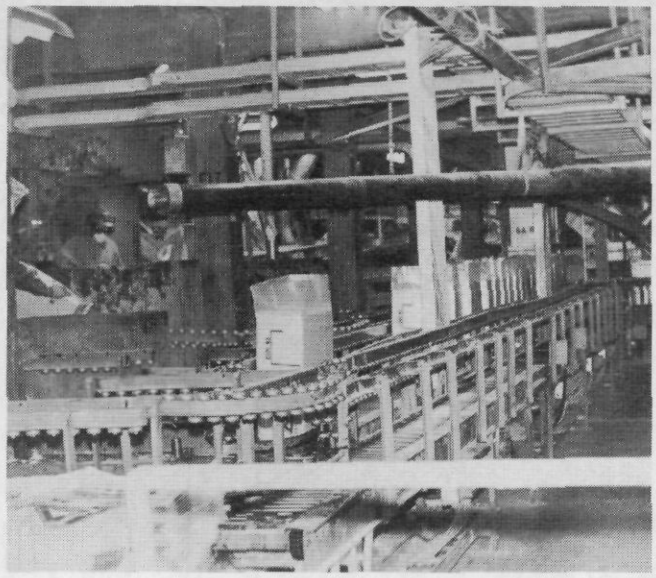


FIGURE 10 — Packaged asbestos after pressure packing.

removes the liberated dust. The shorter fibres fall through the screen and are conveyed to subsequently shorter fibre grades. The oversize of the screen, which has been cleaned and graded, is conveyed onto additional horizontal gyratory Hall screens, and then aspirated to the final product cyclone. This process of cleaning fibre is repeated with some variation for each of the five basic fibre grades. In all grades, except the spinning fibre, the final aspiration is through air classifiers of one type or another (Fig. 9). These classifiers complete the removal of the fine grit which has not been cleaned in the screening process.

The fibre aspirated to the final product collector is discharged into a 6-ton live-bottom bin, which is used for final blending before bagging. The live-bottom bins discharge via screws to weigh hoppers, where the fibre is weighed into 100-lb portions prior to delivery to the bagging machines. Fibre, which has an approximate density of 5 to 10 lb per cu. ft at this stage, is discharged into a vertical chamber, where it is compressed by a hydraulic plunger or ram. The pressure-packed fibre, which then has a density of approximately 40 lb per cu. ft, is ejected through a horizontal spout into a woven polyethylene bag, the approximate package size being 24 by 16 by 8 in. The actual size of the package varies according to the grade and openness of the fibre. The fibre package is then sewn, sorted into various grades, palletized into one-ton lots, strapped and sent to fibre storage to await test results prior to shipment (Figs. 10 and 11).

Specialty Grades

Fibre as milled has a specific surface area of approximately 4000 cm^2 per gram. Manufacturing processes in both spinning and cement fibre plants require fibre openness from 4000 to 10,000 cm^2 per gram. Within the milling circuit, facilities are provided on a small scale to treat a portion of the basic fibre grades to produce a more highly opened specialty grade. This opening process is achieved by passing the fibre through high-speed, specially designed impact fans. The opening is done only after fibre specifications have been met on basic grade, preserving the fibre quality and preventing degradation by inclusion of rock and dust particles.

Quality Control

With the highly competitive nature of today's market and its demand for high standards of quality and uniformity, it becomes quality control's function to ensure that all fibre shipments are within the narrow limits specified for all grades produced. A close continuous liaison among mine, mill and quality control is required to ensure that properly blended ore is delivered to meet the demands of the production schedule. Quality control must also maintain a relationship with the sales department to assist in customer problems, which may include fibre quality, bag weights and packaging.

During the course of its work, quality control uses many different tests and procedures in both wet and dry tests, such as Bauer McNett, T & N Classifier, filtration rate, wet volume and buoyancy, surface area, Quebec Standard Test, dry bulk, colour, magnetic, Ro-tap, grit determination and the modified Suter Webb test.

To maintain fibre quality, regular samples are taken of both the finished product and the fibre at various stages of the milling process. The maintenance of uniformity in fibre quality is of prime importance, and this is maintained by setting all fibre specifications within narrow limits demanded for maximum and minimum shipping standards.

Fibre samples are taken immediately before the bagging operation on each grade. These samples are composited and tested for various fibre qualities. The fibre is tested for dry screen analysis, wet screen analysis, fibre length, -200 mesh content and surface area. In the spinning grades, length of fibre is an important characteristic and a small sample is taken and physically combed on the modified Suter Webb, with the fibres separated in various lengths. The dry tests, although more relative than absolute, are used as a quick test of fibre quality and production control. Wet screen analysis, which is the better evaluation of fibre length and dust content, is a longer process. For cement fibre grade, the more important fibre qualities are strength, fibre length, filtering rate and dust content. Filtration rates are important, because fast-filtering fibre will allow faster production in the customers' plants.

Facilities exist at the plant to fabricate asbestos cement plaques, which are used to evaluate the action of Cassiar fibre in an asbestos cement product. Basic research has been done in this field to determine the optimum condition for the various cement fibres to yield ultimate strength.

Air-Handling System

The total air handled in the milling process is over 660,600 cfm, representing 80 tons of air for each ton of fibre produced. Approximately 80% of the air is used for processing; 20% is used for environmental dust control.

CONCENTRATOR AND DRYER CIRCUIT

Process and hygiene air in the concentrator and dryer circuit is generated by seven individual fans aspirating from seven individual pulse-type fabric filter collectors. The fans are located on the clean-air side of the dust collectors. All transfer points and screen enclosures are fitted with dust take-offs having a pick-up velocity of 300 ft (91.5 m) per minute.

Air, used in the drying circuit both for drying the ore and dust control, is drawn through three pulse-type dust collectors equipped with Nomex filter fabrics. This allows the operating temperature to exceed 400°F. The bag collectors are insulated to maintain the temperature above the dew point. The total air handled in the dryer and concentrator circuits is 160,000 cfm.

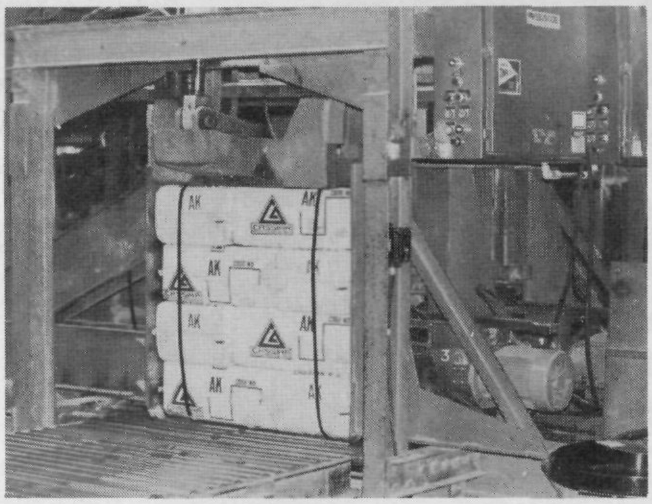


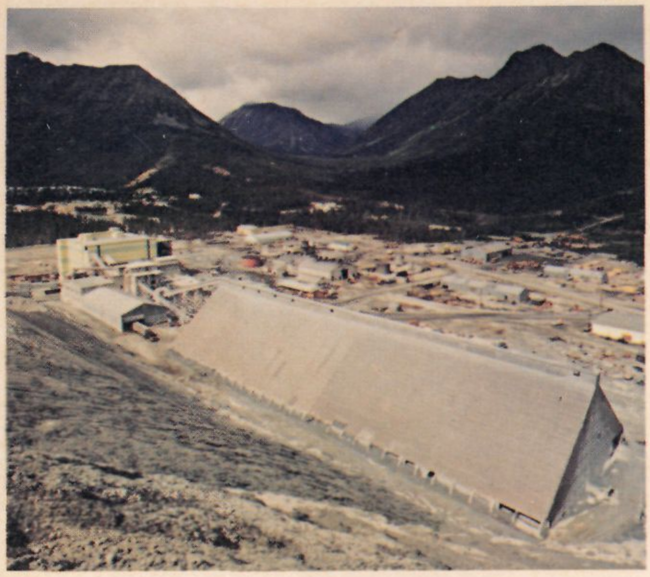
FIGURE 11 — Strapping machine with a one-ton unit of cement (AK) fibre.

MILL CIRCUIT

More than 500,000 cfm of air is required for the mill operation. This air is generated by approximately 40 fans pulling air from individual cyclones assisted by five booster fans to maintain a negative pressure inside the gathering plenums. The system now used at Cassiar is designated as a push-pull air-handling system. The individual fans create a negative pressure from the pick-up point to the inlet of the fan. From this point, the fan pushes air toward booster fans which are operating at the end of major plenums. The inlet to the booster fans may be slightly negative or positive, depending on air conditions and conditions in the dust filter. The booster fans push air at a positive pressure of up to 6-in. water gauge through fabric filters located in the 1954 mill ground floor; air must find its own way through various openings to the aspirating point.

Modifications to the original fabric air filter system installed in 1956 have permitted an increase in its capacity from 230,000 to the present 500,000 cfm. This has been achieved by additional units, fabric selection and increased cloth area with the use of smaller-diameter filter tubes. Increased air-to-cloth ratio has resulted in a high pressure drop across the filtering units.

In 1975, additional air was required to improve product transport and to meet tighter environmental regulations in regard to air-borne asbestos dust. Studies were made of various alternatives, with the most obvious being expansion of the present system. This was ruled out on environmental considerations due to the location of the fabric filter on the bottom floor of the original mill. In most mills the mill air is recirculated, and the air returning to the top floor of the 1970 mill has to pass all floors, where contamination could be picked up and dispersed over a wide area. Velocity in stairways is excessive. There is difficulty in balancing the fan system with different fan characteristics. With the pressure system, any circuit that is required to be isolated for the purpose of repairing



The Cassiar Asbestos mill, with the mountains in the background.

Truck-loads of packaged asbestos fibre leaving the plantsite.

ducts is affected by pressure from other fans in the system, which cause major dust dispersions. When mill fans are shut down for weekly maintenance, the push system discharges dust into the air through all openings.

The second alternative was to install a pull-through system. Instead of blowing dusty air through the filter, fans are used to draw clean air through the filter fabric. It was desirable to install fan plenums and dust collector units at the top of the mill, as they would be closer to the 7th-floor cyclones, which handle over 85% of the mill air. However, because it is a timber structure, the 1970 mill could not support the extra weight. A separate building was constructed in 1977 to supply 480,000 cfm for process and environmental air in the 1970 mill. The adjacent location added an approximate 2-in. loss to the proposed system. An additional floor was added to the 1970 mill to house the air plenums collecting air from the cyclones and dust risers. Three plenums feed an air manifold, which distributes the dusty air to eight 54-in.-diameter downcomers to the mill air building. Dust-laden air passes through two settling chambers before being discharged to nine hoppers under the automatic shaker-type bag filter. The dust collected from the bags falls into the hopper, from which it is then conveyed by screw conveyors and belts to a pug mill, where it is wetted and discharged to tailings. With the high settlement rate in the plenums, grain loadings of two grains per cu. ft are expected — or 2 tons per hour of collected dust. The air is generated by four Chicago blower fans of 120,000-cfm capacity each, operated at a negative pressure of 16 in. water gauge. The clean air is distributed through the fans into a vertical clean air plenum, where it is returned through louvered opening to each floor or discharged to the atmosphere as desired. The ground floor of the mill air building houses the mill maintenance shop. The second floor contains the quality control laboratory and the mill offices.

Mill Modifications

With the connection of the new mill air building, additional air is available for processing and for environmental dust control. Mill circuits in the 1970 mill were redesigned to make use of the additional air to improve working conditions. The ducts and aspirating lines were rerouted to eliminate long horizontal runs and permit additional dust take-offs. Dust take-offs have been revised to reduce pick-up velocities to 300 to 400 ft per minute. This aids in keeping transport velocities low and reduces loss of fibre values within the system.

Leaks in ducts and chutes are a major source of the environmental contamination within the mill. High conveying velocities in ducts of the aspirating and dust collection system cause high wear and result in short duct life. Various liners, including Ni-hard and rubber, have been used to overcome these problems. Recent experience has shown the superiority of ceramic liners, and new installations are being equipped with these liners at elbows and transition areas. Screen enclosures have been modified based on operating experience at Cassiar and other asbestos mines. These modifications are designed to more effectively contain the dust normally associated with the gyrating screens at both feed and discharge points. Similar modifications have been designed on inspection door and conveyor enclosures.

In order to maintain an effective dust system, long horizontal lines have been avoided. During 1977, eight vertical dust risers were located in close proximity to dust sources to reduce the horizontal distances. These risers extend from the ground floor to the dusty air plenums on the 8th floor.

Tailings Disposal

As 90% of the mill feed as well as 30% of the concentrator feed is discarded as waste, material handling of dry, finely ground rock is another major environmental problem. Various methods have geen tried to wet the tailings for dust control.

The only successful treatment to date has been to collect all the waste components in one bin and treat them at a controlled rate through a pug mill fitted with water sprays. Some problems are encountered with the automation of sprays, which causes sticking and freezing to the exposed conveyor belt.

The milling process at Cassiar has, within the last two years, undergone major redesign and modification of machinery and processes. The improvements in environmental control and production facilities will maintain and improve Cassiar's position in the supply of quality asbestos fibre.

Projected improvements and capital expenditures over the next three years are designed for environmental improvement, increased recovery from the mill feed, and improvement of fibre quality and customer service.

Maintenance

Peter C. Jones, Plant Superintendent,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

The paper gives a broad view of the maintenance procedures at the Cassiar Asbestos Corporation property. Comparisons are drawn to indicate the changes in facilities and approach that have evolved over the last 25 years while coping with the northern British Columbia climatic conditions. Special emphasis has been placed on the importance of the support groups providing services for both the mine site and the town.

Maintenance

THE STEADY GROWTH OF CASSIAR over the last 25 years has left us with a mixture of old and new. The shops are widely spaced out, which leads to supervisory problems. Add to this the normal problems of supply delivery and shortage of skilled manpower and one can imagine the problems that supervisors are faced with in order to maintain continuity. The supervisory force is strong, as indicated by the organization chart (Fig. 1).

The snow-capped peaks provide a scenic background for these large haulage trucks.

Keywords: Cassiar Mine, Asbestos, Maintenance, Concentrators, Power plants, Electrical systems, Plumbing.

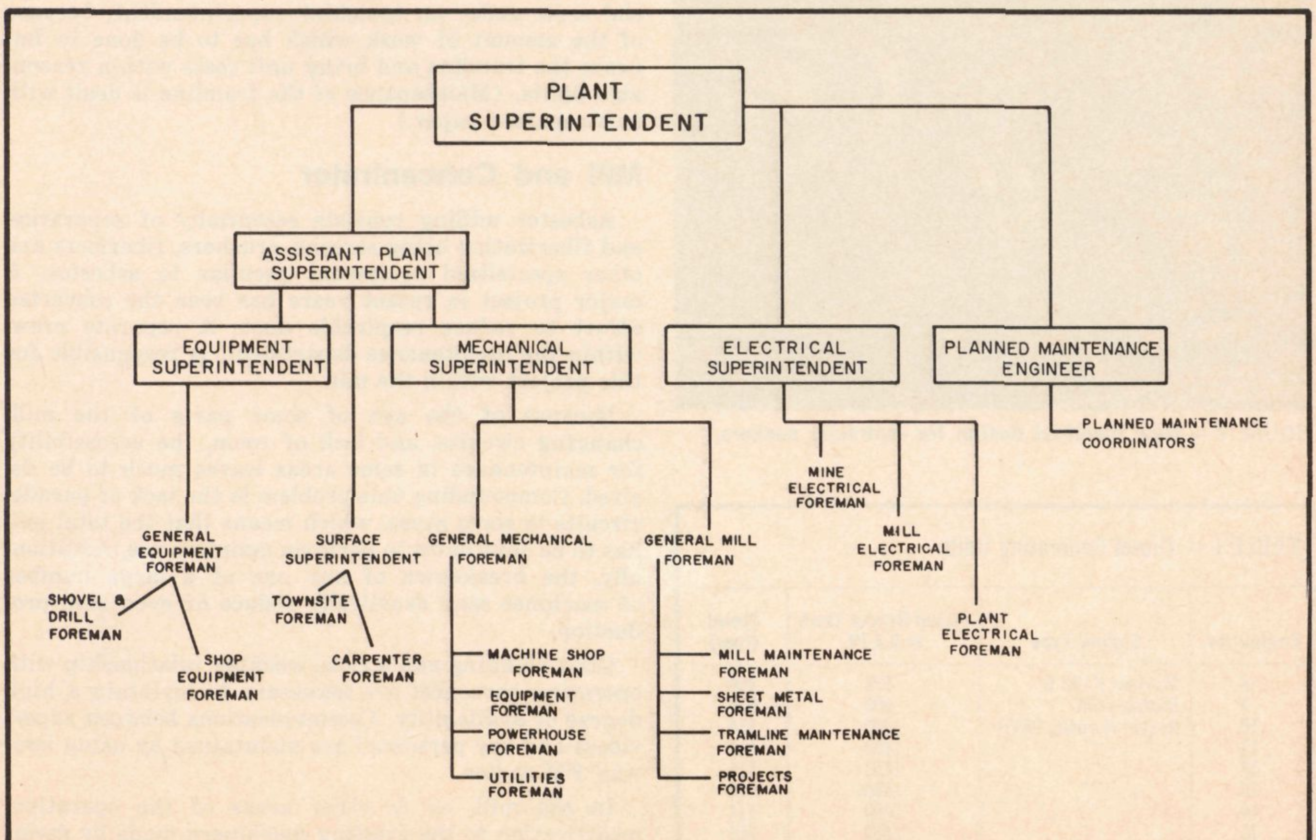


FIGURE 1 — Organization chart at Cassiar.

The Mine

The mine mechanical department is responsible for the maintenance of 19 trucks in the 50- to 85-ton class, two 11-cu.-yd shovels, and two electric and two diesel-driven blast hole drills. Until 1974, the garage consisted of a 32-ft (9.1-m) by 40-ft (12.1-m) building in the mine area. Most maintenance was done in the field; consequently, there was poor maintenance in cold, especially 40°C below, weather.

In 1974, a new two-bay garage was opened and two years later was expanded to a five-bay unit with a separate steam bay and welding area. The garage is fitted with a 10-ton crane, centralized lubrication system for the service bay and underfloor, and infra-red heating systems to give better working conditions. In 1977, a new lunchroom, changeroom and office were constructed on a mezzanine floor.

In-field servicing is done using a specially fitted lube van complete with hose reels, compressor, etc. (Fig. 2).

The graders, front-end loaders and bulldozers are maintained under a separate lease and maintenance contract with Finning Tractor and Equipment Co. Ltd., Caterpillar distributor for British Columbia and Yukon.

Winter conditions dictate that major work on shovels and drills be accomplished between June and September. As can be imagined, the intense cold weather provides its own problems. Freezing of water and air lines, brakes, etc., are frequent problems, especially if a heater fails or if there is a power outage. Trucks are left with their engines running when not being driven to keep them warm.

The better facilities have naturally led to greater efficiency, but an added help has been the many innovative and progressive ideas put into practice by Cassiar's employees. These range from the centralized lubrication and transmission oil empty-and-fill fittings on each truck to the modified hydraulic deck drive on a hydraulic-electric drill.

FIGURE 2 — Field service vehicle.

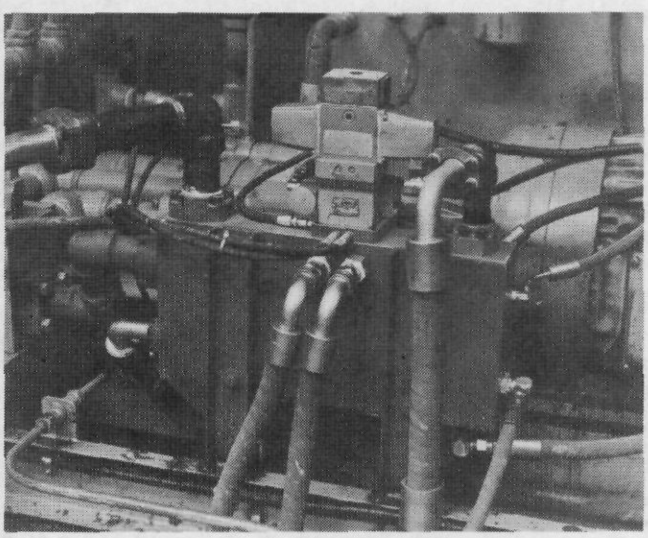


FIGURE 3 — New manifold design for hydraulic packers.

TABLE 1 — Diesel Generating Units

Engine No.	Engine Type	Rated Output (kw) @ 0.8 Pf	Speed (rpm)
8	Mirrless K1SS 5	972	450
9	Ruston 8ATC	900	514
10	Ruston Hornsby 9ATC	1400	514
11	"	1400	514
12	"	1400	514
13	"	1400	514
14	"	1400	514
15	"	1400	514
16	"	1400	514

Crusher and Tramline

The crusher and tramline are maintained by a special crew under an assistant superintendent, because of the amount of work which has to be done to improve the tramline and bring unit costs within reasonable limits. (Maintenance of the tramline is dealt with in a separate paper.)

Mill and Concentrator

Asbestos milling consists essentially of separation and fiberization using screens, crushers, fiberizers and other specialized equipment peculiar to asbestos. A major project in recent years has been the concerted effort to reduce respirable dust. A separate crew, within the maintenance department, is responsible for this project within the mill.

Because of the age of some parts of the mill, changing circuits, and lack of room, the accessibility for maintenance in some areas leaves much to be desired. Compounding this problem is the lack of parallel circuits in some areas, which means that the total mill has to be shut down to perform maintenance. Additionally, the breakdown of any one of a large number of machines may drastically reduce or even stop production.

Good planning and a close working relationship with operating personnel are necessary to maintain a high degree of availability. Communications between supervisors and key personnel are maintained by using two-way FM radios.

In the mill, as in other areas of the operation, modification to the existing machinery made by maintenance and operating personnel is helping to increase availability and efficiency.

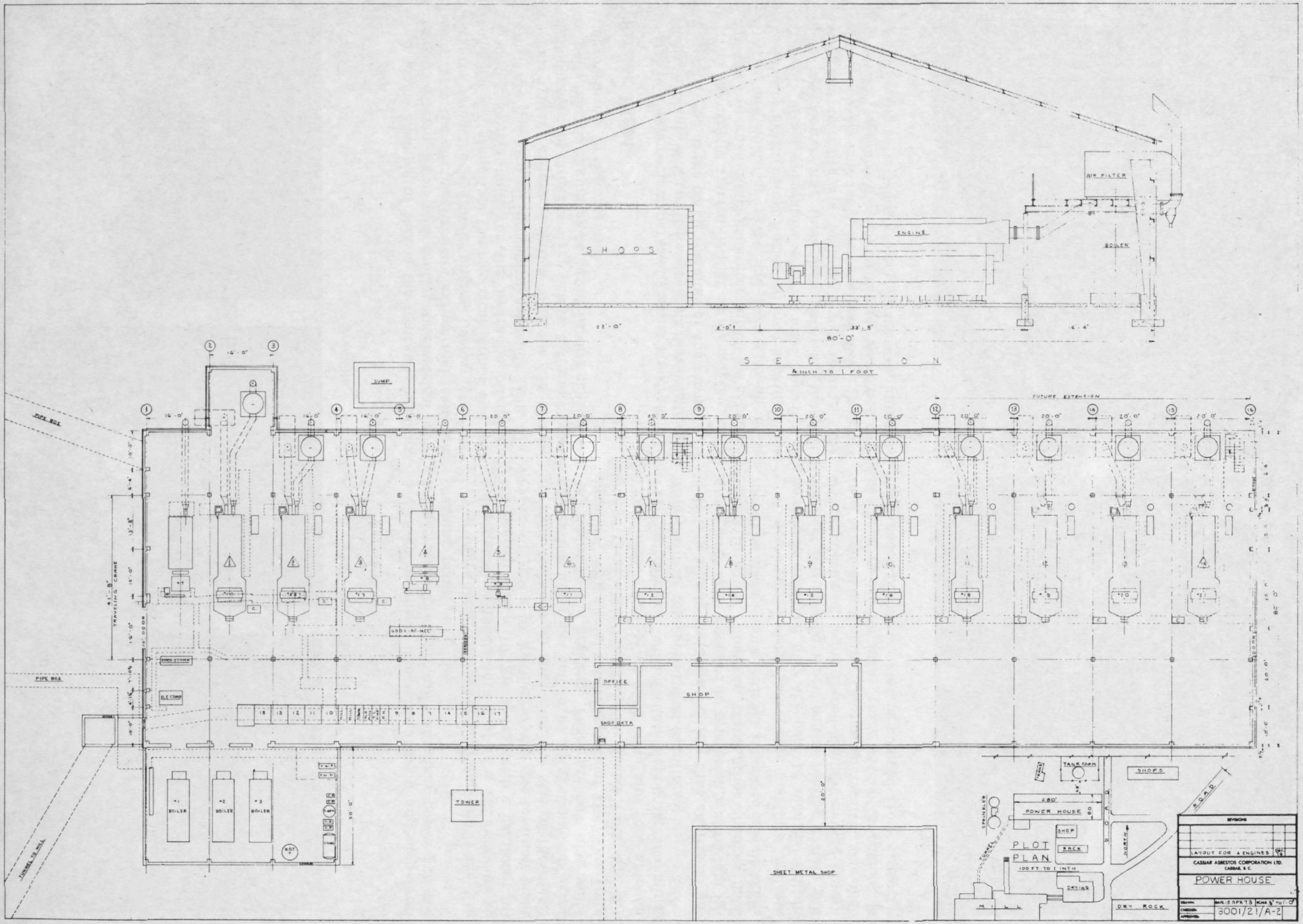
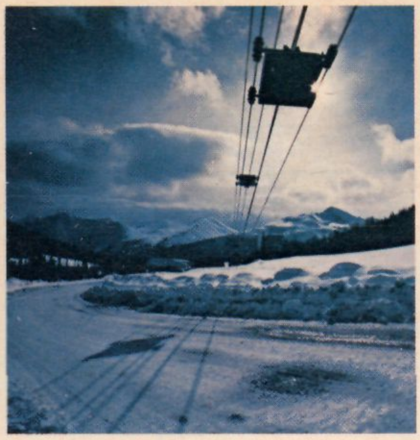


FIGURE 4 — Powerhouse layout.

Trucks



Tramline

Shovels

A prime example of this is the hydraulic pressure packing system. Faced with continuous electrical problems from the electrical control system and leaks and failures from the hydraulic piping, Cassiar staff redesigned and simplified the electrical circuit and had a solid manifold built for the hydraulic circuit on which are mounted all the control solenoids (Fig. 3).

This type of change, together with the use of new lining materials in heavy wear points, different types of belts, etc., is beginning to result in very noticeable improvements in the operation.

Power Plant

One of the important parts of Cassiar's operation is its powerplant. Without it we would have no electricity or steam heat in the plant, the mine or the townsite.

The powerplant started in 1952 as a small log cabin housing two small diesel generators of 50-kw capacity. By the mid 60's, the plant had grown to an 80-ft (24.4-m) by 82-ft (25-m) Butler building housing eight 2300-volt diesel generators with a total capacity of 3673 kw. Power was then distributed to the mill, plant and townsite via four 2300-volt feeders. The mining shovels and drills, being either diesel powered or having separate diesel generators, were not connected to the powerplant system. There was, however, a 13.8-kv overhead line with step-up and step-down transformers to supply mine auxiliary power and the crushing plant.

Ten years later, the power consumption at the site has risen from 1964's output of 14 x 10^6 kw hours to the present 43.3 x 10^6 kw hours, with a projected increase of 12.9 x 10^6 kw hours/year due to a new plant in 1978. To keep up with this, the size of the diesel generating units has had to be changed and an extra bay added to the building. There are now nine units installed, with a total rated capacity of 11,672 kw. Only one of these dates back to 1964. (Table 1 and Fig. 4.)

All units generate at 2300 volts and feed onto an ITE switchboard, with feeders supplying the mine, mill, townsite and powerplant auxiliaries.

It is a simple statement to say that additional engines have been installed to cope with a threefold increase in power. However, many other changes have had to be made to cope with the increased installations.

A new control room, new crane and lube systems have been installed over the past 18 months. Within the next few months, a new engine cooling system will be commissioned, using forced-draught cooling towers in a closed-loop system. The present system, one that has been in use for many years with little change, is open ended. Cooling water is pumped from a nearby

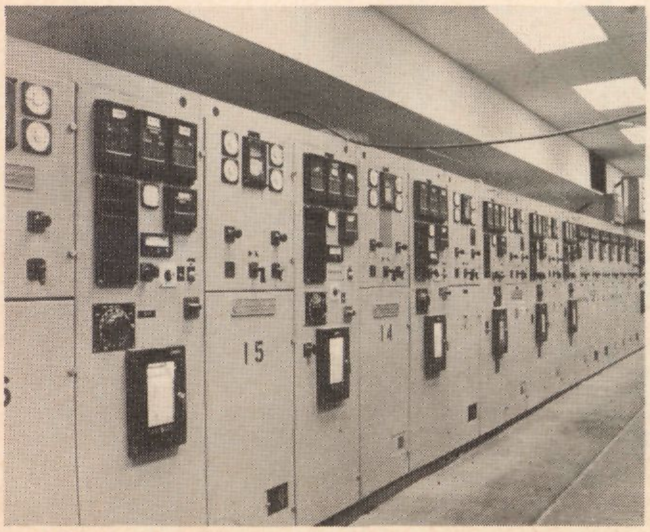


FIGURE 5 — 1400-kw generator in the powerhouse.

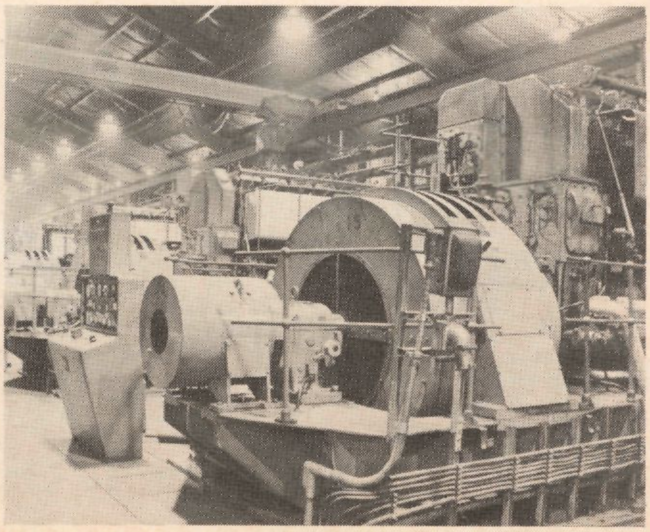
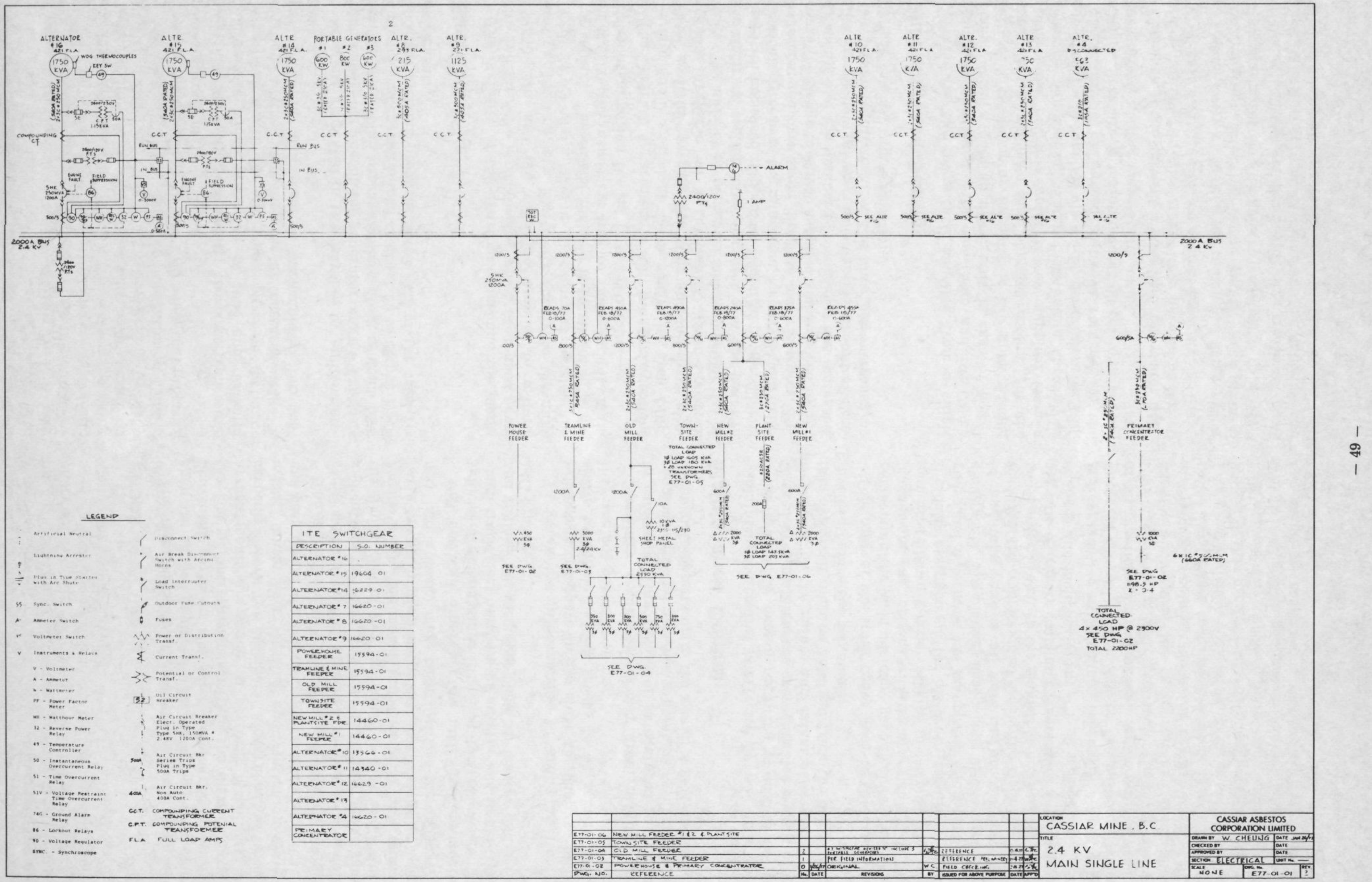


FIGURE 6 — Main 2300-volt switchboard in the new control room.



ITE SWITCHGEAR	
DESCRIPTION	S.O. NUMBER
ALTERNATOR #16	
ALTERNATOR #15	19604 01
ALTERNATOR #14	16229 01
ALTERNATOR #7	16620-01
ALTERNATOR #8	16620-01
ALTERNATOR #9	16620-01
POWERHOUSE FEEDER	15594-01
TRAMLINE & MINE FEEDER	15594-01
OLD MILL FEEDER	15594-01
TOWNSITE FEEDER	15594-01
NEW MILL #2 & PLANTSITE FDR.	14460-01
NEW MILL #1 FEEDER	14460-01
ALTERNATOR #10	13566-01
ALTERNATOR #11	14340-01
ALTERNATOR #12	16629-01
ALTERNATOR #13	
ALTERNATOR #4	16620-01
PRIMARY CONCENTRATOR	

FIGURE 7 — 2.4-kv main single line and feeders.

stream through the engines (some not fitted with heat exchangers) and back into a make-up sump, with eventual discharge back to the stream. To cope with the previously mentioned increase in 1978, two 600-kw and one 800-kw portable units are being connected into the main switchboard (Fig. 5). The reason for using mobile units, which are more expensive to operate, is that they will tide us over until five Ruston Hornsby 9 ATC engines (Fig. 6) currently operating at Clinton Creek become available in mid 1978 when that property ceases operation.

Besides generating electricity, the powerplant runs compressors which are rated at 1000 and 650 cfm respectively, with the air being piped to different locations in the plant. Because of increased usage, a new 1200-cfm compressor is contemplated for next year.

Steam is generated in two ways — by eight waste-heat boilers on the diesel generator engine exhausts and by two 200-hp package boilers. Both systems feed a steam header maintained at about 50 to 80 psi. Each of these systems produces about half of the total requirement.

Electrical (Fig. 7)

The majority of the distribution around the plantsite and townsite is 2300 volts, which is stepped down to 550 volts and 110 volts for supplies to motors, heating, lighting, etc. Electricity supply to the mine is at 24 kv via a step-up transformer outside the powerplant. The reason for this is to minimize losses on the long transmission line to the open pit. Main distribution within the pit is at 4160 volts.

The electrical department is split into three sections — mine, mill and plant — to cover the area fully. Those sections covering the mine and mill have electricians available on a three-shift, seven-day/week basis. The number of personnel and responsibilities of the electrical section have increased over the years in line with increased reliance on electrical power. This is particularly true in the mine area, as shovels and drills have changed over from diesel to electric power. Because the townsite is also supplied and serviced by the company, and the number of residences has increased, this area is now taking a considerable amount of time. The power requirement of the town has increased by 65% in the last twelve months!

Plumbing and Sewage

Water for the townsite is supplied from a single pumphouse equipped with one 75-hp and two 30-hp pumps, with a back-up diesel unit in case of power failure. Total pumping capacity is 875 gpm. At one time, this pumphouse also supplied cooling for the powerhouse, but this water is now supplied from a separate pumphouse. Water distribution throughout the townsite is supplied via a 6-in. main line with 4-in. branch lines. The unused water is returned to the stream. Pressure is maintained at 60 psi by a pre-set pressure-regulating valve at the outlet.

Because of the extreme winter cold, any water lines which are not run with steam lines have to be buried to at least 8-ft (2.44 m) to combat freezing.

The majority of the sewage is handled using either individual or collective septic tanks; good natural drainage makes this an effective system.

A single 25,000-gal.-per-day sewage treatment plant handles effluent from the kitchen, the laundry and some of the single quarters.

Garbage collection is handled by the company using a modern compacter truck which delivers to the garbage disposal area. The process utilizes specially made bulk containers in both the plant and townsite. Use of such modern equipment speeds up service and reduces manpower requirements.

Shops

The machine shop, carpenter shop and equipment shop are all well equipped with machinery to make replacement parts on site when necessary. Because of the lack of entrepreneurs within the district, special arrangements are made to perform machining and carpentry work for outsiders. Modernization in shop layout, lunchrooms, lighting and ventilation systems has been one of the recent improvements.

The machine shop includes lathes with up to 28-in. swing and 120-in. bed, broacher, shaper, rolls, welding and cutting facilities, and even a small forge. Although the capital cost of this equipment is high and its utilization is low, the money saved by being able to make parts for production machinery (air-freighting a part may take a week) more than justifies the expense.

Pick-ups and other service and small production vehicles are maintained in the equipment garage. The small truck fleet numbers 54 and the additional trucks and vehicles 30, including 980 Cat. loaders and a Cat. 14G grader.

To lessen the inconvenience of losing an item of production transport on day shift, the shop works two shifts to enable pick-up trucks to be serviced when they are not normally in use.

Carpentry requirements vary from making signs to cabinet making. The shop is equipped to meet all requirements and also has a paint bay which will accommodate objects up to the size of a pick-up truck.

Maintenance Control

Maintenance control is effected by way of a work-order system. A person requiring a task to be performed by a maintenance crew initiates a work order. Depending on the nature and extent of the work, certain approvals must be obtained from the departments involved before the job starts.

One copy of the work order goes to the supervisor performing the task, one copy goes to the maintenance planner and one remains with the originator. The foreman uses his work-order backlog to set priorities, plan the daily assignments for his crew, requisition materials, and determine his manpower and overtime requirements. The planner uses a copy of the work order to maintain records of work in process and work backlogs. On completion of a job, the field copy of the work order is sent to the planner, who then can generate data on repair costs and frequencies and report on changes and modifications.

A preventive maintenance scheme has been introduced in the mill which will be the basis for control of all repetitive inspection and repair on mill production machinery. A base frequency of eight weeks was adopted on most of the equipment and check sheets are issued to coincide with the regular weekly shutdown day. Major maintenance work is scheduled on a longer frequency, such as annually or semi-annually. Although this program currently represents a theoretical approach to scheduled maintenance, its real worth will be realized as changes in content and frequency are made in response to equipment problems and malfunctions.

The Environment

Melvin S. Taylor, Chief Engineer - Environmental,
Cassiar Asbestos Corporation Limited,
Cassiar, B.C.

Abstract

The environmental control program at Cassiar Mine is an active one involving government, unions and company personnel all working together to achieve a goal — "A clean and safe working environment". This can only be achieved by utilizing three of Cassiar's most important resources; i.e., people, time and money. Past environmental programs have more than proved that the goal can be reached and, with the future programs now in progress, Cassiar Asbestos will achieve its environmental goals.

Introduction

CASSIAR ASBESTOS CORPORATION LIMITED operates the only asbestos mine in British Columbia. The plantsite and town are located in close juxtaposition, creating special environmental considerations. Cassiar's environmental and health record has been very good. In over 25 years of asbestos mining and milling operations at Cassiar, B.C., there has not been one reported case of an asbestos-related disease.

Until the late 1960's, little was known about the effects of asbestos on the respiratory system. Some clarification has been gained as the result of extensive medical and technical research, certainly enough to establish that the excessive inhalation of asbestos fibre over extended periods of time may be detrimental to health.

To reduce this possible hazard to a minimum, Cassiar Asbestos has undertaken an extensive program of construction, monitoring and maintenance to meet government environmental standards.

Environmental Control at Cassiar

Since the inception of the mine in 1953, Cassiar Asbestos has carried out preventive medical programs by performing pre-employment and employee chest X-rays. The medical testing facilities are located at Cassiar, and definite guidelines are followed to examine employees for possible ill-effects which might result from asbestos dust exposure. To further these studies and to confirm that no asbestos-related diseases had been found, Cassiar Asbestos recently completed a survey of employees to determine the effects of asbestos exposure.

This survey, independently conducted by Dr. Stefan Grzybowski and staff from the University of British Columbia, involved employees who have worked at Cassiar for a period of ten years or more. This ten-year classification was arbitrarily chosen, as it appears to be the earliest time period in which an asbestos-related disease is likely to appear. The survey included a lung function test, chest X-rays, sputum tests, blood samples and an examination of employees case histories and employment records.

It has been established that there is a definite relationship between the inhalation of asbestos, cigarette smoking and respiratory problems. One medical authority stated that the excessive inhalation of asbestos fibre can lead to asbestosis. He concluded by saying, however, that curiously it seems to occur most often in those asbestos workers who smoke. Further, it is reported that asbestos workers who smoke incur about 92 times the risk of dying of a bronchogenic carcinoma as do those who neither work with asbestos nor smoke.

In view of the mounting evidence linking cigarettes to respiratory diseases, Cassiar Asbestos developed a realistic program to help those employees who want to quit smoking. This campaign met with encouraging success and included a public-participation campaign at Whitehorse, Yukon. The program has recently been upgraded and intensified and is being conducted by a smoking-cessation consultant, with the assistance of Cassiar personnel who have 'kicked the habit'.

At the Cassiar Mine, a large number of dust-producing areas have been brought under control. Areas that are not yet under control have been identified and dust control programs are underway. In these latter areas, employees are required to wear masks.

Dust Monitoring

Dust monitoring surveys were carried out at Cassiar Asbestos Corporation from 1953 to 1962 by the Workers Compensation Board of B.C. and then from 1963 to date by the Ministry of Mines and Petroleum Resources.

Recognizing the importance of dust control and measurement, Cassiar Asbestos adopted, in 1970, the midget impinger — the best known device for sampling air-borne dust at that time. Cassiar used this method to conduct its own first plant-wide survey in 1971.

The midget impinger bubbles dust particules through isopropyl alcohol, impinging the particles to the bottom of a collection tube. Dust samples thus obtained are then mounted on slides and counted using a 100X microscope and recorded in million particles per cubic foot.

Subsequently, in 1973, the improved filter membrane technique was adopted. This method enables the actual asbestos fibre to be counted, whereas the midget impinger includes all types of dust. The filter membrane technique, which was developed by the National Insti-

Melvin S. Taylor was born in Asbestos, Quebec, where he graduated from secondary school. He then received a diploma in both the technical and engineering fields from various institutes in Quebec. Following this, he furthered his studies in the business administration and management field, successfully obtaining his diploma in business administration. Mr. Taylor worked for 20 years in the mining and milling of asbestos fibre while living in Quebec. Prior to joining Cassiar Asbestos Corporation in July, 1977, he worked in various capacities with Canadian Johns-Manville Company Limited. He spent the last nine years on the administration and engineering of environmental control techniques in the asbestos industry prior to joining Cassiar Asbestos Corporation Limited as chief engineer - environmental.

Keywords: Cassiar Mine, Asbestos, Environmental control, Dust monitoring, Fibre counts, Ventilation.

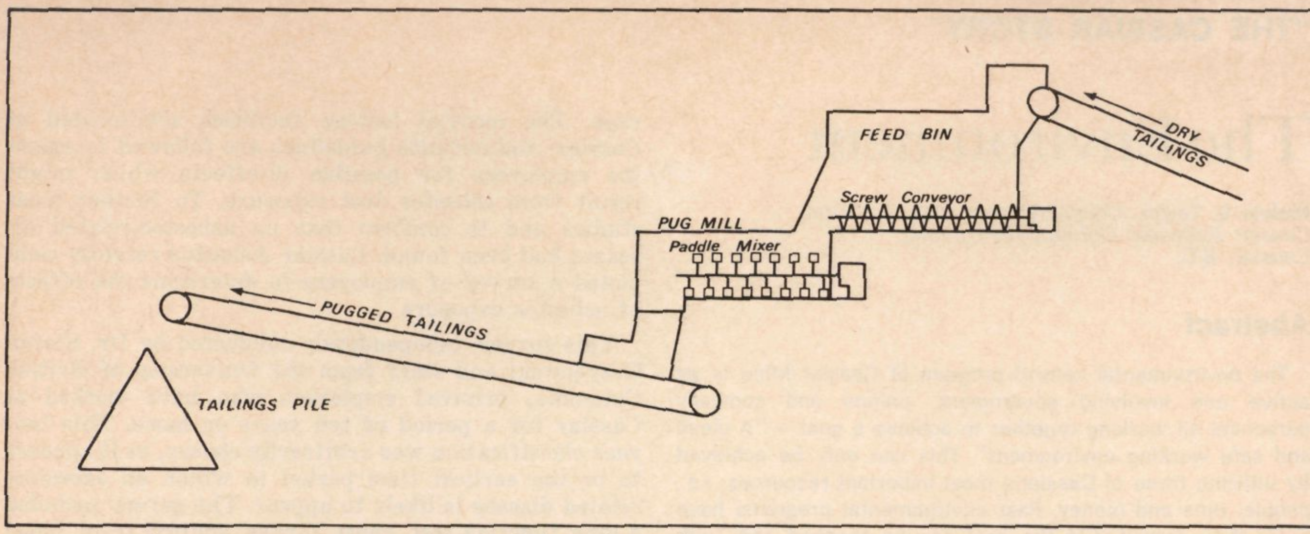


FIGURE 1 — The agglomeration system ahead of the tailings pile.

tute of Safety and Health in the U.S., is now used throughout the asbestos industry. It involves passing two litres of air per minute through a filter membrane which entraps the fibre. The filter is then mounted on a microscopic slide and fibres are counted by using a 450 to 600X microscope and recorded as fibre per cubic centimeter (f/cc).


Cassiar's new mill air building provides 480,000 cfm of process and ventilating air.


FIGURE 2 — Diversion of Troutline Creek in 1976.

Since June 1973, with the introduction of the filter membrane technique of sampling, Cassiar has reduced its average fibre counts from approximately 16.0 to less than 5 f/cc by instituting concrete environmental programs along with education of the employees on the proper handling of asbestos fibre. It was not until the early 1970's that engineers were able to develop the necessary improved technology to reduce dust emissions. In some instances, the technology is still to be attained.

Company Action Taken

By 1974, Cassiar had embarked on a program to meet government standards wherever possible by practical economics and existing technology. The following are examples of action taken.

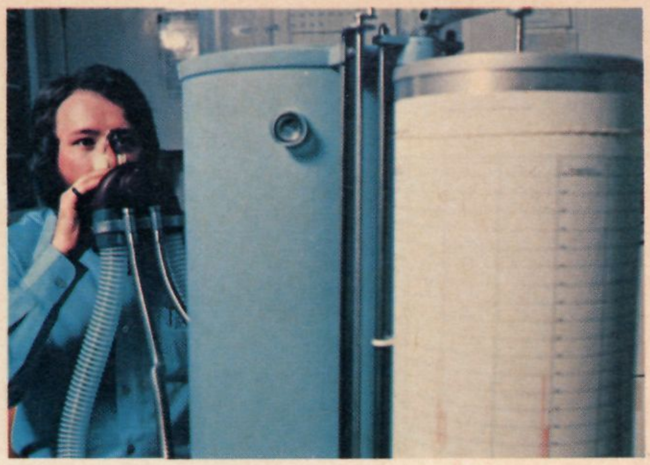
(a) Prior to 1973, wet collectors or cyclone collectors were known as providing the best suitable emission control techniques for hot gases from ore dryer stacks. However, with the development of filter fabrics for dry collection of high-particulate hot gases, bag filter units were adopted and proved to outperform conventional wet cyclone collectors. In 1973, Cassiar reduced emissions from its ore dryer stacks by removing its conventional cyclone collectors on three rotary dryers and installing bag filters; this reduced emissions from 300 lb/hr to less than 25 lb/hr per dryer stack.

(b) In 1974, Cassiar directed its attention to controlling dust emissions from the tailings pile. To reduce these emissions, an agglomeration system — or pug mill — was installed (Fig. 1). This reduced emissions to an acceptable level; there was no visible dust on discharge to the storage pile. It is now planned to install a second pug mill to wet other low-grade fines emanating from other product processing operations.

(c) To avoid possible contamination of the town's water supply, Cassiar re-routed Troutline Creek in 1976. This involved bulldozing a new creek bed and diverting the flow of water away from the low-grade storage pile (Fig. 2). The new water course has been seeded with suitable grasses and legumes to restore the beauty of the landscape.

(d) In 1977, Cassiar completed the construction of a changehouse (Fig. 3) for employees entering or leaving the plant. This facility is designed to prevent employees from carrying free asbestos fibre into their homes.

FIGURE 3 — Interior of the changehouse.



Medical tests are mandatory at Cassiar.

(e) During 1977, Cassiar completed a $7,000,000 mill air building, which provides 480,000 cfm (13,440 m^3) of process and ventilating air. Construction of this building started in late 1975. It was in its final break-in stage in late 1977 (see colour photo).

(f) The new mill air building will provide additional air for the processing operation and all the equipment enclosures now being installed in the mill. Equipment enclosures will include the enclosing of conveyors, screens, pressure packers or any piece of equipment used in the processing of asbestos that may emit dust into the mill atmosphere. Cassiar has installed three central vacuum systems throughout the plant for cleaning equipment and floors.

(g) To avoid contamination in the townsite from vehicles travelling from plantsite to townsite a truck-wash was constructed in 1977. All vehicles coming out of the mine area must pass through the wash area to be cleaned prior to passing into the town.

These are only a few of the environmental projects in which Cassiar is involved. To continually improve its dust emissions from 1970 to 1977, Cassiar Asbestos Corporation has spent in excess of 13.5 million dollars for capital-cost improvements. Within the next few years, the company is expected to spend in excess of $500,000 annually to control plant emissions through a continuous program of upgrading its existing facilities. All new installations will be equipped with environmental control machinery, representing a further heavy investment.

The environmental control department at Cassiar has trained unionized employees in the methods of collecting and analyzing dust counts under company supervision (Fig. 4). Every two months a survey of the complete plant is done by the qualified union employees. These results are forwarded to the British Columbia Ministry of Mines and Petroleum Resources as official records.

Also, in 1975, the company formed a Cassiar Mine Environmental Control Committee consisting of company and union representatives. This committee is responsible for surveillance of dust control measures and ensuring that the environmental control programs are progressing satisfactorily. The committee carries out an inspection of the plant once a month and then meets with management to discuss all phases of environmental control.

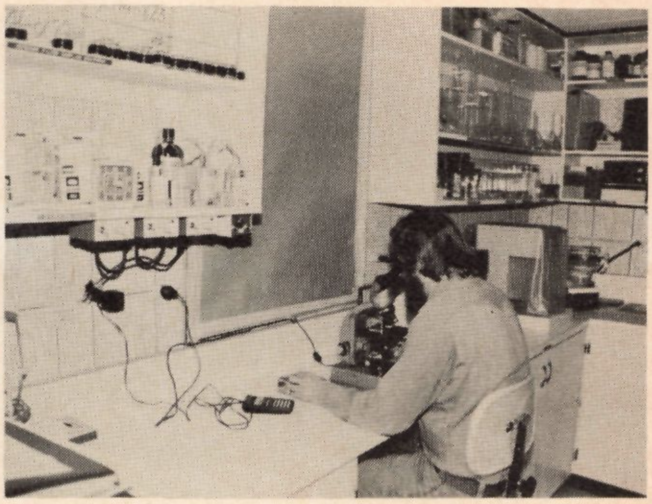


FIGURE 4 — An environmental technologist counting fibres per cc through a 600X microscope in Cassiar's modern environmental control laboratory.

In addition, Cassiar Asbestos and the United Steelworkers of America approached the provincial government with a proposal to form an asbestos environmental committee consisting of company, union and government representatives to evaluate environmental control programs and standards, and to analyze the environmental aspects of dust control at Cassiar.

In 1975, Cassiar joined the Quebec Asbestos Mining Association as a member of an environmental control group. This part of the QAMA shares information on dust control systems and improved techniques for testing air-borne asbestos fibre. The committee also evaluates the testing methods and instruments for asbestos sampling, as well as the health of employees. Cassiar is actively involved in research on the ill-effects of asbestos on health and is contributing its share to the funding of the Association and its work.

Since 1974, Cassiar has produced realistic programs to control its industrial and domestic environment. Its goal is to surpass government standards. Its aim is to ensure that company employees and their families have a pleasant, safe and healthy place in which to work and live.

(Reprinted from The Canadian Mining and Metallurgical Bulletin, April, 1978)
Printed in Canada