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MCDAME REPORT

Report on the 1987 Exploration Program

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Cassiar Mining Corporation
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McDame Exploration Project - 1987

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1.0 Summary:

In 1987, Cassiar Mining Corporation continued exploration of the McDame Asbestos Deposit located at Cassiar, British Columbia. The McDame Deposit is situated near the existing Cassiar open pit mine and is accessable to the present open pit and mill infrastructures.

The McDame Deposit was originally discovered in 1979 by development crews drifting towards the bottom of the Cassiar open pit for diamond drilling purposes. Prior to 1987, a total of 1,941 M. of lateral and raise development had been performed on two levels. In addition to the development, a total of 17,863 M. of diamond drilling had been drilled.

For 1987, an exploration development program was designed to obtain a bulk sample from the 1395 M.L. to test ground support techniques and not to interfere with future exploration and development. In addition to obtaining a bulk sample, a steel and wood ground support structure located in 6706N D.D.S. was scheduled to be removed. To accomplish these objectives, a decline from the 1415 M.L. adit had to be developed and access to the 6706N D.D.S. re-established.

As the program progressed, the criteria for the development program had changed as additional diamond drilling in the southeast portion of the McDame Deposit was required to determine long term ore reserves. After consultation with Cassiar Mining Corporation consultant, Dr. D. Laubscher, the program was changed from a bulk sample program to a diamond drilling program.

Development during the program was performed by both jacklegs and a single boom hydraulic jumbo in serpentinite and argillite respectively.

Ground support experience from the previous exploration development programs was used to design and implement an optimum support system. The new support system performed satisfactorily throughout the program.

In order to diamond drill the southeast portion of the Deposit, the 1415 M.L. beyond the 1563 M.L. ventilation raise to the start of the 7610 Drift E had to be rehabilitated. Once rehabilitation of this section of the adit was complete, development of the 7610 D.E. and subsequent diamond drill stations could commence.

Development of the 7610 Drift east and diamond drill stations were completed in late October, 1987. The diamond drilling program was deferred until a production decision regarding the McDame Deposit was made. Once a production decision is made, then the diamond drilling program will be re-activated.

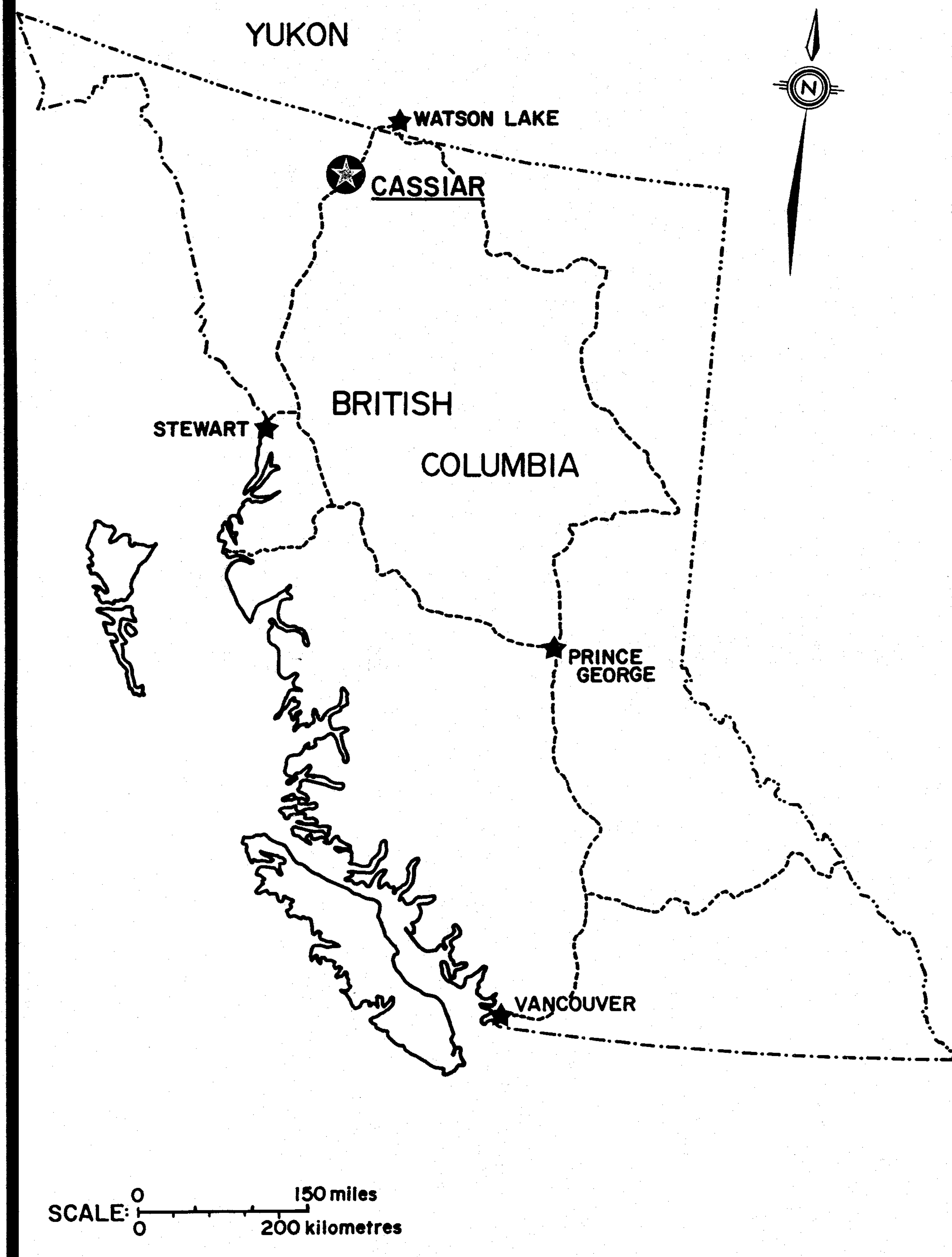


Figure 1 : LOCATION MAP

2. INTRODUCTION

2.1 Location

Cassiar, a town of 1200 people, is located in the Cassiar Mountains of northern British Columbia approximately 160 kilometers southwest of Watson Lake, Yukon. Access is by an all weather road from the Cassiar-Stewart Highway (Number 37). The closest airport is Watson Lake, which is serviced by Canadian Airlines' Boeing 737 jets three times a week. Figure 1.

Cassiar has been the site of open pit asbestos mining since 1953 and a well developed infrastructure for mining and milling of asbestos is established. The McDame Deposit is located to the south of Cassiar pit and at a lower elevation.

Access to the 1415 m level portal (lower) is by good dirt road from the mine haul road.

2.2 History

An upper adit at 1563 m elevation driven in 1978-1979 to allow diamond drilling under Cassiar Pit intersected an unsuspected ultramafic body containing abundant asbestos fibre of long length. A 290 M drift was extended southerly to provide better access to the new deposit and a total of 12,092 m in 37 holes was drilled in 1980 and 1981 to outline a new, buried deposit now called McDame.

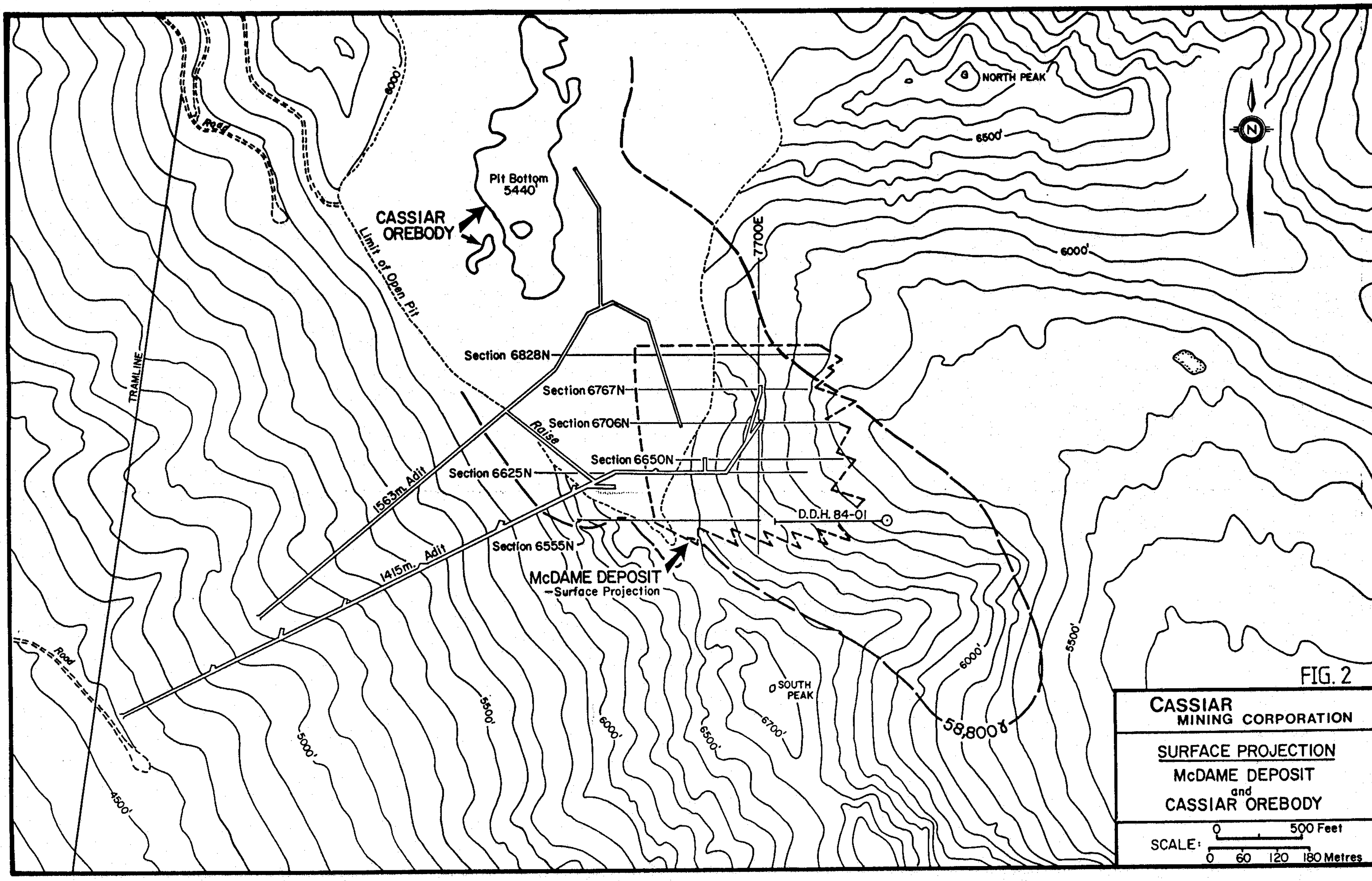


FIG. 2

CASSIAR MINING CORPORATION

SURFACE PROJECTION
McDAME DEPOSIT
and
CASSIAR OREBODY

Airborne magnetic surveys in 1983, followed by a deep drill hole from surface in 1984 indicated the ultramafic body was large and that fibre content extended southerly. Estimated reserves were 62 million tonnes of a grade and fibre value similar to the existing Cassiar Mine.

In 1985, the (lower) 1415 m elevation drift was driven 1081 m to intersect the deposit 150 m below the previous workings to confirm the nature of the mineralization, and to obtain a bulk sample for mill test purposes. A total of 875 tonnes was milled averaging 9.65% fibre with a high percentage of long fibre.

In 1986, further exploration development was performed on the 1415 M.L. The 1415 M.L. adit was extended and x-cut developed to a total lateral development of 329.4 M. In addition to the lateral development, a 241 m ventilation raise connecting the 1415 M.L. and 1563 M.L. was developed. Diamond drilling from both the 1563 M.L. and 1415 M.L. was performed to a total length of 4961 m. Figure 2.

A 846 tonne bulk sample from the 1415 M.L. in the hangingwall of the deposit was obtained. Of the 846 tonnes, 588 tonnes was processed producing 24.5 tonnes of fibre for a grade of 4.2%.

2.3 Objectives

In May 1987, an exploration development program was designed to access the 1395 M.L. footwall of the McDame Deposit via an -8% decline developed in argillite from the 1415 M.L. A x-cut would be developed across the entire width of the orebody from which a bulk sample would be obtained for test milling purposes. The x-cut would also provide fresh exposures of the geological structure of the ore zone. It is anticipated that detailed mapping of the geological structure across the entire width of the ore zone will assist in developing the block cave design and computer block model. Figure 3.

In addition to the development of the ore x-cut, the wood and steel ground support structure located in 7606N D.D.S. would be removed by the underground contractor.

During the month of July, after consultation with Cassiar Mining Corporation's Mining Consultant, Dr. D. Laubscher, it was determined that further diamond drilling of the orebody to the southeast was required. The diamond drilling was required to prove up an area of concern where an intersection of 155 meters, grading 7% fibre, was indicated from previous drilling. The intersection could not be accurately located however, due to inconsistencies in diamond drill hole survey data and further drilling was advised.

Therefore, the principal objectives of the underground development program were changed to the following:

1) Establish diamond drill station located off of 1415 M.L. that would permit drilling of the southeast section of ore.

2) Test an improved ground support system, and improve operational support techniques to react with changing ground conditions.

3) Test different round blasting methods in order to permit full face blasting instead of blasting by the baby arch approach.

Secondary objectives included

1) To ensure that exploration development did not interfere with future mining development.

2) To locate exploration development to aid future development.

3) Monitor ground support system for future planning.

With these objectives, the exploration development program was modified while the development of the 7235 decline was in progress. In order to develop the diamond drill stations to drill the southeast section of the orebody, rehabilitation of the 1415 M.L. adit from the 1563 ventilation raise to the 7600E diamond drill station was required. The removal of the steel and wood support structure in 6706N D.D.S. would be deferred.

All funds allocated to the argillite development and bulk sample would be diverted to the rehabilitation and development of the diamond drill stations and serpentinite drifting.

In order for diamond drilling of the southeast corner of the ore zone to be accomplished, a drift collared from the southeast of 7600E D.D.S. or the 1415 M.L. had to be developed. The 7610 D.E. drift gave access to two diamond drill stations, located on 6600N and 6550N section lines.

2.4 Drifting

Initially the exploration development drifting program consisted of the following: Figure 3.

1) Develop a 4.0 m x 4.0 m minus 8% decline from the 1415 M.L. (7325 decline east) to the 1395 M.L. The decline was to not intersect the orebody footwall contact, but to remain approximately 40 meters out from the contact to permit the extension of the decline to the first drawpoint production level (1380 M.L.) .

2) Once the decline reached the 1395 M.L. a cross-cut into the footwall ore zone would be developed at a grade of +3%. The level x-cut would be developed as a 4.0 m x 4.0 m development heading in the argillite, but at a point located approximately 10 meters from the contact, the x-cut heading would be reduced to a 3.35 m x 3.35 m opening for development in the barren and ore bearing serpentinite.

3) Located at the junction of the decline and footwall x-cut, a water collection sump and truck turn around would be located. The water sump at a later date would be used as the continuation of the decline to the 1390 M.L. drawpoint production level.

Tender documents for the proposed development were released in early May to three underground contractors. Only two of the three underground contractors submitted proposals at the closing date in May 21, 1987.

The contractors who submitted proposals complete with tender prices are as follows:

TABLE 2.40

McDAME EXPLORATION DEVELOPMENT TENDERS

Company	Tender Prices
J.S. Redpath (Rockbore)	$ 1,732,128.00
Canadian Mine Development Ltd.	$ 1,098,495.25*

* Excludes cost plus work

Tonto Mining Services Ltd. decided not to submit a proposal for the project.

Canadian Mine Development Ltd. was selected solely upon their low bid and quick mobilization. Canadian Mine Development (C.M.D.) stated that they would be ready to commence the project in 5 days upon notification of being successful. C.M.D. had left all their underground equipment at the mine site from the previous 1986 development program and this is the reason why they could mobilize so quickly. However, instead of being able to mobilize in 5 days, C.M.D. took two weeks to arrive on site stating that they were having problems obtaining manpower.

Canadian Mine Development arrived on site on June 18, 1987 and commenced to rehabilitate their shop facilities and perform site preparation. Rehabilitation of the 1415 M.L. as far as the 7235 decline was completed by June 26, 1987 and the first development round of the decline took place on June 27, 1987.

Due to the change in the development program the 7325 decline was stopped on July 27, 1987 after 80.014 m had been developed.

Rehabilitation of a 190 m long section of the 1415 M.L. adit from the 1563 ventilation raise to approximately 10 meters beyond the 7610 Drift E. commenced on July 27, 1987 and was completed on August 17, 1987.

Development of the 7610 D.E. and the two diamond drill stations commenced on August 18, 1987 and was completed on October 29, 1987 after 142.667 m (eqiv.) had been developed.

Total development meters for the project over the 135 day period was 222.681 m (eqiv), not including rehabilitation meters.

2.5 Claims

All exploration development was restricted to 2 separate claims. Claim names and type of work are:

Name	Record No.	Type of Work
Goat No. 1	L6501	Rehabilitation, decline
Goat No. 2	L6502	Rehabilitation, decline, drifting

3.0 Electrical Power

In 1986 a powerline was constructed from the main open pit feeder line to the project site. The powerline supplied electrical power to the project for the program. No electrical supply or equipment problems were experienced during the program.

4.0 Rehabilitation of the 1415 M.L. Serpentinite Development

Rehabilitation of the 1415 M.L. Adit in Serpentinite commenced on July 27, 1987, immediately following the closing of the 7375 E Decline face. Rehabilitation started at the 1563 Ventilation Raise and stopped 10 meters beyond the proposed start of the 6710 D.E. Drift.

A summary of the Adit Rehabilitation is as follows:

TABLE 4.0.1.

Rehabilitation Advance Rate

			Total M	Average Advance	
Period	Days	Shifts	Rehabilitated	M/Day	M/Shift
July 27 - Aug. 17	21 1/3	60	190	8.91	3.17

Additional ground support was installed over and above the existing ground support in making the adit safe. The total amount of ground support installed and the amount per meter is shown on Table 5.02.

TABLE 4.02.

Rehabiitation Ground Support

Splitsets		Dywidag		Screen		Shotcrete	
No.	No./M	No.	No./M	M^2	M^2/M	M^3	M^3/M
333	1.75	424	2.23	354	1.86	167	0.88

Rehabilitating the 1415 M.L. provided training for the development crews in the installation of the type of ground support system required for Serpentinite Development.

5.0 Drifting

Logistics/Methods

Underground development crews from Canadian Mine Development Ltd. arrived on site June 18, 1987 and began to rehabilitate their maintenance shop facilities. Most of the underground equipment from the 1986 exploration development program remained on site during the winter. A Tamrock single boom hydaulic jumbo and accessories and to other small equipment was shipped to the project site.

All of the underground services were re-connected and rehabilitation of the 1415 m level commenced. Scaling loose down from the back and sidewalls and re-gravelling of the adit roadway surface was performed.

Re-routing of the ventilation system for the development of the decline was established.

Rehabilitation of the 1415 M.L. adit to the proposed start of the 7325E decline was completed on June 26, 1987.

Development of the 7235E decline commenced on June 27, 1987. Advance in the argillite was slow due to a shortage of contract miners, ground conditions and equipment mechanical problems. It was not until better ground conditions were intersected and once a full complement of contract miners was reached on July 19, 1987, that the development rates did increase.

On July 27, 1987, development of the 7235E decline was terminated in order that development of diamond drill stations could commence.

The Tamrock single boom electrical hydraulic jumbo was used in the development of the decline in argillite. For the serpentinite development, the jumbo was replaced by jacklegs as mudding up of the drill steel was a problem in previous programs. Jacklegs were used to complete the serpentinite development program.

TABLE 5.01

COMPARISON OF RATES OF ADVANCE

Method	Time Period	Days	Advance (m)	Average	Location
Jumbo	Jun 27-Jul 27	30 2/3	80.014	2.61	7325E Decline
Jackleg	Aug 18-Oct 29	68	142.667	2.10	7610 D.E.

Average = 2.257m/day over 98 2/3 days

TRAMROCK JUMBO IN 7325 E DECLINE

7325 E DECLINE FACE ON JULY 27/87

5.1 Development by Drilling and Blasting

At different times during the development program jacklegs and a jumbo were employed for drilling of drift rounds. The procedure with both followed conventional drilling procedures and standards for ground support technique and face round cycles were different for each.

For both Argillite and Serpentinite, development crews had to follow a period of learning before they became experienced in performing ground support in poor ground conditions. Ground support designs and specifications had been provided by Cassiar. Development support techniques had to be taught to the crews by Cassiar personnel and as they obtained more experience, a more systematic approach to ground control was taken.

A direct comparison of production rates for jumbo and jackleg development cannot be made as the two units did not operate in the same ground for any significant length of time.

5.2 Discussion

Comparison between Jumbo and Jackleg Development:

A comparison between the two development equipment is shown on Table 5.20.

TABLE 5.20

COMPARISON BETWEEN JUMBO AND JACKLEG EQUIPMENT

	Jumbo	**Jackleg**
Boom (slide length)	6.1 m (20 ft.)	-
Manufacturer	Tamrock	Secan
Drill Operation	Electric/hydraulic	Pneumatic
Tramming	Diesel	-
Bit	70 mm (2 3/4")	35 mm (1 3/8")
Drift	4.0 m x 3.5 m (Argillite)	3.35 x 3.35 m (Serpentinite)
Drift Round Drilled	3.0 m	2.0 m

5.2.1. Jumbo

The Tamrock electric/hydraulic single boom jumbo was employed initially for argillite development. All ground support drilling was performed by jackleg off a platform or "Long Tom" attached onto an L.H.D. bucket.

Drilling of the round was performed with a 70 mm (2 3/4") drill bit and the round drill length in good ground was 3 meters. The cut holes were reamed out to 76 mm.

INITIAL SLASH OF 7325 E DECLINE

7610 D.E. ROCKBOLT DRILLING

Average drill cycle time for drilling off a round of 45 holes at a length of 3.0 m was 2 hours (approximately). The drill was not equipped with parallelism and grade lines had to be provided for the jumbo drill operator. Re-positioning the drill to the grade lines caused some delays in face drilling.

In areas where the jumbo drill encountered graphitic schist the soft ground would cause the drill bit and steel to mud up. Increasing the flushing water pressure and reducing the hydraulic pull down pressure did not completely solve this problem. Drill steel and bits did become stuck and in some cases the steel and bit were lost when the round was blasted.

Initial development of the 7375 decline in argillite was performed by jacklegs until there was enough room available for the jumbo to operate.

TABLE 5.21

ARGILLITE DEVELOPMENT RATES

Period	Advance m (Equiv)	Advance m/day	Advance m/shift	Method
June 27-July 2	16.301	2.72	1.36	Jackleg
July 3-15	23.70	1.82	0.76	Jumbo
July 16-27	40.013	3.43	1.25	Jumbo
Total	80.014			
Average		2.81	1.13	

Total jumbo operating hours - 83.0

Total feet/meters Drilled - 7,907.0/2,409.94

Average feet drilled/hr - 95.27*

Average meters drilled/hr - 29.04*

Average feet drilled/round - 439.28+

meters drilled/round - 133.90+

Note: * Includes Jumbo travel time, set-up and tear down time

+ 3 m average round length

The development rate in argillite fluctuated from period to period as a result of the following:

1. Initial development by jackleg.

2. Equipment breakdowns and delays.

3. Variable ground conditions.

4. Reduced length of round due to poor ground.

5. Performing ground support in addition to the normal ground support required for competent argillite.

6. Mucking and remucking cycle time.

For the period June 27 to July 16, 1987, the decline was developed in poor ground conditions (graphitic schist) which required extensive ground support. Once the decline encountered good argillite requiring minimum support on July 16, the development rate increased.

Mucking of the face was a problem for the program as the closest remuck station was approximately 100 meters to the west of the 7325E decline. As the decline developed the mucking cycle time increased. If decline development had not been halted on July 27, 1987, another remuck station would have been developed approximately 10 meters beyond from the stopped face.

The Jumbo operated for several shifts in Serpentinite development; the results were considered to be unsatisfactory for the following reasons;

1. Bit Flushing:

   A 4 m steel of 38-40 mm in diamter and a 32 mm was required for drilling spile holes. The water flushing channel in the steel was too small to provide satisfactory bit flushing. Frictional losses in the 4 m steel were high enough to reduce the bit water pressure and the quantity of water reaching the bit significantly resulting in inadequate flushing performance.

2. Jumbo Water Cooling:

   The Jumbo water cooling system operates at a high pressure and a low water flow. The cooling system is also used for bit flushing and is not a separate system. The cooling system water pressure and flow settings cannot be significantly altered for bit flushing purposes without having detrimental affects on the Jumbo cooling function.

3. Hydraulic Drill

The Tamrock hydraulic drill and matching jumbo was designed for hard rock conditions. For soft rock conditions, such as in Serpentinite, the energy per blow cannot be significantly reduced without damaging the drill and associated hydraulic components. Excessive energy output and inadequate bit flushing led to continual problems with mudding up of the drill steel and bit. The problem continued even after the main flushing hole was welded up and increasing the water pressure and volume had no effect.

4. Bit Drill Down Pressure

As described above, the Jumbo was designed for hard rock ground conditions which require a high bit pull down pressure. This high pressure is not required for Serpentinite. Major design and operational changes to the jumbo would be required to reduce the pull down pressure and were not considered to be warranted.

5.2.2. Jackleg Development

Initial argillite decline development was performed by jackleg until the decline was established to accommodate the jumbo.

All other jackleg development was performed in serpentinite in the development of the 7610 D.E. and diamond drill stations 7600E and 7700E.

Serpentinite drift development was performed off a modified "Long Tom" platform that was mounted in a L.H.D. bucket. Drilling of the upper portion of the face would be performed with 3 jackleg drills, drilling a two (2) meter round. The lower half of the round would be drilled off the drift running surface.

Total time to drill off a two meter round in serpentinite including set-up and tear down time was approximately 4 hours. A average of 45 holes were drilled per round. The cut holes were not reamed.

All ground support drilling was performed by jacklegs off of the "Long Tom" platform.

TABLE 5.22

SERPENTINITE DEVELOPMENT RATES

Period	Advance m (Equiv)	Advance m/day	Advance m/shift	Method
August 18-31	29.677	2.119	0.742	Jackleg
September 1-15	20.345	1.909	0.636	Jackleg
September 16-30	29.240	1.949	0.650	Jackleg
October 1-15	33.739	2.249	0.750	Jackleg
October 16-28	29.666	2.282	0.761	Jackleg
Total	142.667			
Average		2.119	0.714	

Total feet/meters drilled = 28 800.616/8 778.0

Average feet drilled/round - 429.860

Average meters drilled/round - 131.015

The development rate in serpentinite fluctuated from period to period as a result of the following:

1. Variable ground conditions.

2. Longer ground support cycles due to poor ground conditions.

3. Equipment breakdown and/or delays.

4. Mucking and remucking cycle times.

5. Experience of development crew for each of the serpentinite development cycles.

During the 1986 exploration development year, The average development rate for the jackleg development was 2.77 meters/day.

The reason for the reduction in development rate are as follows:

1. Specified ground support requirements and improved ground support control.

a) Spiles - increased number and length.

b) Uniform thickness of shotcrete - thicker.

c) Dywidag bolts - increased number and length.

2. Underground haulage trucks

For 1986, two underground haulage trucks were available and were used to haul muck out. During the 1987 program, only one haulage truck was used.

3. Experience and quality of underground personnel.

Crew staffing was a problem during the 1987 development year due to the number of other underground projects occurring across Canada. The quality and level of experience of personnel was lower than in 1986.

4. Larger size drift development opening.

Towards the end of the program, the finished size of the drift increased from 3.35 x 3.35 to 3.35 x 4.0. This increase resulted in a 6% increase in the drift perimeter and a 20% increase in the area, resulting in increased mucking cycle times.

The reason for the drift size increase is as follows:

1. Sidewall ground control problems.
2. Increase in width to install additional sidewall spiles and failure to reduce the width after the section of additional sidewall spiles had been completed.

5.30 Development Cycles

Development cycles for argillite and serpentinite development are as follows:

TABLE 5.30

DEVELOPMENT CYCLE

	Argillite	Serpentinite
Equipment used	Electric/hydraulic Jumbo	Jacklegs
Drilling	45-70 mm holes 3 m long Center hole of cut reamed to 76 mm (3")	45-34 mm holes 2 m long
Spiles	No spiles required except under extremely poor conditions	8 spiles installed on 0.45 m spacing. Spiles grouted in with cement fondue. Spiles pinned to back by a re-bar strap.
Blasting	Load all holes as per the normal drift round. Amex used whenever possible, but if wet, Magnafrac is used. 5 m Nonel detonators used.	Load holes as high as the diamonds or baby arch. De-sensitized Magnafrac with electric detonators used.

Ventilate	Ventilate
If shotcrete is required to be applied muck out upper half of round.	Depending upon the blast, if the blasted muck works its way up to the perimeter holes, no additional blasting is required. If additional blasting is required then Xactex is employed to trim the back.
Shotcrete sealing layer of 50 mm on high points. Shotcrete previous round with second coat	Ventilate
Install screen and rockbolt with Dywidags if necessary.	Muck out upper half of round
Ready to drill face.	Screen and rockbolt back and sidewalls.
	Ready to drill face.
Elasped time: 16 hours.	Elasped time: 18 hours.

The cycle time could vary ± 1-2 hours depending upon the following:

1. Equiment breakdowns and/or delays.

2. Drilling delays - steel and bit mudding up.

3. Shotcrete application - thickness, rebound, technique.

4. Crew experience.

5.4 Ventiliation

5.4.1. Air Flow

Ventilation of the 1415 M.L. adit was performed by bringing in fresh air on the 1563 M.L. and down the 1563 ventilation raise as shown on Figure 4. A 50 HP and a 40 HP Wood's Axial vane fan were located in a bulkhead constructed at the base of the 1563 ventilation raise on the 1415 M.L. Fresh air was diverted to the development face via 42 inch diameter ventilation ducting. The exhaust air was allowed to exhaust to atmosphere via the 1415 M.L. adit. A total of 31 400 C.F.M. was used to ventilate the 1415 adit.

A ventilation bulkhead/regulator constructed during the 1986 development program on the 1563 M.L. adit was not in service during the program.

No ventilation problems were experienced during the 1987 development program.

5.4.2. Environmental Control

Testing of airborne asbestos dust and dust from shotcreting were performed on regular intervals. Ventilation surveys were performed on a regular basis.

Gas tests at various locations were within threshold limits set by the Department of Mines.

Airborne dust and fibre monitoring was performed by the Cassiar Environmental Department throughout the program. Their monitoring program provided technical environmental data that will be useful for future drift development and design.

5.5 Ground Support

5.5.1. General

Ground support experience obtained from the 1986 exploration development program was used in the design of an optimum ground support system for both argillite and serpentinite development.

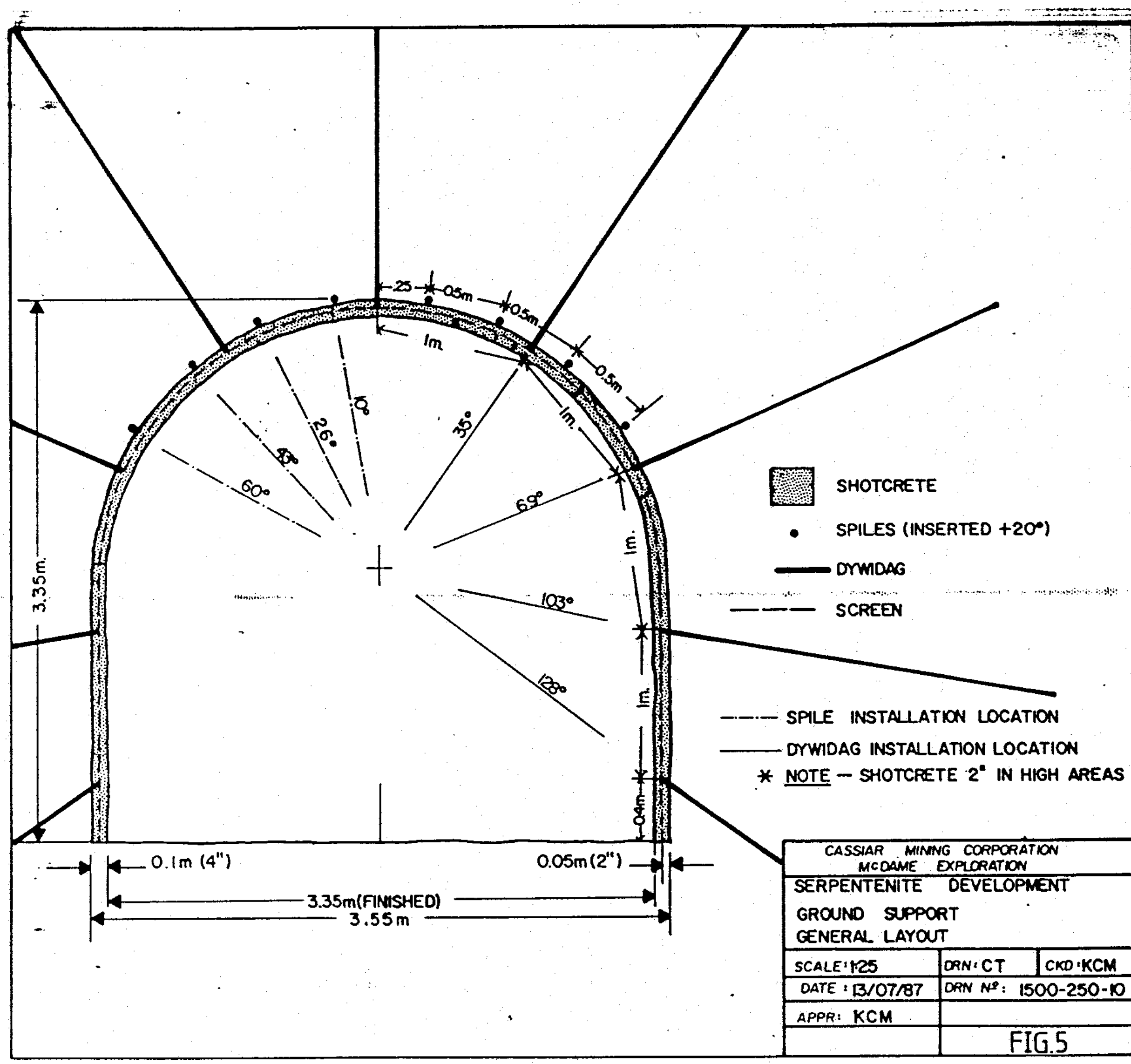
Monitoring of the ground support installed during the 1986 program was performed on a regular basis during the winter of 1987. Areas where the ground support was performing provided additional information in the design of the optimum ground support system.

Dr. D. Laubscher, Cassiar Mining Corporation's consultant on the McDame Project, assisted Cassiar personnel in the development of the optimum ground support plan.

The optimum ground support plan for serpentinite is shown on Figure 5. This ground support plan was to be used on the standard plan for all serpentinite ground support. Depending upon ground conditions the plan could be adjusted to suit the conditions.

The ground support plan consisted of the following:

Material	**Description**
Shotcrete	applied to a thickness of 100 mm, 2 x 50 mm passes, on the high points
Spiles	25 mm x 4.88 m, grouted into place with fondu grout.
Spile pin straps	Pin the spiles to the back to provide continuous back support coverage. Fabricate from 13 mm rebar.
Rockbolts	Dywidag 22 mm x 2.44 m long bolts complete with nuts and plates. Both grouted into place with 32 mm x 0.91 m long C-90 resin cartridges.
Screen pins	Pins used to tie the screen tight to initial layer of shotcrete.



25
0.5m
0.5m
0.5m
1m
1m
1m
1m
0.4m
10°
26°
43°
60°
35°
69°
103°
128°
3.35m
0.1m (4")
0.05m (2")
3.35m (FINISHED)
3.55m
SHOTCRETE
SPILES (INSERTED +20°)
DYWIDAG
SCREEN
SPILE INSTALLATION LOCATION
DYWIDAG INSTALLATION LOCATION
* NOTE — SHOTCRETE 2" IN HIGH AREAS
CASSIAR MINING CORPORATION
McDAME EXPLORATION
SERPENTENITE DEVELOPMENT
GROUND SUPPORT
GENERAL LAYOUT
SCALE: 1:25
DRN: CT
CKD: KCM
DATE: 13/07/87
DRN Nº: 1500-250-10
APPR: KCM

FIG.5

This support system was the basis for all ground support. A variation of the system was used in argillite development as the spiles and the second coat of shotcrete were not installed. Figure 5.

Once the crews became experienced with the installation and operation of the support system, optimum sequences were developed.

5.5.2. Shotcrete

All shotcrete applied during the 1987 development program was of the dry type and applied pneumatically.

The premixed shotcrete was manufactured in Vancouver by Target Products Ltd. and shipped to the project site by Arrow barge. The ingredient mixture of the shotcrete is as follows:

Ingredient	Description
Silica fume	12%
Accelerator	3% - used to reduce setting time
Type 10 Cement	Used until ambient temperature dropped below -10C
Type 30 Cement	Used after ambient temperature dropped below -10C

No steel or polypropylene fibres were used in the shotcrete as these fibres cannot be separated in the Mill and contaminate the product.

One batch of shotcrete shipped to Cassiar contained steel fibres that were remnants from another customer's shotcrete batch. Target was notified of the problem and it was remidied by ensuring that their production plant was cleaned out before manufacturing shotcrete for Cassiar.

Serpentinite ground support specifications required that 100 mm of shotcrete be applied to the high points of the back and side walls down to the spring line. Coring of the applied shotcrete was performed to monitor the following:

1. Consistent thickness.

2. Shotcrete quality.

3. Shotcrete bonding to application layers.

4. Record of shotcrete applied.

5. Review of quality of shotcrete and assist the shotcrete crew in improving their shotcrete application technique.

Initial application of shotcrete at the start of the project was substandard even when the contractor did provide qualified shotcrete personnel. The quality of shotcrete did improve and become satisfactory after shotcrete personnel changes were made and after a shotcrete consultant was brought on site by Cassiar. Since crew changes occurred every 45 shifts, the contractor could not guarantee that more than one qualified shotcrete person would be on each shift for shotcreting.

A report on the shotcreting performed by Canadian Mine Development is presented in Appendix I.

Shotcrete application technique and crew experience did improve following demonstrations provided by the consultant, but resistance to suggested improvements continued to be a problem.

Critical areas that affect the quality of shotcrete are as follows:

1. Air pressure.

2. Pre-damping of mixture.

3. Water mixing at nozzle.

4. Equipment operation.

If all the above factors are in harmony, then the shotcrete will be of a high quality. If any of the factors are not set or operating correctly, then the shotcrete quality will deteriorate with the quantity of rebound and the percentage of voids increasing significantly.

5.2.2. Shotcrete Usage

Serpentine Development shotcrete application volume as per the specified ground support system was used as the base volume Theoretical Amount) of shotcrete required for necessary ground support. A factor of 1.73 was used for rebound and filling in the low points with shotcrete. This is the same factor that is used at Rodgers Pass Tunnel project by Manning Kumazai Joint Venture. A shotcrete usage factor of 100% has been proven to be too low. The theoretical amount of shotcrete applied did not include shotcrete application to the face.

TABLE 5.52

Serpentinite Development Additional Shotcrete Application Factor.

Drift size opening 3.35 x 3.35 (finished).

Theoretical M^3/M	Theoretical @ Factor 173%	Actual M^3/M	Actual Factor
1.03	2.76	4.19	307%

* 20% Rebound
* 120% over consumption

Reasons why the shotcrete consumption factor is much higher are as follow:

1. Shotcrete crew application technique (Rebound should not be more than 15-20% with Silica fume).
2. Excessive thickness of shotcrete shot on the face.
3. Shotcreting Procedures - Shooting shotcrete through untightened screen, which increases shotcrete rebound.
4. Initial size of exploration drift larger than the design drift size opening by 20%. Larger size opening increases shotcrete consumption.

   Larger size opening due to Contractor's problem with sidewall ground control.
5. Excessive thickness of shotcrete. 100mm of shotcrete requested but core sampling indicated that as much as 254mm or more had been applied.
6. Reluctance of development crew to employ gauges to indicate a satisfactory thickness of shotcrete.

5.5.4. Rockbolts

Two type of rock bolt systems were employed on the project, as follows;

(a) Ingersol Rand - split sets (Argillite only)

(b) Resin Grouted Dywidags (Serpentinite and Argillite)

From previous exploration development experience, split sets cannot be employed successfully in poor fractured ground conditions such as

INSTALLING RESIN CARTRIDGES FOR DYWIDAG ROCK BOLTS

INSTALL CEMENT FONDU INTO DRILL HOLES (ROCK BOLT TEST) graphitic schists and Serpentinite. Split sets of 1.83M in length can only be employed under good Argillite ground conditions.

Resin Grouted Dwyidag bolts of 2.44M in length are employed exclusively in Serpentinite and in any poor ground conditions that may be encountered. Resin with a setting time of 90 seconds is used. Portland Cement Grout was tested as a substitute for resin grout and found to be unsatisfactory as the setting time was too long, in excess of 30 hours due to cold rock temperatures of +5 - +7°C.

Grout setting times of half hour maximum and at least 20% ultimate strength within 40 minutes are required as the Dwyidag bolts are tensioned during the screen installation process. Fondu Grout is used for the 4.88 M spiles which sets within the required time but does not have the desired strength characteristics. Resin cartridges cannot be used for spiles as there is no method of spinning the re-bar spiles and mixing the cartridges.
Resin Grout cartridges measuring 32mm and 915mm were found to be the optimum size cartridge for use with the 22m x 2.44m Dywidag bolts, as only two cartridges were required.

Table 5.54

Rock Bolt Usage

Ground	Split Sets 1.83M	Split Sets/M	Dywidag 2.44M	Dywidag/M
Argillite	573	7.16	230	2.87
Serpentinite	28	-	1 405	9.85

5.5.5. Spiles

Based on previous development experience spiles are required to maintain ground support during the application of the initial sealing layer of shotcrete. Without the spiles the back of the round could deteriorate beyond the perimeter holes after the round has been blasted. Once the process begins it is very difficult to stop and only shotcrete can be used to provide the initial additional support.

The spiles are installed so as to act as a continuous steel beam along the back of the drift. Spiles are installed at an angle of +10°C and pinned to the previous round back. The spiles are of sufficient length to reach into the next round ahead of the round to be blasted, as shown in figure . Spiles fabricated from 25mm x 4.85m re-bar and grouted in with cement Fondu and pinned with Dywidag to the previous round back with 15 mm re-bar straps proved to be the best spile installation.

The number of spiles installed depended upon ground conditions. Under normal circumstances, 8 spiles are installed.

Spile consumption for Serpentinite development is as follows:

Table 5.55

SPILE CONSUMPTION (SERPENTINITE)

Spiles Installed	Spiles /M Advance
1,342	9.41

Reasons for the high consumption rate are as follow;

1. Poor ground conditions, especially through shear zones, required additional spiles.
2. Spiles are installed at the discretion of the Shifter driling the spiles. An overly cautious approach to the ground condition could have contributed to the high installation rate.

5.5.6. Screen and Straps

For both Argillite and Serpentinite development screen and straps were employed in the support of the openings. Welded steel mesh of 8 gauge thickness with 100mm x 100mm openings in 13.01 M^2 sheets is used. Diamond mesh galvanized mesh was not used during the project as the mesh does not form well to the back or sidewall openings. Crew experience and dedication was instrumental in attaining satisfactory installation.

Forming and anchoring of the screen was accomplished with split sets, Dywidags, and rock clips. Rock clips are an inexpensive method of tying the screen to the back and side walls. The clips worked better in Argillite than in Serpentinite because of the harder ground. Pulling out of the rock clips in Serpentinite was a continual problem.

Straps were mainly used in Serpentinite development for pinning of the spiles. The straps were made out of two lengths of 15mm x 1 M re-bar welded with cross braces at three points. This design permitted the strap to become part of the ground support structure when shotcrete is

7325 E DECLINE FINISHED

7610 D.E. FINISHED

applied. The strap does not prevent the shotcrete from adhering to the back.

TABLE 5.56

SCREEN AND STRAP CONSUMPTIONS:

Ground	Screen		Straps	
	M^2 Installed	M^2/M Installed	Units Installed	Units/M Installed
Argillite	848	10.60	0	0
Serpentinite	1,466	10.27	212	1.49

5.6. Blasting and Explosives

Different blasting techniques, explosives and accesories were required for development rounds in Argillite and in Serpentinite. The Argillite is generally far more competent than the Serpentinites.

TABLE 5.60

Comparison of Rock Strengths for Argillite and Serpentinite

(Reproduced from the 1986 Development Program Report).

Material	Unconfined Compressive Strength
1. Serpentinite	13,370 - 13,590 p.s.i.
2. Serpentinite with asbestos fibre	3,150 p.s.i.
3. Argillite	6.870 - 28,110 p.s.i.

5.6.1. Comparison of Explosive Techniques

A comparison table of explosive techniques and requirements is shown below;

Description	Argillite	Serpentinite
1. Explosives a) Type	Explosive contamination not a problem.	Explosive contamination a problem - unable to separate explosive wrappers from asbestos fibres in mill process.
	No restriction on type of explosives that can be used.	Restricted as to type of explosives that can be used.
b) Amex	Okay to use if holes are not wet; unable to use due to wet conditions. Aluminum additive okay.	Explosive energy too high; cannot reduce energy output by reducing density without contaminating ore. Not suitable for baby arch development.
c) Cilgel	Okay to use, no restrictions. Aluminum additive okay	Suitable for use, magnetic clip ties have to be used to remove plastic wrapper from ore. Cannot use aluminized Cilgel due to contamination problems.

Description	Argillite	Serpentinite
d) Water Emulsion (Magnafrac)	Okay, suitable for use.	Suitable for use. Use magnetic clip tie to remove plastic wrappers. Cannot use aluminized Magnafrac due to ore contamination.
e) Xactex (Ore shearing Explosive)	Okay, suitable for use.	Not suitable for use as plastic connectors and wrappers contaminate ore. Only used in controlled conditions. Suggest replacement with Primer-Flex or 4 grain ft ignitor cord.

2. Explosive Accessories:

Description	Argillite	Serpentinite
a) Nonels	Okay, suitable for use.	Not suitable, plastic tubes and connectors contaminate ore. Unable to separate in mill.
b) Electrical Blasting Caps	Okay, suitable for use.	Suitable for use with steel lead wires that are magnetic. Can be removed in Mill process.

Description	Argillite	Serpentinite
c) B-Line	Okay, suitable for use.	Not suitable, contaminates ore, unable to remove in mill process.

5.6.2. Explosive Consumption

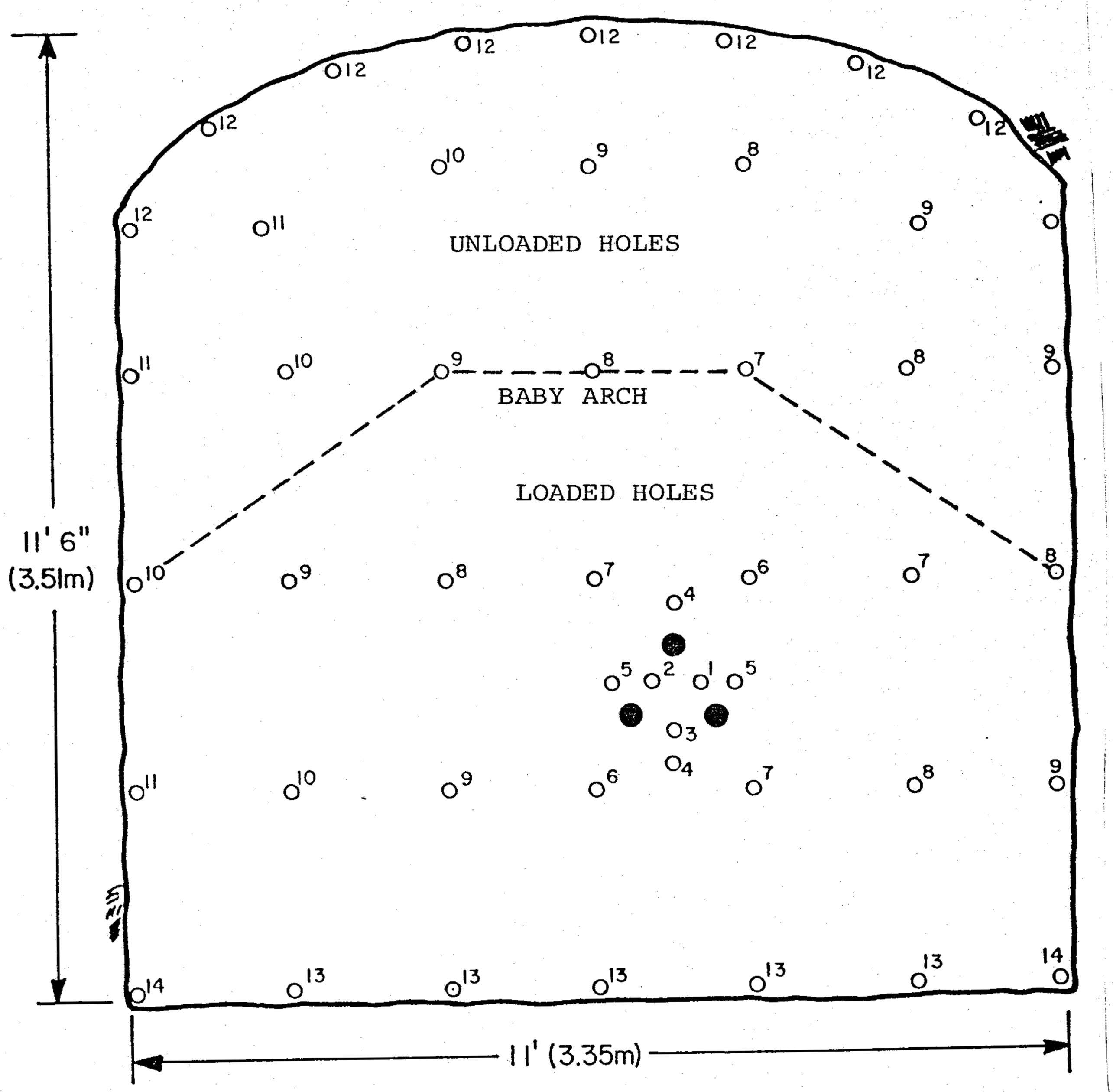
Ground	Explosives						Nonels and Electric Blasting Caps	
	Cigil & Magnafrac		Amex		Xactex			
	Kg	Kg/M	Kg	Kg/M	Kg	Kg/M	Total	Number/M
Argillite	1,770	20.55	850	10.62	239	2.84	1,160	13.84
Serpentinite	1,577	11.05	0	0	247	1.73	1,998	14.00

7.4.1 Description of Serpentinite Blasting Operation. Blasting as per the Baby Arch Approach

Procedure of blasting the Baby Arch for a Serpentinite development round is as follows:

(refer to Figure 6)

1. Drill round off as shown on Figure 6
2. Load round as shown on Figure 6
3. Blast round.
4. (a) Inspect round after blasting. If blast has caused the back of the round to cave up to the perimeter holes then no additional explosive is required as the back height of the drift round has been obtained.



STANDARD DRILL PATTERN

BABY ARCH BLASTING - HOLE LOADING PATTERN

1. Adjust to suit field conditions.
2. Alternate cut from side to side.
3. Arch holes (#12) to be blasted with Xactex if required.

FIGURE 6

4 (b) If blast round has not caused the ground to cave up to the perimeter holes, then trimming of the round is required with the use of Xactex.

5. Once the round blasting process has been completed 3 - 4 scoops of muck are removed to permit shotcreting of the back.

6. Shotcrete back and walls as far as the muck pile.

7. Muck out remaining round.

8. Shotcrete remaining exposed ground and follow ground support procedures for Serpentinite.

This procedure for blasting a Serpentinite round worked successfully for the exploration development program.

Consideration has been given to eliminating the baby arch blasting process but poor Serpentinite ground conditions have precluded this development. The baby arch blasting method provides a margin of safety, working in conjunction with the structural integrity of the ground and with the spiles to maintain the opening. Elimination of the baby arch blasting process was attempted in previous exploration development programs, with and without spiles. Results have not been satisfactory, with break back above the back and side walls causing serious ground control problems. Once serious problems such as rat holing have commenced, it is difficult to bring the ground back to a controlled condition. It is imperative that ground control measures such as described earlier are employed in order that safe and economical advance is maintained.

5.7 Surveying

Standard practice for surveying during advance on the 1415 m level involved front and back lines for a first slash being marked at the start of a new heading and, if necessary, remarked for the second slash. On curves reference line plugs were set and when the new bearing was reached, a pair of line plugs and spads were set on line. When the face was approximately 30 - 35 m beyond the first set of plugs, or when a change in bearing was necessary, a new set of line plugs was set approximately 5 m from the face. Off sets were taken after advance to determine the shape of the drift.

A contract surveyor was brought up to the project site to tie the undergrond coordinate system into the Mill coordinate system. During the 1986 Development program the underground development was tied into the Mine coordinate system. A review of the relationship between the mine and mill coordinate system revealed a discrepancy that dictated the change over. Figure 7.

6.0 Geology

6.1 Stratigraphy

The McDame Asbestos Deposit occupies the central part of a serpentinite body lying within rocks of the Sylvester Group (Figure 8) of Northern British Columbia. The Serpentinite has not been dated but the hangingwall sediments have been dated as Lower Mississipian to Permian age (345-250 my) by fossil conodonts.

The Sylvester Group is an allochthonous stratigraphic sequence as it has been thrust from its place of origin to its present location by plate tectonic processes. In general, the sequence consists of numerous fault slices of ocean floor sediments, basalts and ultramafic intrusions that were imbricated amongst themselves by thrust faulting, then thrust and uplifted as a mass onto the continental platform in the Mezozoic Era. The units now dip easterly at around 40 degrees because of regions warping.

The McDame Serpentinite is from 20 to 100 m above the basal thrust surface within a predominantly argillite sequence. In the hangingwall above the Serpentinite are some basaltic layers that gradually increase to about 50% of the rock at more than 30 m above the contact. The Serpentinite itself consists of an outer mantle of older darker green Serpentinite with an overprinting green Serpentinization of the core that contains most of the ore grade asbestos.

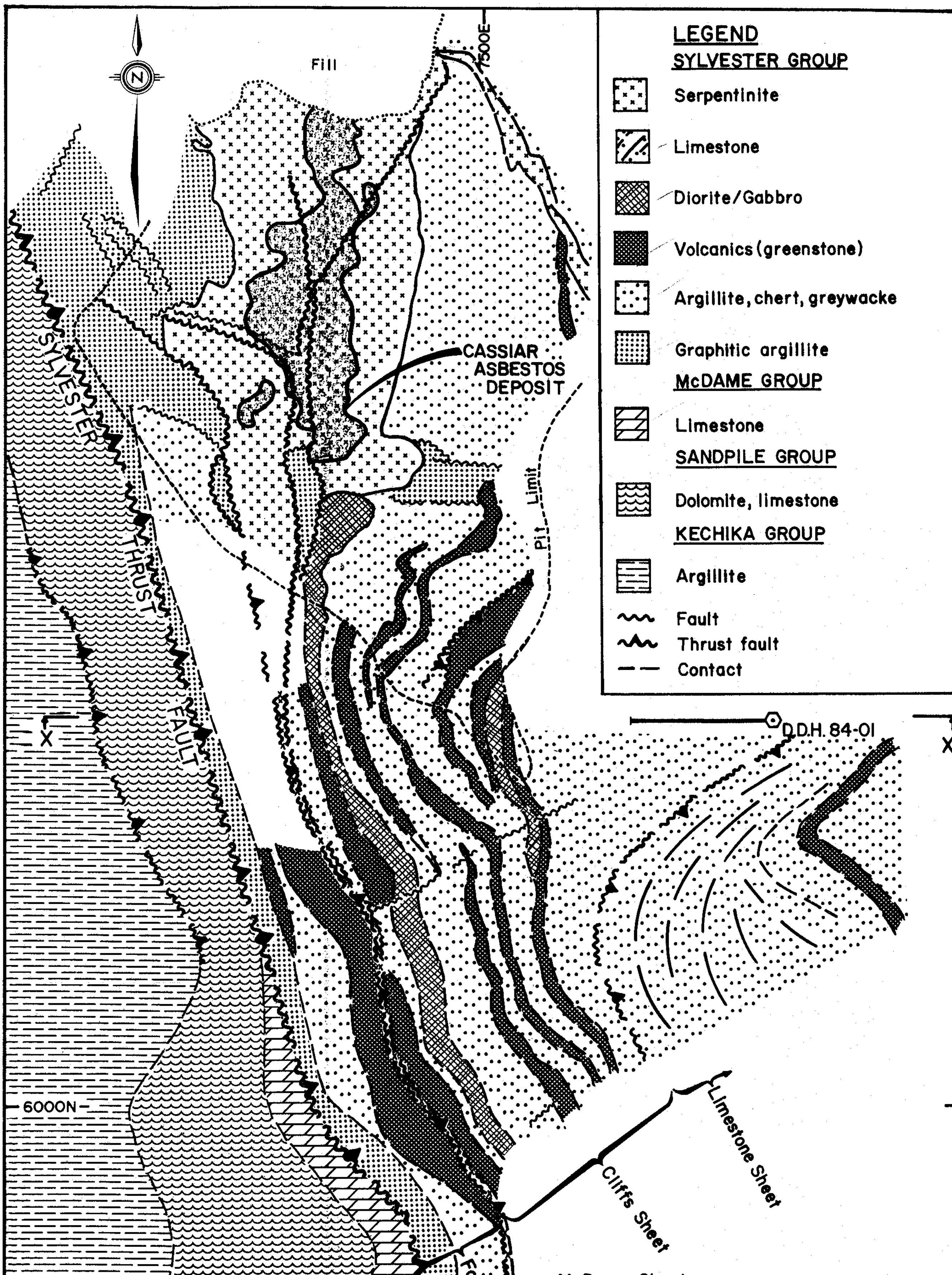


Figure 8
LOCAL GEOLOGY
X-X' = SECTION 6540N

Between the Serpentinite and hangingwall Argillite is an alteration zone up to 5 m thick, through which Serpentinite grades into Argillite. The alteration is mainly tremolite-amphibolite. It is overprinted by talcose alteration, which also occurs in the Serpentinite.

The footwall Serpentinite contact is predominantly fault bounded with local remnants of alteration.

Below the Sylvester Group are Paleozoic rocks of the North American platform. They are a sequence of formations of generally alternating Argillite and carbonate, the oldest being Late Precambrian. The McDame workings intersect the three upper Groups which go from Middle Cambrian to Upper Devonian (550 - 350 my). The oldest, the Kechica Group, is a black Argillite that is overlain by the Sandpile Group of dolostone and quartzite and the McDame Group of dolostone and limestone capped by black Argillite. All three Groups have been strongly deformed by thrust faulting. Close to the McDame Deposit the McDame carbonate rocks have been faulted out completely and the Sandpile thinned severely. The black Argillite capping the McDame Group formed a relatively soft lubricating layer over which the Sylvester Group was thrust.

A granitic batholith that intruded the Paleozoic rocks in the Cretaceous Period (100 - 70 my) lies about 1 km west of the McDame deposit. The heat of the intrusion caused contact metamorphism of the Kechica Argillite to hornfels at the 1415 m portal.

6.2 Structure

The geological history and development of the McDame deposit is dependent upon and related to three important structural events. The first of these occurred during Devonian and Mississipian times when the Sylvester Group basic volcanic and sedimentary rocks were being assembled on the ocean floor. This was followed by movement eastwards and then northwards of a segment of these rocks, now known as the Sylvester Allochthon, to its present site on the continent. The third and final event was the emplacement of the granitic Cassiar Batholith to the immediate west of the Allochton in Cretaceous times (Harms, 1984, 1985).

Regional and detailed structural studies in the Cassiar area and its surroundings have shown that intense tectonic and structural deformation accompanied these three events, and that from an economic standpoint it is the two later episodes that are important, for they are the events that are associated with the development of the asbestos fibre. This occurred in two distinct phases. The first phase is related to motion on faults similar to motion in the allochthon as a whole, the second phase is related to a local motion that was probably induced by the emplacement of the batholith. In addition, it has been found that lineations in the allochthon are northwest trending, indicating a northeasterly direction of motion.

On a local scale, underground mapping in the adits and logging of diamond drill core has disclosed the presence of numerous minor and several major faults and fault systems. These have occurred over an extensive period of time, concomitant with the main tectonic events, and display varying age relationships, both with respect to other faults with respect to the fibre. Four main sets of faults have been recognized, a set that strikes north-south and dips to the east, a set that strikes northwesterly and strikes steeply to the northeast, a set that strikes eastwest and dips to the south, and a set that strikes northeasterly and dips very steeply to the northwest.

Several of these faults are post-fibre in age and are of decided economic importance, for they either dislocate and form boundaries to the orebody, or else, on a small scale, dislocate and form the boundaries between the individual blocks of ore and waste within the orebody itself. The most important of these faults are a northwesterly striking fault that forms a hangingwall contact, a north-south striking fault that forms a footwall contact and a northeasterly striking fault that is thought to throw the orebody downwards in the southeast area of the deposit. Geological structures intersected in the 7610 E Drift are illustrated in Figure 9.

Harms, T. 1985. Pre-emplacement thrust faulting in the Sylvester Allochthon, northeast Cry Lake map area, British Columbia. Current Research Part A, Geolgical Survey of Canads Paper 85-1A, p. 301-304.

Harms, T. 1984. Structural style of the Sylvester Allochthon, northeastern Cry Lake map area, British Columbia. Current Research, Part A, Geolgical Survey of Canada Paper 84-1A, p. 109-112.

7.0 PERSONNEL:

On a project of this magnitude, many people are involved. Those direcly involved with onsite work or in report preparation are listed below. All contributed to the completion of the project. Many other individuals offered advice and assistance and their help is appreciated though for brevity, their names are not listed.

Person	Responsibility
Mr. T. Carew	Project Supervisor
Mr. K. C. Minty, P. Eng.	Field Supervisor, Assistant to Mr. Carew
Mr. D. Laubscher	Consultant
Mr. R. Tyne	Geology, structure
Mr. D. Kenny	Geology, structure
Mr. P. MacRae	Assistant to Mr. K. C. Minty
Mr. C. Turek	Surveying, Drafting
Mr. W. Day	Surveying, Drafting
Mr. G. Valgardson	Site Accounting
Mr. J. McGill	Site Accounting
Mrs. E. Cavanagh	Clerk
Mrs. B. Minnaar	Word Processing

and the personnel of:

Cassiar Mine
Canadian Mine Development Limited
Mathews and Associates
Steffen, Robertson, and Kirsten
B.B.T. Hardy and Associates

8.0 COST STATEMENT

During the course of the program, an accounting system was set up on site to monitor costs and changes on an ongoing basis. After implementation of the accounting system, all purchases were made by purchase order. Invoices were sent to the project site for authorization by project supervisors and records were maintained on site. Authorized invoices were paid from the project site except for the underground contractor's invoice.

Costs of services were accrued on a continual basis to allow estimation of total expenditures incurred at any time.

Invoices paid were tracked by computer in Vancouver. A generalized list of costs for the project are:

Cost Statement

		$
1. Contract Mining Services		
Labour		
Supervision		93,000
Equipment Operators		183,000
Labourers		101,000
Miners		188,000
Tradesmen		121,000
Other		
Equipment Rental		491,000
Air Fares (Rotations)		47,000
Camp/Accomodations		286,000
	Subtotal	1,510,000
2. Cassiar Mining Costs		
Labour		
Equipment Operator		17,960
Tradesmen		9,760
Labourer		21,540
Secretarial		4,860
Technician		18,680
Geology		3,500
Accounting		110
Goods and Services		
Meals/Accomodation		1,590
Transport		10,930
Equipment Rental		54,250
Consulting		10,850
Communications		2,190
Supplies (Ground Support and U/G Explosives)		433,450
Power		50,400
Explosives (Cassiar Open-pit magazine)		23,570
Union dues (Contractor)		10,700
Shops/Fuel/FVT		95,280
	Total	2,279,620

9. ACKNOWLEDGEMENTS

On a project of this size many people are responsible for a successful completion and compilation of the results. All those listed in Section 6. Personnel are acknowledged for their contributions.

Receipt of a Financial Assistance for Mineral Exploration (FAME) grant provided funds to supplement the program. The Ministry of Energy, Mines and Petroleum Resources of British Columbia is gratefully acknowledged for their incentive program which has directly aided and extended exploration on the McDame project.

APPENDIX 1

SHOTCRETE TESTING

Hardy BBT Limited
CONSULTING ENGINEERING & PROFESSIONAL SERVICES

VA-01060

September 28, 1987

CASSIAR MINING CORPORATION LTD.,
CASSIAR, B.C.
VOC 1EO

ATTENTION: Mr. Keith Minty, P.Eng.,
Senior Chief Engineer

Dear Sirs:

Re: Evaluation of Shotcrete, McDame Exploration Adit

As requested, Hardy BBT has carried out an evaluation of the shotcrete application presently underway in the McDame Exploration Adit. The field evaluation was carried out on September 18 through 20, 1987. This evaluation included:

1. Observation of shotcrete application techniques.

2. Demonstration of shotcreting techniques to mine personnel with a view to improving shotcrete productivity and quality and reducing rebound.

3. Inspection of in-place shotcrete and assessment and testing of cores taken between stations 100LP+15 and 101+12.

4052 GRAVELEY STREET, BURNABY, BRITISH COLUMBIA V5C 3T6 TELEPHONE (604) 294-3811 FAX (604) 294-4664 TELEX 04-354841
GEOTECHNICAL AND MATERIALS ENGINEERING — ENVIRONMENTAL, MATERIALS AND CHEMICAL SCIENCES
BONNYVILLE CALGARY EDMONTON FORT McMURRAY KAMLOOPS LETHBRIDGE LLOYDMINSTER MEDICINE HAT
PEACE RIVER PRINCE ALBERT PRINCE GEORGE RED DEER REGINA SASKATOON VANCOUVER

1. During the period September 18 to 20, the day and afternoon shifts were observed shotcreting. While it was clear the crews were generally familiar with shotcreting procedures, there were several areas where improvement in shotcreting technique is required for optimum shotcrete quality and materials efficiency. These items are listed below.

   (a) The relationship between air pressure in the hose and distance of the nozzle from the face were, at most times of observation, incorrect. Table 1 which follows itemizes many of the problems observed and the remedial action required for correct shotcrete application.

TABLE 1

PROBLEM	ACTION	CORRECT APPLICATION
Shotcrete builds up on face of mesh creating voids behind.	Increase air pressure in gun and/or move nozzle closer to the face (See ACI 506R-85 Fig.8.4)	Shotcrete should build from behind the mesh to encase it; build-up on the front face of the mesh should be avoided.
Shotcrete peels off rock face. Caused by insufficient compaction energy.	Increase air pressure and/or move nozzle closer to the face. (Assuming water content is correct.)	Fresh shotcrete in excess of 6 inches in thickness should adhere well to a clean rock surface when air pressure and distance of nozzle from face are correct.
Freshly applied shotcrete is blown off the surface.	Increase nozzle distance from face and/or decrease air pressure. (Assuming water content is correct.)	Shotcrete should not be blown off the surface if the nozzle is held at the correct distance from the face.
Excessive coarse aggregate particles rebound from plastic material.	Increase air pressure and/or increase moisture content of shotcrete i.e. shoot at "wettest stable consistency".	Coarse aggregate particles should be completely embedded in the plastic matrix.

b) Shooting distances were generally too great (in excess of 10 feet in some cases) and resulted in poor visibility, difficulty in accurately directing the spray pattern for optimum results and excessive overspray and rebound. The distance of the nozzle from the face should normally be between 3 and 5 feet.

c) The water content of the shotcrete was generally too dry, resulting in excessive rebound, overspray and dust. The optimum water content of shotcrete is the wettest consistency possible without sloughing.

d) Control of shooting angle was poor. This resulted in excessive rebound and overspray and poor compaction or voids in "shadow" areas.

For proper control of shooting angle when shotcreting blocky, fractured rock, it is important to keep the shooting distance within the range of 3 to 5 feet in order to minimize the diameter of the spray pattern. It is equally important to manipulate the nozzle to first fill crevices and joints. The nozzle stream should be oriented at right angles to the face of the actual rock surfaces being shot, rather than at a general 90° to the tunnel wall.

2. Correct shotcreting techniques were demonstrated to both the day and afternoon shifts on September 18 and 20, 1987.

3. The in-place shotcrete appeared visually to be generally acceptable. However, a number of cores extracted from the shotcreted adit wall showed many manifestations of poor shooting technique. These deficiencies included small voids, inclusion of rebound and

overspray and some areas of poor bond between layers (sand lenses). Photographs of the extracted cores are given in Appendix A. Results of compressive strength and boiled absorption and permeable voids tests conducted on these cores are given in Appendix B and summarized in Table 2.

TABLE 2: CORE TEST DATA

LOCATION	AGE (DAYS)	COMPRESSIVE STRENGTH (MPa)	BOILED ABSORPTION % (2)	VOLUME OF PERMEABLE VOIDS % (3)
100LP+15	26	-	9.1	19.9
100LP+20	24	43.9	-	-
100LP+25	21	41.1	9.8	20.7
101+00	20	36.4	8.6	18.5
101+5	17	21.8 (1)	8.6	18.7
101+8.3	15	27.0	10.0	21.1
101+12	12	28.7	9.8	20.7

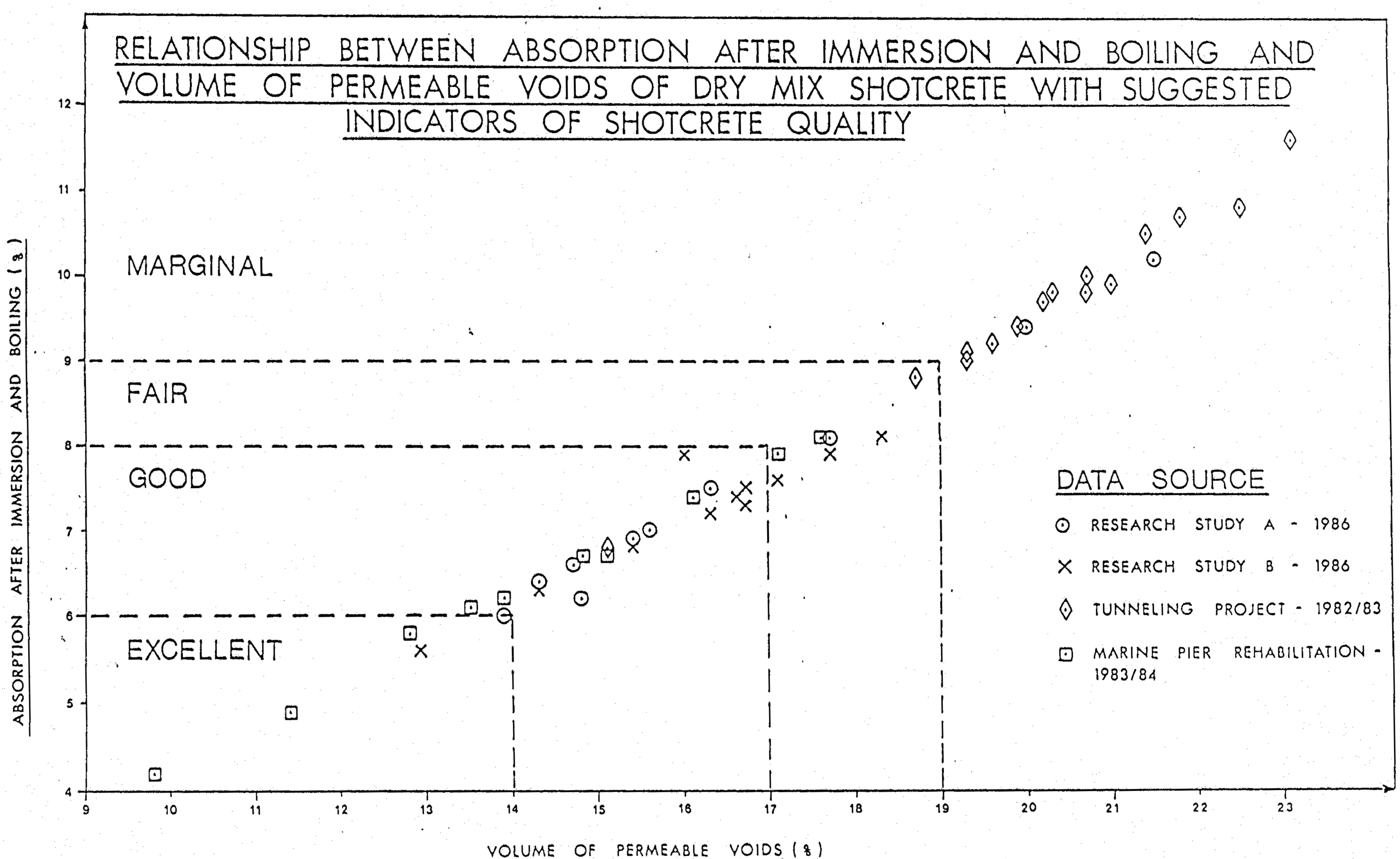
Notes:

(1) Small voids noted in core caused by poor shooting technique. It is doubtful whether this core will achieve a 28 day strength of 35 MPa.

(2) It has been our experience based on analysis of results from various shotcrete projects that boiled absorption values of:

<6% represents excellent quality shotcrete,
6% - 8% represents good quality shotcrete,
8% - 9% represents fair quality shotcrete,
>9% represents marginal quality shotcrete.

(See appended graph from paper by: Morgan, D.R., Neill, J., McAskill, N. and Duke, N., "Evaluation of Silica Fume Shotcrete". International Workshop on Silica Fume in Concrete, Montreal, May, 1987.)



RELATIONSHIP BETWEEN ABSORPTION AFTER IMMERSION AND BOILING AND VOLUME OF PERMEABLE VOIDS OF DRY MIX SHOTCRETE WITH SUGGESTED INDICATORS OF SHOTCRETE QUALITY
MARGINAL
FAIR
GOOD
EXCELLENT
DATA SOURCE
RESEARCH STUDY A - 1986
RESEARCH STUDY B - 1986
TUNNELING PROJECT - 1982/83
MARINE PIER REHABILITATION - 1983/84
ABSORPTION AFTER IMMERSION AND BOILING (%)
VOLUME OF PERMEABLE VOIDS (%)
4
5
6
7
8
9
10
11
12
9
10
11
12
13
14
15
16
17
18
19
20
21
22
23

FIGURE 20:

(3) It has been our experience based on analysis of results from various projects that volume of permeable voids of:

<14% represents excellent quality shotcrete,
14% - 17% represents good quality shotcrete,
17% - 19% represents fair quality shotcrete,
>19% represents marginal quality shotcrete.

(See appended graph.)

CONCLUSIONS

Based on our observations of shotcrete application between September 18th and 20th, and inspection and testing of cores extracted on September 19, 1987, the following conclusions are drawn:

1. The shotcrete application techniques observed were not consistent with good shotcreting practice. Proper control of shooting angle, distance of the nozzle from the face, air pressure and water content, will improve the quality of the in place shotcrete, reduce the amount of rebound, and increase the productivity and economy of the operation.

2. The strength of the six cores tested, with the exception of the core at Station 101+5, is expected to meet a 28 day strength specification of 35 MPa. It should, however, be remembered that core strengths, by virtue of the trimming process, will not reflect all of the defects within the overall length of a core.

3. Coring through the applied thickness of shotcrete with a diamond drill and inspection of the extracted cores revealed in many cases a marginal standard of shotcrete workmanship. This condition was not readily evident from simple visual examination of

exposed shotcrete surfaces. The boiled absorption and permeable voids data on extracted cores reflects the fair to marginal quality of the shotcrete application. Thus a continuing program of coring to check the quality of the in situ shotcrete is recommended.

We trust this report meets your immediate requirements. Please call if you have any queries or if we can be of any further assistance.

Yours truly,

Hardy BBT Limited

Per:

N. McAskill, A.Sc.T.,
Senior Materials Technologist

Per: D. R. Morgan

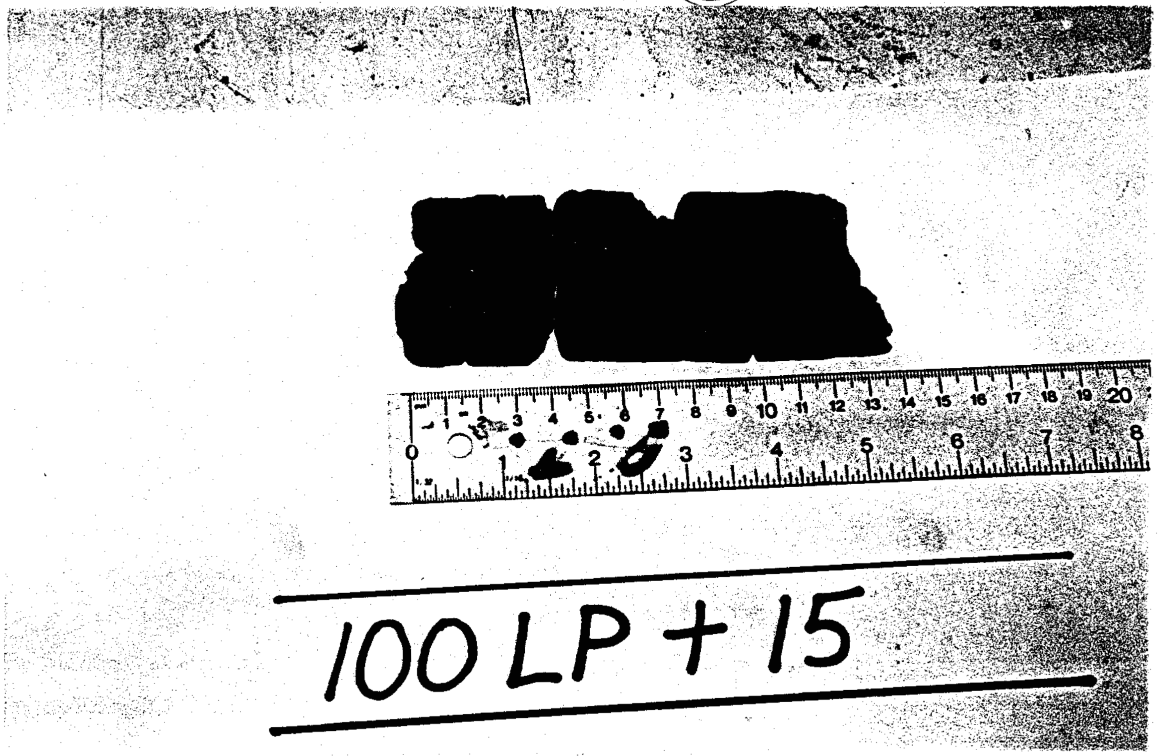
D.R. Morgan, Ph.D., P.Eng.,
Manager, Materials Engineering Division

NMcA,DRM:cg,nmd

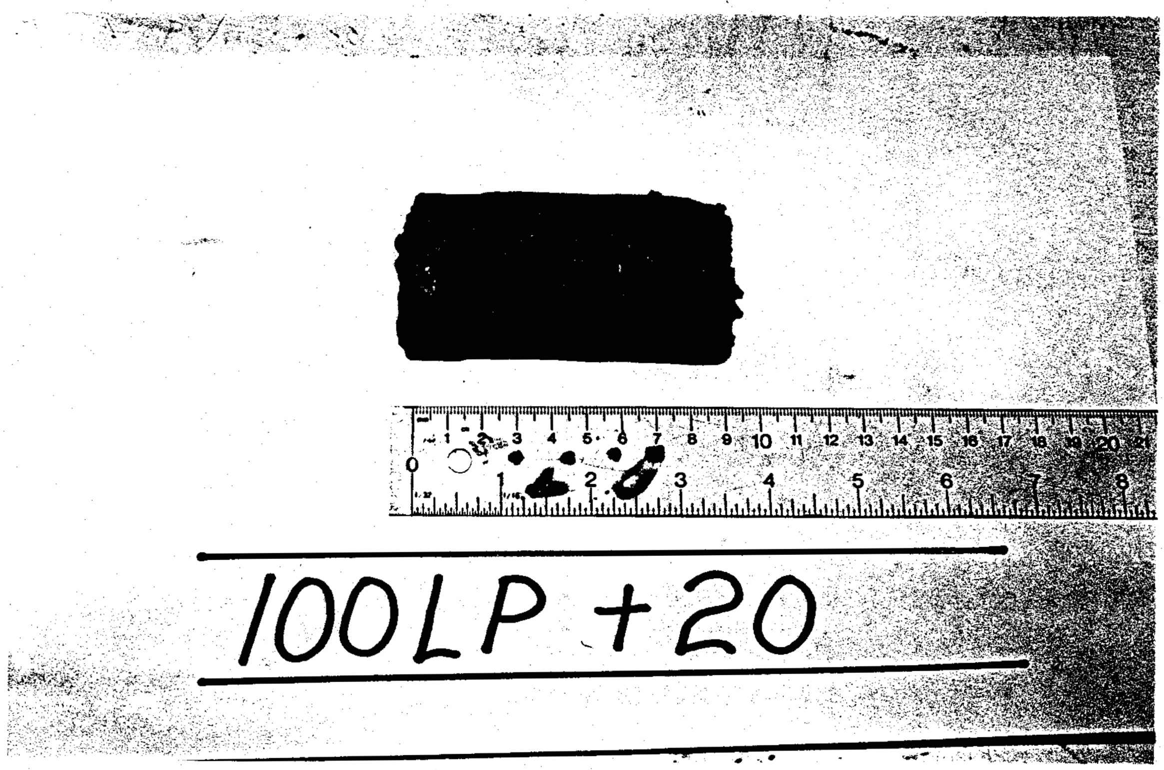
Enclosures

APPENDIX A

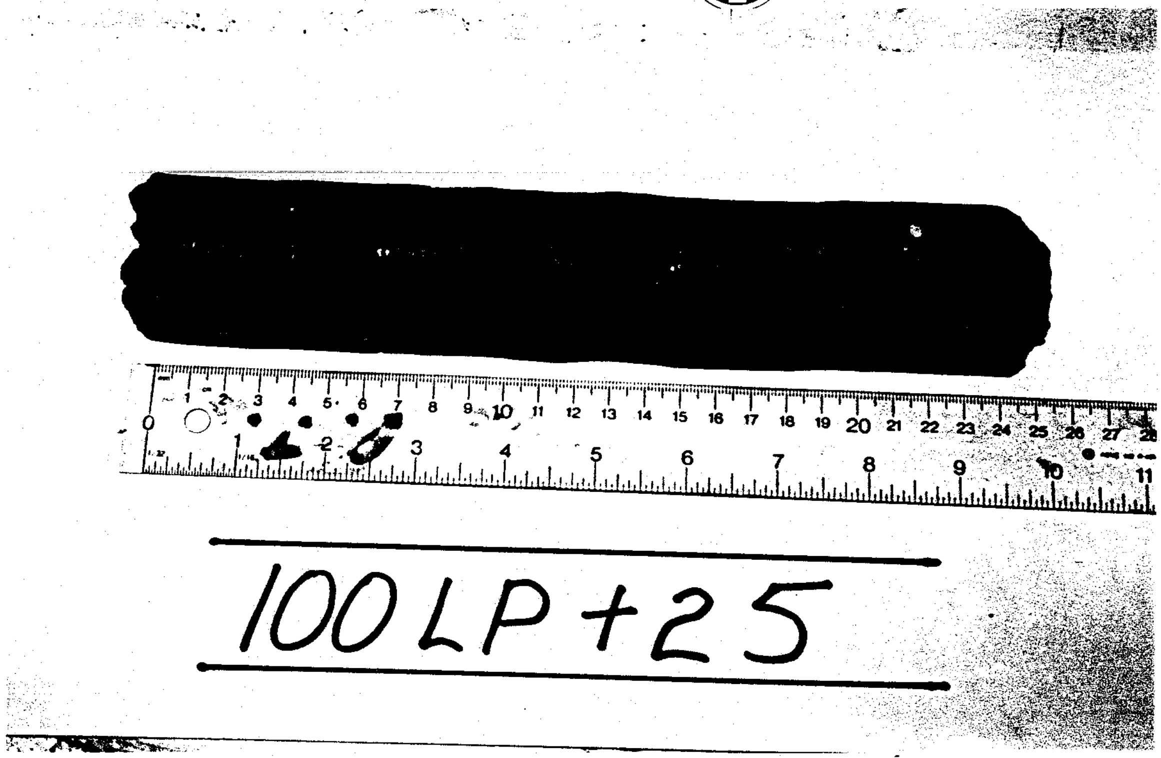
PHOTOGRAPHIC DOCUMENTATION



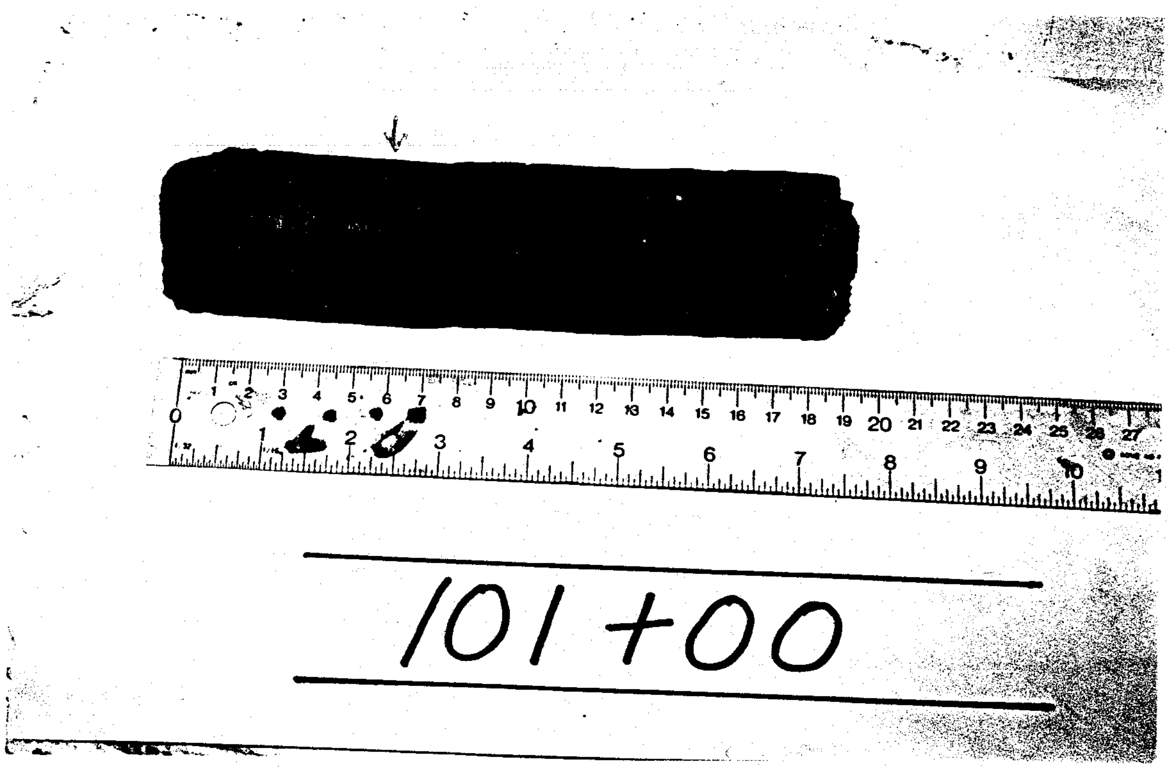
PHOTOGRAPH 1: Core extracted from 100 LP + 15 left. The fragmented sections are likely a result of poor surface preparation between lifts.



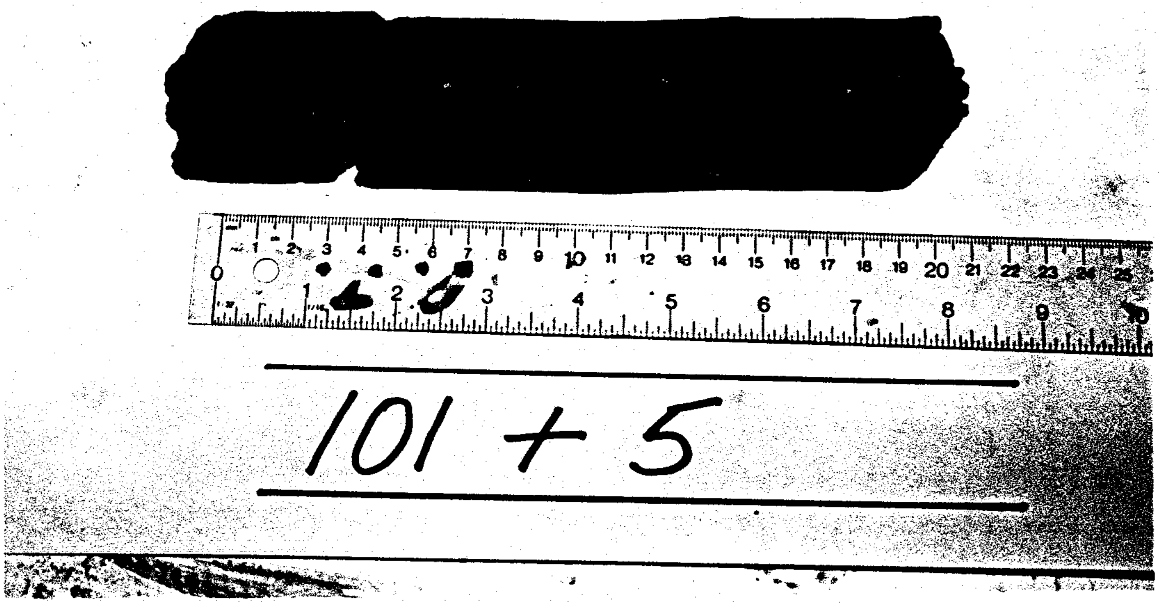
PHOTOGRAPH 2: Core from 100 LP + 20 left. Note void approximately 1 1/2 cm from face.



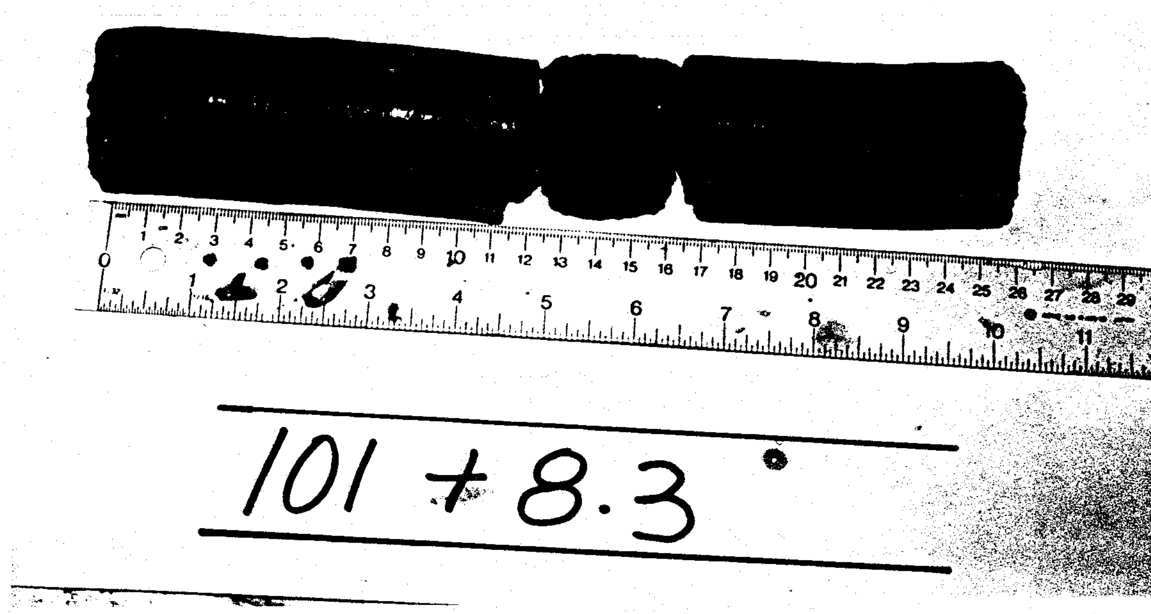
PHOTOGRAPH 3: Core extracted from 100 LP + 25. The core is generally sound with exception of the crack between 5 and 9 cm from the surface.



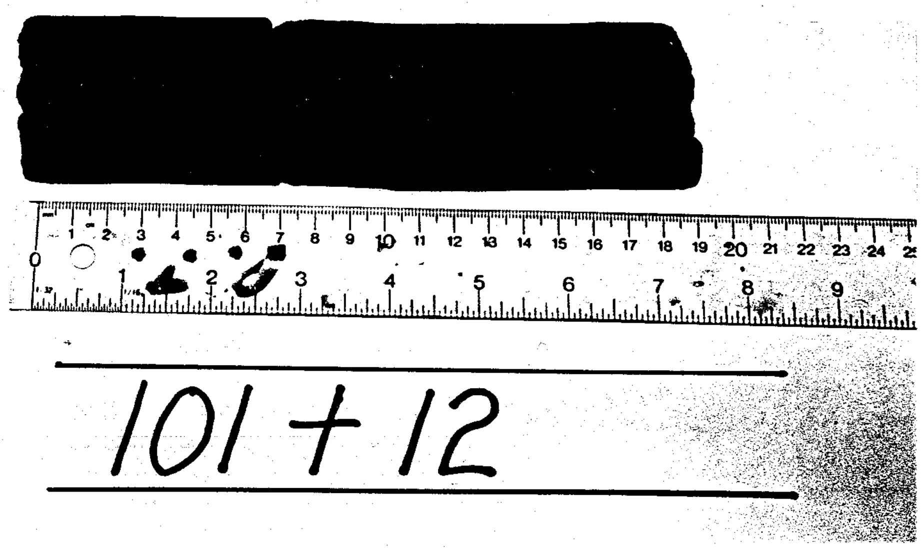
PHOTOGRAPH 4: Core extracted from 101 + 00 left. The core is sound with exception of a crack 10 cm from the surface. The arrow denotes a normal joint between two shotcrete passes.



PHOTOGRAPH 5: Core extracted from 101 + 5 left. The joint at 3 cm depth was poorly consolidated and contained some overspray. The remainder of the core was of acceptable quality.



PHOTOGRAPH 6: Core extracted from 101 + 8.3 left. Note the area of lower strength ravelled shotcrete in the middle of the core.



PHOTOGRAPH 7. Core extracted from 101+12 left.
Note rod joint at 7 cm from face.

APPENDIX B

COMPRESSIVE STRENGTH OF CONCRETE CORES TEST DATA

HARDY ASSOCIATES (1978) LTD.
CONSULTING ENGINEERING & PROFESSIONAL SERVICES
Certified Concrete Testing Laboratory* In Accordance With STD A283

CALGARY* DAWSON CREEK EDMONTON* FORT McMURRAY LETHBRIDGE MEDICINE HAT PRINCE GEORGE RED DEER SASKATOON VANCOUVER* WINNIPEG

COMPRESSIVE STRENGTH OF CONCRETE CORES

TO:
CASSIAR MINING CORPORATION LTD.,
CASSIAR, B.C.
VOC 1E0

ATTENTION: Mr. Keith Minty, P.Eng.,
Senior Chief Engineer

OFFICE BURNABY
PROJECT NO. VA-01060
CLIENT
CC:

PROJECT McDAME EXPLORATION ADIT - EVALUATION OF SHOTCRETE

DATE CAST
DATE CORED 87-09-19
DATE TESTED 87-09-22

SPECIFIED STRENGTH 35.0 MPa
AGGREGATE SIZE 10 mm
COMMENTS

CORE NO.	LOCATION	LENGTH (mm)	DIAMETER (mm)	DENSITY (kg/m³)
C2	100 LP + 20	55	43	
C3	101 + 12	60	43	
C4	100 LP + 25	80	43	
C5	101 + 00	60	43	
C6	101 + 5	72	43	
C7	101 + 8.3	70	43	

CORE NO.	AGE (days)	LOAD (kN)	AREA (m^2)	COMPRESSIVE STRENGTH (MPa)	LENGTH / DIAMETER	CORRECTION FACTOR	CORRECTED COMPRESSIVE STRENGTH (MPa)
C2	24	68.5	0.00145	47.2	1.28	.93	43.9
C3	12	43.8	0.00145	30.2	1.39	.95	28.7
C4	21	60.2	0.00145	41.5	1.86	.99	41.1
C5	20	55.6	0.00145	38.3	1.39	.95	36.4
C6	17	32.6	0.00145	22.5	1.67	.97	21.8
C7	15	40.4	0.00145	27.9	1.63	.97	27.0

TECHNICIAN D. Hatch, A.Sc.T. REPORT CERTIFIED N. McAskill, A.Sc.T. DATE 87-09-23

APPENDIX 2

STATEMENT OF QUALIFICATION

STATEMENT OF QUALIFICATIONS

Keith C. Minty, do hereby certify:

1. I am a Mining Engineer residing at 125 Hunt Street, Box 52, Cassiar, B.C. VOC 1EO.

2. I am currently employed by Cassiar Mining Corporation, Cassiar Mine, Cassiar, B.C. VOC 1EO as the Chief Engineer.

3. I graduated from Queen's University at Kingston, Ontario in 1978 with a Bachelor of Science in Mining Engineering.

4. I am registered as a Professional Engineer in the Association of Professional Engineers for the Province of British Columbia. I am also registered as a Professional Engineer in the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories.

5. Work on the Underground project was done under my direct supervision.

Respectfully

Cassiar Mining Corporation

Keith C. Minty, P. Eng.

Dated at Cassiar, British Columbia

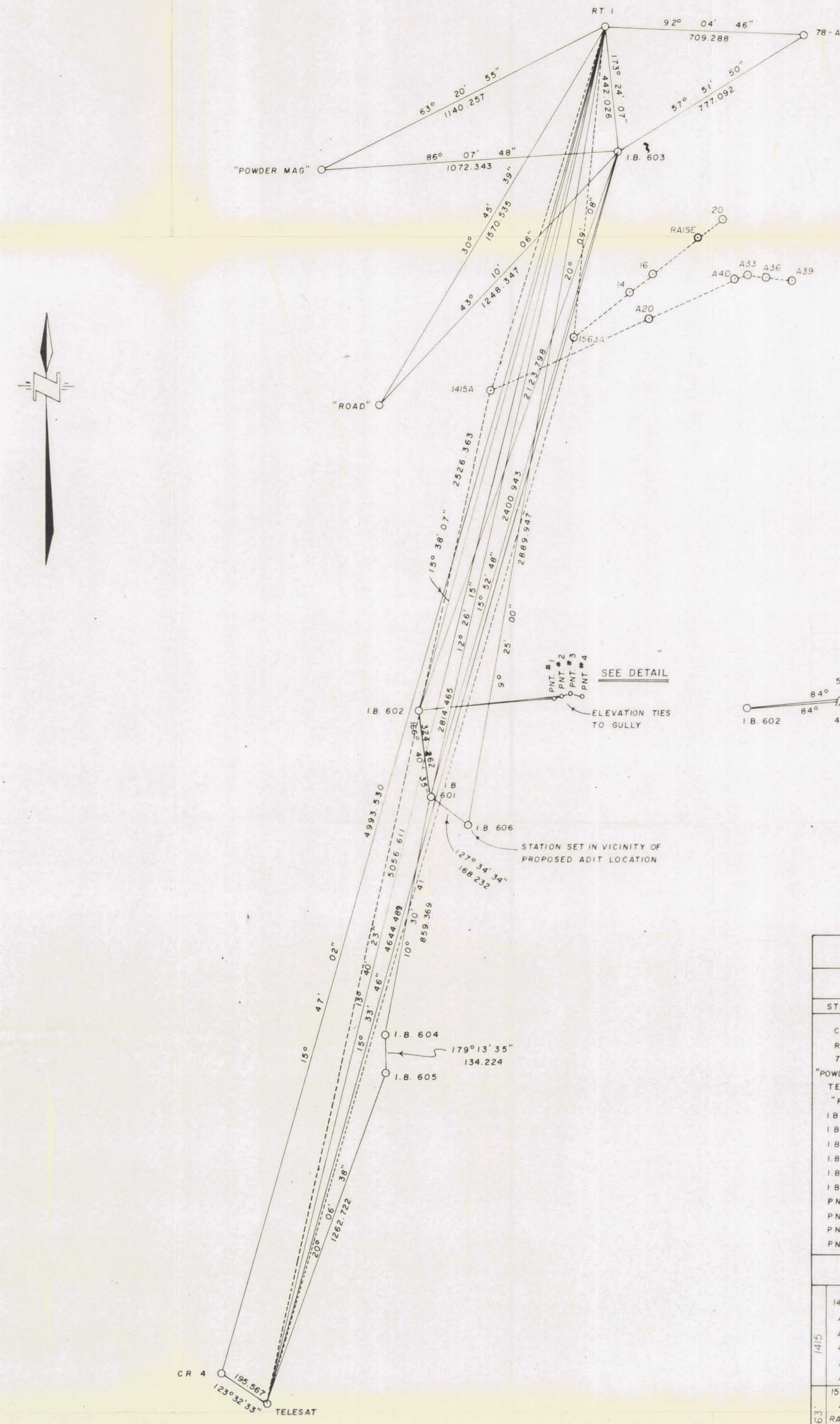
this Friday, 19 day of February 1988

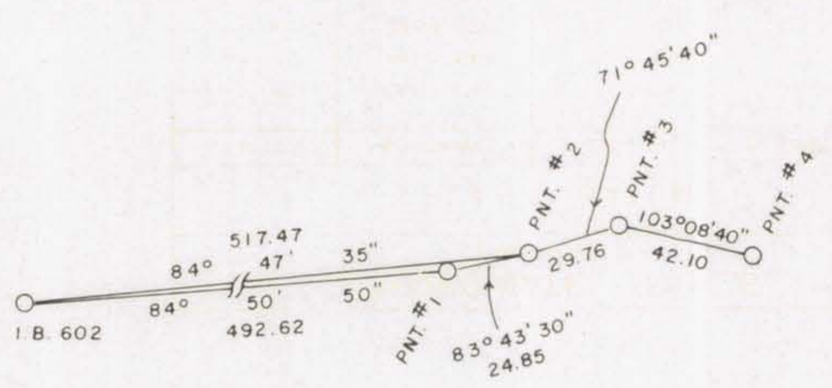
FIG. 4

CASSIAR MINING CORPORATION — McDAME EXPLORATION

Ventilation System of Cassiar's Mcdame Exploration

DRN W.J.D.	
CKD	
APR	
1500 2000-4	
1: 2000m.	SEPT. 24 1987





CO-ORDINATE TABLE (METRIC)			
PLANT DATUM			
STATION	NORTHING	EASTING	ELEVATION
CR 4	2685.297	5547.287	1080.58
RT 1	7490.544	6905.575	1920.32
78 A	7464.808	7614.396	1967.71
"POWDER MAG"	6979.069	5886.468	1379.20
TELESAT	2577.235	5710.287	1079.86
"ROAD"	6140.969	6102.315	1291.19
I.B. 601	4742.130	6299.414	1175.40
I.B. 602	5057.664	6224.688	1207.52
I.B. 603	7051.446	6956.366	1803.56
I.B. 604	3897.182	6142.639	1158.52
I.B. 605	3762.970	6144.451	1148.43
I.B. 606	4639.540	6432.745	1178.10
PNT. 1	5101.9	6715.3	1263.1
PNT. 2	5104.6	6740.0	1268.2
PNT. 3	5113.9	6768.3	1276.8
PNT. 4	5104.4	6809.3	1290.1
REVISION 1			
1415: 1415A	6189.726	6530.136	1393.53
1415: A20	6460.072	7091.943	1408.15
1415: A40	6610.818	7396.804	1415.14
1415: A33	6630.104	7440.984	1415.40
1415: A36	6629.079	7506.943	1416.96
1415: A39	6629.016	7599.539	1418.08
1563: 1563A	6374.586	6816.700	1563.67
1563: 20	6808.194	7344.063	1571.08
1563: RAISE	6737.377	7258.378	1570.84
1563: 16	6605.085	7096.397	1569.66
1563: 14	6539.267	7016.130	1569.47

GEOLOGICAL BRANCH
ASSESSMENT REPORT

16,776

100 50 0 100 200 300 400 500 1000 m
SCALE, 1:10000

NOTES:

1. ALL DISTANCES ARE IN METRES AND DECIMALS THEREOF AND ARE REDUCED TO A DATUM OF 3545.2 FEET (1080.58 METRES) RADIUS OF CURVATURE = 6,370,320 METRES

   PRIOR TO COMPUTING CO-ORDINATES ON THIS DATUM THE MEASURED DISTANCE MUST BE REDUCED USING THE FOLLOWING:

   $$CD = \frac{HD(6,370,320 + 1080.58)}{(6,370,320 + H)}$$

   WHERE HD = HORIZONTAL DISTANCE
   H = AVERAGE ELEVATION BETWEEN THE TWO STATIONS IN METRES
   CD = CORRECTED DISTANCE AT DATUM

   PRIOR TO LAYOUT HD MUST BE CALCULATED

2. CO-ORDINATES ARE PLAIN RECTANGULAR AND ARE REFERRED TO PLANT DATUM.
3. ELEVATIONS AND CO-ORDINATES ARE BASED ON CONTROL POINT "CR 4" AS RECEIVED ON SITE.
4. BEARINGS ARE DERIVED BY CONVERTING RT 1 CO-ORDINATES FROM MINE TO PLANT DATUM USING DATA RECEIVED ON SITE AND INVERSING BETWEEN CR 4 AND RT 1

   NOTE: INVERSE CASSIAR CO-ORDINATES CR 4 TO RT 1 = 4993.315 METRES AS COMPARED TO OBSERVED DISTANCE 4993.530 METRES.

5. USING CASSIAR CONVERSION FROM MINE TO PLANT DATUM, THE ELEVATION OF RT 1 = 1920.484 METRES AS COMPARED TO AN OBSERVED ELEVATION OF 1920.32 METRES.
6. I.B. 601 – 606 CONSIST OF 1" DIAMETER REBAR WITH A NUMBERED PLASTIC TAG ATTACHED. ELEVATIONS ARE TO THE TOP OF THE BAR.
7. REVISION 1 SURVEY INDICATED BY DASHED LINES: -----------

FIG.7

REV NO	DATE	REVISION	DR.	CH.	APP.
1	14/09/87	TIE TO ADITS 1415 & 1563	W.C.	KMH	KMH

CASSIAR MINING CORPORATION

PLAN SHOWING HORIZONTAL AND VERTICAL CONTROL FOR PROPOSED ADIT 1170.

MATTHEWS & ASSOCIATES
PROFESSIONAL LAND SURVEYORS
NO. 5–5763 OAK STREET, VANCOUVER, B.C.

DESIGNED: KMH	SCALE: 1:10000 (METRIC)
DRAWN: KMH	DATE: AUGUST, 1987
APPROVED: KMH McClary	JOB NO. 1012 / 87
CLIENT DWG. NO.	MATTHEWS DWG. NO. 1012 / 87 – 1


GEOLOGICAL BRANCH
ASSESSMENT REPORT
16,776
6675 N
6650 N
6625 N
6600 N
6575 N
6550 N
7600 E
7625 E
7650 E
7675 E
7700 E
7725 E
7750 E
7775 E
7800 E
N
DDS 7600E
Blocky, moderately hard serp with many faults, slips + joints.
Strongly cleaved crumbly serp. with faults and Fractures
Blocky and Broken dark green serp. Minor Fibre.
Blocky with talcy joints + slips 2-4% fibre
Weakly to moderately sheared 2-3 m lensoidal blocks bounded by slips. In Fault zones blocks are strongly cleaved or gouge. Serpentinite caving on slips and by disintegration of soft blocks. Minor Fibre.
Blocky, hard weakly sheared serpentinite, dark green with minor fibre and occassional talcy gouge along slips + fractures.
Moderately sheared, crumbly serpentinite, small lensoidal blocks (fishscale) bounded by talcy gouge. Minor fibre.
Blocky dark green serpentinite very minor Fibre.
Highly sheared, soft Fishscale serpentinite, trace Fibre
Blocky dark green serpentinite
Moderately to highly sheared + broken serpentinite minor fibre.
Blocky moderately hard dark green serpentinite with occassional talcy shears
Moderately to highly sheared, soft to very soft serpentinite; occassional blocks of hard serpentinite surrounded by gouge.
Hard to moderately hard, blocky, dark green serpentinite occassional fibre veinlets
Bedding
Foliation
Lineation
Joint
Fault
Slickensides
Mineralized Vein
Asbestos Vein
Fold Axial Plane
Plunge
CASSIAR MINING CORPORATION McDAME EXPLORATION
14-7610 DRIFT EXTENSION
FIG. 9
CT
KCM
KCM
1500-250-14
1:250
05/08/87

CASSIAR MINING CORPORATION	McDAME EXPLORATION
PROPOSED UNDERGROUND DEVELOPMENT PHASE (III) REVISION 1 FIG. 3	DRN: CT CKD: KCM APR: KCM DRN Nº 1500-30001-3 SCALE: 1:1000 DATE: 21/07/87