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INTERIM FEASIBILITY STUDY

HIGHMONT MINING CORP. LTD.,

GNAWED MOUNTAIN MINERAL DEPOSITS

HIGHLAND VALLEY, B.C.

W. G. Hainsworth

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INTRODUCTION

At the request of Mr. R.W. Falkins, President of Highmont Mining Corp. Ltd., and the directors thereof, the writer has carried out a study and evaluation of all available data pertaining to this company's property situated on Gnawed Mountain in the Highland Valley area of the Kamloops Mining Division of British Columbia. The purpose of this report is to assess the potential of mineral deposits of copper, molybdenum and minor values in gold and silver indicated to be present by a drilling program subsequently followed by an underground bulk sampling program, and to recommend the type and the amount of additional work which is required, in the writer's opinion, to determine the advisability of putting the property into production.

The writer has been in constant touch with operations at this property since 1962 and has acted in an advisory capacity through many of the various programs carried out on the property.

It should be stated that several reports have been written concerning this property with the latest, and most authoritative, being the preliminary feasibility report of Chapman, Wood & Griswold Ltd. of July 6, 1967. The writer makes frequent references to this publication.

CONCLUSIONS

The mineralization as exposed in the underground workings to date has not, as yet, come up to the predicted grade. This is most notably true with the molybdenum value.

The adit drive cut an average of 0.20% copper and 0.024% molybdenum disulphide along a length of 586 feet (Williams' figures). With selective mining, this zone can produce a grade of 0.275% copper and 0.038% MoS2 on a waste to ore ratio of .72 to 1. In this area, the Chapman, Wood & Griswold report of July 1967 gives a grade of 0.298% copper and 0.076% MoS2 with a stripping ratio of .46 waste to 1 ore. On this basis, the copper values closely approximate one another, whereas the molybdenum values are almost halved to the predicted values. The stripping ratio has been increased by 50% but is still within an acceptable ratio. Reference should be made to Appendix I which breaks down the sampling of this main crosscut by the four assayers and arranges them according to a copper cut-off grade.

The P-20 drift west averaged out at 0.187% copper and 0.011% MoS2. This is a very weak average but geological evidence tends to show the drift was poorly located, lying in between mineralized lenses.

The first raise completed to within ten feet of surface - P-19 - averaged 0.257% copper and 0.016% MoS2. This raise was run up a percussion and a diamond drill hole which had been purposely located side-by-side in the surface drilling of 1966. The percussion hole - P-19 - over the raise section ran 0.393% copper and 0.086% MoS2. The diamond drill hole core - #66-9 - assayed 0.226% copper and 0.022% MoS2 over the same section. It should be stated that the drill hole wandered off and was contained in the raise less than half the distance. Nevertheless, it is obvious that there is more correlation between the raise rounds and the diamond drill hole than the percussion hole with either of the other sample methods.

The second raise - P-20 - was put up on the percussion hole of the same number 200' west of the previous raise. This raise encountered bad ground throughout with the result that it was abandoned at 115 feet, some 55 feet from surface break-through. The raise averaged 0.229% copper and 0.012% MoS2. This compares unfavorably with the percussion hole results of 0.56% copper and 0.116% MoS2. Diamond drill hole #66-15 drilled for metallurgical purposes was picked up in the raise for 91 feet. Across this length, the core assayed 0.26% copper and 0.033% MoS2. The raise rounds over this abbreviated distance averaged 0.21% copper and 0.012% MoS2. Diamond drill hole 66-1 also drilled at this site was not located in the raise.

It should be obvious that the percussion results in both raises are open to doubt. The diamond drill core results are closer to the raise averages, particularly with reference to copper values. The molybdenum values from the raises are considerably below the drill averages. This raises the question of whether the grinding in the underground sampling circuit is fine enough.

On the basis of the underground work correlation with surface work, it is evident that underground results will fall far shy of the surface results, particularly in the molybdenum values. This would almost eliminate reliance on the percussion drill results. However, Appendix II, comparing the side-by-side assay results of diamond drill holes and percussion holes shows that the overall copper average compares favorably. The molybdenum results in the percussions run 50% higher. These results extend to the 250 foot horizon only.

In comparing Appendix II with Appendix III, which evaluates the diamond drilling to the 500 foot level, a slight diminution in copper values is evident, whereas the molybdenum values remain the same.

In totalling the results of these three appendixes, the conclusions are:

1. That diamond drilling will yield values more closely approximating the underground results.
2. Overall copper values of the percussion holes will approximate that of diamond drilling, but percussion molybdenum values are untrustworthy.
3. Copper values fall off very slightly from the 250 foot to the 500 foot level.
4. With a slight increase in the stripping ratio, the copper grade as predicted can be maintained. The molybdenum grade could approximate 60% of the predicted value.

The grade of the East Zone as predicted by Chapman, Wood & Griswold involved some 45 percussion holes over an area of 3-1/2 million square feet. Eleven diamond drill holes represent this same area emphasizing the need for more drill holes. In the West Zone, only one diamond drill hole represents 11 percussion holes.

Referring to Appendix II, diamond drill holes #66-1, 3B, 4 - 10 inclusive represent side-by-side drilling with percussion holes in the East Zone. The diamond drill holes grade out at 0.231% copper and 0.034% MoS2. The corresponding percussion holes average 0.284% copper and 0.056% MoS2. Percentage-wise, the diamond drill holes represent 82% of the percussion copper grades and 60% of the percussion molybdenite grades. If these percentages are applied to the grades given the East Zone as mentioned in the Chapman, Wood & Griswold report, then the East Zone becomes:

Cu = 0.298% x 82% = 0.233%

MoS2 = 0.076% x 60% = 0.046%

The West Zone has insufficient diamond drilling to apply the same formula. An interesting point in the comparison of diamond drill hole 66-11 and P-134, coincident paired holes in this zone, is the relative agreement of molybdenite assays in this higher grade section. The diamond drill copper grades are 35% higher than those resulting from the percussion boring.

The depth representation is very small. The percussions, representing a tonnage of 48.187 million tons in the East Zone, only extend to the 250 foot horizon. The determining factor in depth here was the physical penetration of the percussion drills in 1966. Beyond 250 feet, progress was slow and unreliable. It is quite evident now in viewing Appendix III that the doubling or possibly trebling of the tonnage can be achieved by diamond drilling to the 500 or 1000 foot level.

The West Zone, although smaller in size than the East Zone could possibly be doubled from its present 9.618 million tons reserves.

The writer sees no difficulty in obtaining a tonnage in excess of 100 million tons.

One of the plus factors in the Highmont ore is the minor amount of oxide copper present. In raise P-19, the rounds average out at 0.062% oxide copper while P-20 raise shows 0.029% oxide copper. In the latter raise, two assayers ran the rounds for oxidation products. One averaged 0.013% oxide copper whereas the other averaged 0.046%.

A bonus contribution to any concentrate produced will be silver values. Silver has been shown to be present in minor amounts (0.10 - 0.15 oz/t) in many of the rounds sampled.

As mentioned in the chapter dealing with metallurgy, the observations of Britton Research in their Progress Report #2 and the Nippon metallurgical reports are definitely not encouraging as regards molybdenum concentration. It becomes evident that a recovery of 85% to 87% copper and from 45% to 54% molybdenum can be expected based on these preliminary bench tests. This is in contrast to the earlier tests run on NX core which, according to Britton Laboratories, showed a copper recovery of 87% minimum and a molybdenum recovery of 80% minimum. This very confliction in results on different types of material suggests further benchtests are necessary.

The underground diamond drilling results were such as to maintain the underground copper grade and to approximate the molybdenite values. It also emphasized the tabular relationships of the molybdenum mineralization.

It is the writer's opinion that the Highmont property as presently evaluated is some distance away economically from being a producer. It is bordering on very tight assay results. The molybdenum grade has been extremely disappointing in the underground phase. The copper, although far from being satisfactory, can be termed adequate, dependent on final molybdenum results.

To bring the property into the realm of economic possibilities, a copper grade exceeding 0.30% and a molybdenite grade exceeding 0.045% must be achieved. A concentrate with good recovery and grade must be resolved from these figures. Emphasis, therefore, must be placed on sampling - both underground and surface drilling - and on flotation tests. It is possible that a larger stripping ratio could bring the present grade in line with the recommended minimum. However, this would also increase the production costs with a resultant lowering of net profit.

It would be unwise at this point to fully evaluate the present status of the property as the initial recommendations have not been fully implemented. Therefore, a program involving the remainder of the recommended work plus additional work to decipher the percussion results would be the most logical course of action.

RECOMMENDATIONS

The Highmont property in its present state requires three essential and interwoven programs.

It is obviously necessary to re-sample by diamond drilling the main ore zones as outlined by the Chapman report. It is an equal necessity to know the correlation between these results and underground sampling. Lastly, tests must be conducted to prove up the possibilities of deriving an economical finished product.

The diamond drilling of the deposit should be done in two operations - a surface and an underground program. Both can be conducted simultaneous although there may be disruptions in the underground drilling due to underground mining operations.

Surface Diamond Drilling

It is recommended that the surface diamond drilling consist of 12,750 feet of BQ wire line drilling distributed in 17 holes. This is considered to be a minimum footage for intelligent assessment of the zones. The drilling is laid out in north-south sections across both zones at varying intervals. The south bearing drill direction was chosen because this is nearly at right angles to the suspected line of strike of the mineralization. At each drill set-up, a 700 foot south bearing hole should be put in at 45°. Following this and from the same set-up, a vertical hole of 100 foot depth should be put down. This short vertical will greatly assist interpretation of the geological and mineralogical results.

East Zone

Working from the east side of the deposit, the first drill section should have holes collared at P-29 and P-15. Stepping 400 feet west, another section should be collared at P-31 and P-17. Moving 400 feet to the west and now drilling on the adit section, holes should be located at P-33 and P-19. As both these holes have previous vertical holes located at these positions, the recommended verticals can be eliminated here. Another section should be located on holes P-35 and P-21. Single hole sections should be located at P-87 and 800 feet further west at a point 150 feet north of P 112. In addition, a single hole section should be spotted 100 feet north of P-48. An isolated vertical hole to the depth of 150 feet should be put down on the site of the proposed raise at P-25.

This requires 8750 feet of drilling in 12 holes.

West Zone

The most easterly section consists of a single location 200 feet north of P 217. P 133 would be the next section. Following this, a hole 100 feet north of P 226 and one 100 feet east of P 135 would constitute another section. The last section would be collared in at a point 200 feet north of P 136.

West zone drilling involves 4000 feet distributed in 5 holes.

This recommended surface drilling is not to be misconstrued as circumscribing pit outlines. Its primary purpose is to re-evaluate the zones and, if possible, to relate them to the past percussion drilling. It also seeks tonnage and grade figures to the 500 foot level.

Underground Diamond Drilling

All underground holes are horizontal and are put in for one of two purposes:

(1) As a guide for underground bulk sampling operations and to correlate drill results to this sampling.

(2) To assist surface drilling interpretation.

3000 feet of underground AQ wireline drilling in 6 holes is recommended.

The first hole is a guide-correlation hole. Located at P-40, it should be drilled south-east for 700' to pass under P-25 location. The recommended drive will follow this hole.

At the end of this drive and in the close proximity to P 17, two flat holes to assist surface drilling interpretation should be put out. One hole directed due north for 400 feet and the other due south for the same distance.

The present underground hole HU-1 should be extended by an additional 400 feet. This can best be achieved when the drive down its length is completed.

At the west end of the P-20 drift, underground hole HU-4 was previously drilled. This also should be extended an additional 500 feet. Here again this purpose is best served when the recommended underground excavation work on this holes is completed.

Diametrically located to the latter hole, is a recommended boring for 600 feet to the south at the present collar of HU-4. This hole would parallel the present HU-1 hole.

Underground Bulk Sampling

As it is absolutely essential to know whether underground sampling corroborates the diamond drill results, all the recommended underground operations are linked to diamond drill holes. In addition, further bulk material is provided from different locations in the underground for metallurgical tests. Three drives for a total distance of 1600 feet and two raises totalling 400 feet are highly recommended.

The first drive (630') is an extension of the main adit along underground drill hole HU-1.

The second drive (700') is a new opening to be driven from P-40 along the bearing of the flat drill hole put out underground from this point.

The third drive (270') will follow the course of underground hole HU-4.

Raises should be put up at P-5 and P-25 to test the validity of the inclined holes.

At P-5 the breakout distance is estimated at 250 feet, whereas P-25 requires 150 feet of rock excavation.

Metallurgical Tests

Further bench tests are a prime requirement in view of the latest developments in bench testing. The writer recommends that intensive study be developed in this direction.

Respectively submitted,

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Aug. 2, 1968

ESTIMATED COSTS

In view of the previous operation at the property, the cost per foot figure represents charges which are more realistic than estimated costs. Should these figures be adjusted one way or the other by the contractor(s) involved, the final cost figure will shift from the writers total. All figures are rounded off.

SURFACE DIAMOND DRILLING

12,750 feet @ $ 7.50/foot	=	95,600.00
Processing cost of above @ $ 2.50/foot	=	31,900.00

UNDERGROUND DRILLING

3,000 feet @ $ 5.00/foot	=	15,000.00
Processing cost of above @ $ 2.50/foot	=	7,500.00

UNDERGROUND EXCAVATING

1,600 feet of cross cutting @ $62.18/foot	=	99,500.00
Processing cost of above	=	99,500.00
400 feet of raising @ $62.18/foot	=	24,900.00
Processing cost of above	=	24,900.00

METALLURGICAL TESTS

Handling, analysis and interpretation	=	17,500.00

FINAL FEASIBILITY REPORT

Data collection and interpretation	=	40,000.00
Total expenditures		456,300.00
10% for contingencies		45,700.00
		$ 502,000.00

LOCATION & ACCESS

The property discussed in this report is on the westerly slopes of Gnawed Mountain, which is on the south side of Highland Valley, some 25 miles southeast of Ashcroft, B.C. Its approximate coordinates are 50° 25' North and 121° 00' West.

Access to the area is by the Ashcroft - Highland Valley - Merritt secondary improved gravel-pavement highway. It is some 25 miles southeast from Ashcroft or 45 miles northwest from Merritt to the property service road. This 5 mile mining road is in good travel condition.

Ashcroft is on the C.P.R. and C.N.R. mainlines, and on the Trans-Canada Highway #1, approximately 210 miles from Vancouver.

CLAIMS

Following is a list of claims and expiry dates as supplied by Highmont Mining Corp. Ltd.

Claims held by Highmont Mining Corp. Ltd., June, 1968

Name of Claim	Record Number	Expiry Date
AM 1 to 4 inc.	31188 to 31191 inc.	Feb. 18, 1969
AM 5 and 6 Fr.	31192 and 31193	Feb. 18, 1972
AM 7 to 11 inc.	31194 to 31198 inc.	Feb. 18, 1972
IDE 1	24994	Dec. 11, 1973
IDE 3	24996	Dec. 11, 1973
IDE 4 and 5	24997 and 24998	Dec. 11, 1973
IDE 6 to 8 inc.	24999 to 25001	Dec. 11, 1972
IDE 12 to 16 inc.	25710 to 25714	Mar. 19, 1972
IDE 17	25715	Mar. 5, 1972
IDE 18	25716	Mar. 19, 1972
NEW IDE 19	64034	May 8, 1973
NEW IDE 20	64036	May 8, 1973
ANN 3 FR, 4 Fr. and 7 Fr.	45132, 45133 and 45136	Feb. 21, 1973
ANN 18 Fr.	46153	May 20, 1972
ANN 20 Fr.	46155	May 20, 1973
NEW ANN 11 Fr.	64030	May 8, 1973
PHYLLIS Fr.	48513	Feb. 5, 1973

The expiry date of AM 1 to 4 inclusive should be noted and further assessment work applied against these important claims prior to the expiry date.

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4) Geological Map of the Highland Valley Area, B.C. Geology by J.M. Carr 1957-62 and R. Lee 1958. Preliminary Map, May 1966, Sheets 1 and 2.

5) Gnawed Mountain Option, Highland Valley, B.C., South Sheet, Geological Map. J. McA., Anaconda American Brass Ltd., Western Exploration Division, Nov. 1965.

6) Preliminary Feasibility Study, Gnawed Mountain Mineral Deposits, Highmont Mining Corp. Ltd. Chapman, Wood and Griswold Ltd., July 6, 1967.

7) An Investigation of Samples of Copper - Molybdenum Ore submitted by Highmont Mining Corp. Ltd. Progress Report No. 2. Britton Research Limited, April 18, 1968.

8) The Report on Metallurgical Tests of Highmont Copper - Molybdenum Ore, March 1968. Nippon Mining Co. Ltd.

9) The Report on Metallurgical Tests of Highmont Copper - Molybdenum Ore, Progress Report #2, April 1968 Nippon Mining Co. Ltd.

10) Review of Bulk Sampling Data (Letter Report), March 11, 1968 Chapman, Wood and Griswold Ltd.

GEOLOGY

GENERAL GEOLOGY

The Highland Valley is almost totally underlain by the Guichon batholith, an eastern segment of the Coast intrusive complex. This quartz-diorite formation has been dated by Duffel & McTaggart in their GSC Memoir 262 as between early Upper Triassic and early Middle Jurassic time, most probably during the Lower Jurassic.

Following the general trend of the invaded host rocks, the batholith has its long axis trending a little west of north. It extends from just north of Merritt almost to Ashcroft. Its eastern boundary is confined by Guichon Creek, whereas the volcanic-sedimentary formations lining the Thompson and Nicola River valleys limit its western extent.

The Guichon batholith has produced a complexity of differing phases. On the Highmont ground, we are concerned primarily with the Skeena Quartz Diorite and the Bethsaida granodiorite.

LOCAL GEOLOGY

One of the distressing aspects of the Highmont property, and this can be related to the whole Highland Valley, is the large extent of glacial debris. Almost 90% of the property is covered by glacial topography with bedrock being covered by up to 35-40 feet of overburden material. Only on the eastern claims, where the slopes reach to Gnawed Mountain peak, are there sufficient exposures to correlate the geology. Data from the central and western claims are assembled from drill holes. Little trench work was done in these areas.

In a broad sense, the Highmont property can be said to be underlain by the Skeena Quartz diorite with a wide band of Bethsaida granodiorite trending on a rough north-west lineation across the central claims. The Bethsaida locally forms an inner segment of quartz porphyry which culminates in a core of breccia material. The mineralized zones of the property lie along the Bethsaida and Skeena contact within the latter formation.

The Skeena quartz-diorite, which forms better than 80% of the exposed formation on the property, derives its name from its type occurrence in the nearby Skeena Silver Claims.

In all respects, this formation is similar to the Bethlehem quartz diorite. It is a light-grey, medium grained, fresh-appearing rock composed of hornblende, biotite, quartz, plagioclase and potassium feldspar. The mafic minerals are of less abundance in the Skeena than in the Bethsaida formation. An alteration phase that the Skeena undergoes is a mild kaolinization, which gives the rock surface a whitened effect. Chlorite alteration, again of minor intensity, is also distributed through the Skeena exposures.

In comparing the Skeena alteration on the Highmont property with that of the adjoining Lornex property, a great difference is noted. On the Lornex property, intense alteration accompanied by strong shearing and fracturing extends from the Bethsaida-Skeena contact outwards for some 1500 - 2000 feet with progressive weakening. On the Highmont, no such broad alteration zone has been noted.

The Bethsaida formation, appearing as an 800' wide dyke, cuts across the Skeena quartz diorite. The rock appears as a coarse-grained, occasionally porphyritic, dark colored formation with biotite taking preference to hornblende. The large euhedral crystals of quartz often reach a diameter of 1/4", giving the formation the porphyritic structure previously mentioned.

A phase of the Bethsaida, and intimately related to it on the Highmont property, is the quartz diorite porphyry. This rock is made up of numerous rounded quartz phenocrysts averaging 1/8" to 3/16" in diameter, set in a light colored, exceedingly fine grained matrix of feldspar-quartz material. Mafic minerals are notably light with biotite being the most common. Specular hematite often appears as radiating clusters in the quartz veins that are fairly numerous in this formation or as an occasional filling in some of the tight fractures.

Breccia formations are the least commonly exposed rocks on the property but a well defined, almost circular, zone has been outlined at one point on the Highmont-Minex boundary. This formation, which plays a most important part in the localization of ore on the Bethlehem property, is of slight importance in the Gnawed Mountain area mineral-wise.

The Breccia fragments, which vary up to as large a size as 3 inches, are set in a dark, very fine grained matrix. These angular fragments appear to be mostly of the Skeena quartz diorite variety. Mineralization, predominately specularite with some of the copper minerals, lies close to the boundaries of this breccia extending out into the surrounding rock.

In addition to the above rock types, several other formations have been encountered. These include feldspar porphyry and aplite dykes. Both of these formations, cut in diamond drilling, appeared as occasional dykes of small dimensions and unassociated with mineralization.

STRUCTURE

As mentioned earlier, the Highmont claims are underlain by the Skeena Quartz diorite. This formation is exposed in roughly 80% of the outcroppings. Intrusive into this formation and appearing as a narrow dyke of some 800' width are Bethsaida formations of granodiorite and quartz porphyry. Strike of the structure is N70 - 80W with an as yet undefined dip. The dyke is exposed on the Highmont claims for some 3000 feet before dipping under the extensive overburden of the central and western claims. Its point of surface origin is near Gnawed Mountain summit, where rather weak evidence indicates an easterly dip under the intruded Skeena. Associated with a breccia formation, called the Gnawed Mountain Breccia, the structure trends almost due west for some 2000 feet where it suddenly terminates. The extension of the dyke appears a few hundred feet north where it adopts its N70 - 80W strike into the Highmont property. The breccia, which had been an integral part of the dyke to this point, is absent in the offset extension. The terminating force is thought to be a fault, although there is little surface evidence for this theory. A point against this fault abutment possibility is the extension of several fingers of breccia beyond this point. In the absence of the breccia zone within the offset portion, thin and irregularily-shaped dykes of quartz porphyry appear, culminating some 2000' further to the west as the major rock material within the dykes. In many locations, this contact between the porphyry and granodiorite is remarkably well defined; in other exposures, a gradational zone is most evident. The porphyry zone is predominate through the dyke for some 1500 feet before horse-tailing into narrow dykes that gradually die. Located almost centrally within this porphyry zone is a roughly circular plug of breccia material. The plug, of 400' diameter, has, in most cases, strong contacts with the adjoining formations.

Little evidence is available of the westward progress of this dyke. It is evident that a major change in strike occurs between the last known surface exposure on the Highmont ground and the drill outlined contact on the Lornex ground. On the Lornex ground, some 4500 feet further to the north-west, the Skeena-Bethsaida contact has assumed a north-south attitude following a major fault zone.

Two sets of fracture systems are evident. The first and predominating system tends N 20° - 30° W with vertical to steep south westerly dips, the other set varying between N 50 - 70 E with an attitude between the vertical to 60° north. A third joint set, considerably weaker, dips 20° - 30° to the north. Little displacement has been noticed but in several instances the N-S set has offset the E-W system by as little as 1/2 inch. Mineralization tends to favour the stronger north-south set, although the east-west group is noticeably mineralized. The horizontal structures show little affinity for the metals. In the underground workings, the north-east trending set appeared stronger and was more closely associated with mineralization than the other systems.

Fracturing is strong throughout the property but in no way can be compared with the intense rock disruption on the adjoining Lornex ground. Localized heavy fracturing results in chloritized shear zones that tend to concentrate the mineralization. Strong sections running to 100% chlorite were encountered underground over short lengths. Mineralization, particularly copper, increased notably in these altered zones.

Sericite alteration where recognized is extremely weak.

No strong fault structures are exposed through geological mapping, underground operations, trenching nor implied from surface lineation. However, diamond drilling in hole #66-3 cut a 46 foot fault zone whose attitude has not been delineated.

MINERALIZATION

In keeping with the Highland Valley, the Highmont mineralization is chiefly copper with minor to moderate molybdenum metals. The copper mineral is primarily chalcopyrite with noticeable concentrations of bornite. Chalcocite, although evident in the area, has not been identified as yet in the Highmont operation. Molybdenite is the only molybdenum metal. Pyrite is weak to absent through the property.

The copper minerals are normally of a disseminated variety but tied in to the fracture system. Chalcopyrite will often line the slip planes of the fractures and will emanate from here to the surrounding rock. Bornite normally appears as a fine pin-point mineral, finely disseminated throughout the rock. The molybdenite is present as heavy blobs within a quartz structure or as lining for the quartz veinlets. There has been no evidence to date of disseminated molybdenite. Strong concentrations of bornite, often intimately associated with the molybdenum, frequently appear in the siliceous veins. These veins, which appear to have no set pattern, vary from 1/4" up to several inches in width, and are most erratic in their distribution.

The mineralization apparently occurs in all rock types on the property, although there is some rude correspondence to the contact of the Skeena and intrusive Bethsaida. The concentration of metal within the Skeena Quartz Diorite decreases with distance from this contact. The Bethsaida, although mineralized, carries weak values. At the Highmont, it is only the north side of this Bethsaida-Skeena contact which is mineralized to possible commercial values. It is understood that at the adjoining Lornex similar conditions prevail.

There are two large areas of mineral accumulation on the Highmont ground. Both these areas present possible commercial extractive zones.

The first zone labelled the east zone, lies elongated along the Skeena-Bethsaida contact for some 4000 feet. It extends back from the contact, to the north, for 1200 feet. At its highest surface elevation, 5766', it projects some 255 feet higher than its lowest levels. With all values calculated to only a vertical depth of 250' below bedrock, the zone presents a tonnage figure of 48.187 million tons grading .298% copper and .076% molybdenum disulphide.

Some 400 feet to the north west and almost an extension of the previous section is the west zone. It has equal dimensions of 1000 feet in both lateral directions. With a tonnage figure of 9.618 million it represents a grade of .150% copper and .153% molybdenum disulphide.

In total then the two zones carry 57.805 million tons of a grade .275% copper and .085% molybdenum disulphide to a depth of 250 feet.

PREVIOUS WORK

In April 1966, Highmont Mining Corp. Ltd. acquired the 34 claim block from Torwest Resources for the consideration of 1,000,000 shares of escrowed Highmont stock.

At that time, emphasis was laid on the possibility of economic mineralization existing in the vicinity of the claims adjoining the Kennco ground. Basis for this was the fair mineralization exposed in several pits on the mutual boundary. During the month of June 1966, a soil sampling program was carried out across the two suspect claims. Despite the 300 odd samples taken and run for copper and molybdenum, no strong pattern nor heavy concentration of either metal was found. Subsequent drilling in this area bore out the weak surface indication.

The months of July and August 1966 were taken up with the mechanics of organization of the Highmont company. During this period, Rio Algom carried out localized I.P. work with some 2,750 feet of percussion drilling on an option basis. When the option was not picked up, Highmont started out on its main program. The 1962 diamond drilling of Torwest's had indicated a mineralized area in the eastern portion of the claims. Following the old saying "Stick with ore into unknown ground," a grid pattern was laid out over the mineralized area. With drill centres at 200 feet, the grid covered a 1200' x 2200' section. Percussion drilling was chosen as the probing medium. In several early test holes, rotary drilling had proven expensive and slow. All holes were to be taken down to a 250 ft. depth using a 2-1/4" core bit.

Automatic sample splitters were installed at the return vent of the hole with a thirty-second cut being taken from each 10 ft. sample. The samples were dried, weighed and shipped to J.R. Williams & Son in Vancouver for assaying. Here, they were run for copper on the 10 foot sections, while composites of 50 feet were assayed for molybdenum sulphide.

The program started Sept. 8th, 1966 with one truck-mounted percussion machine. A second machine was added in late November. Each machine was normally capable of drilling off a 250' hole, moving 200 feet and setting up on a new hole each shift. Two 10-hour shifts were carried per machine.

At the completion of the grid drilling, an expansion drill program was swung into with holes being located on a 400' spacing and offset from each other. Lateral movement of the program was controlled by claim boundaries on the east and consistently weak results to the north. In all, an area roughly 7800' x 5500' was probed by means of 262 percussion holes, totalling 61,116 feet.

On November 15, 1966, McPhar Geophysics of Toronto began carrying out an I.P. survey over the western portion of the claim. Percussion drilling at this time was still confined to the eastern sector. 15.4 miles of line were run with the line spacing being 400 feet. Several anomalous conditions were exposed through the survey. Resultant percussion drilling on these anomalies outlined what is presently known as the West Zone.

Simultaneous with the initiation of the I.P. survey, diamond drilling was started. 16 holes for an aggregate total of 8,278 feet were directed to depth along longitudinal and cross sectional areas of the East Zone. The BQ-size machine checked results to a vertical depth of 500 feet only. In all, save one hole, complete core plus sludge was sent for analysis. 4 of these holes were of short length and were drilled for metallurgical test cores only.

To further aid geological evaluation, seven trenches covering some 8,160 lineal feet were bulldozed.

The program was completed April 18, 1967 and all data was turned over to Chapman, Wood & Griswold Ltd., Consulting Engineers, for evaluation. At the same time, bench tests on the amenability of the ore as demonstrated in the test cores was conducted by Britton Research Laboratories of Vancouver.

The July 6, 1967 report of the Consultants recommended underground work with bulk sampling to further evaluate the property. The company quickly instituted this recommendation.

UNDERGROUND DEVELOPMENTS & RESULTS

On September 18th, 1967, Rayrich Development Co. Ltd. squared the portal face. From that day to the close-off of operations on April 19th, 1968, a total of 1730 feet of underground excavation was completed. This consisted of 1161 feet of adit work, 287 feet of drifting and 282 feet of raise work. In addition, better than 5000 cubic feet of slashing was done in preparing diamond drill stations, muck car turnabouts, compressed air tanks storage spaces, and other necessary rockwork.

The underground was carried on specific figures with the opening being 7½ feet by 7½ feet on a grade of ½ of 1%. Track elevation at the portal is 5406.07 feet. Bad ground was occasionally encountered which required some 53 sets of timbering in all the lateral work. The raises were completely timbered with a manway running alongside a muck chute.

The mining was carried out by drilling with Holman jack leg machines, mucking with a track mucking machines on a 12 foot moveable track section and emptying into trackless ore cars, each car capable of carrying 1¼ - 1½ tons of material. The material from each car was emptied into one of 16 concrete bins specially built to handle the tonnage from an individual round. A number was then assigned to the bin contents.

The program as laid out was that of an adit drive from the vicinity of Percussion Hole #63 towards Percussion Hole #40, then due south to Percussion Hole #5, a linear distance of 1900 feet. Wing drifts at right angles to the drive were to be put out at Percussion Hole #26, going east for 500 feet and west for the same distance. This, then, gave a total lateral footage of 2900 feet. Three raises were to be put through to surface at the extremities of the three drives. This entailed 625 feet of rock work. In total, better than 3500 feet of underground excavation was recommended as the minimum acceptable footage for the basis of a realistic and acceptable engineering feasibility report. As noted, only half this minimum footage was carried out. The original program was carried through to Percussion Hole #19. At this point, a wing drift was put out westward to Percussion Hole #20. Raises were put up on both these percussion holes. A raise was started at Percussion Hole #33 but was discontinued after 2 vertical rounds. All changes from the original plan were made at the request of Nippon Mining Company, partners in this venture with Highmont Mining Corp. Ltd. It would appear that their thinking was dictated by the weak molybdenum results emanating from the adit drive.

The results of this underground work was to reveal a thin section of the property on which the mineralization tended towards steeply-dipping lenses. The grade and width of these lenses were unpredictable as was the intervening weaker sections. The tendency for heavier mineralization was dependent on alteration. Where heavy alteration in the form of chlorite and/or sericite was present, mineralization, particularly copper, increased noticeably. The molybdenum content was dependent on the fracture density.

From the 343 foot mark in the adit drive, test holing became routine. Walls were alternately test holed with 6 foot steel on each round. The sludges collected were assayed for copper and molybdenum.

BULK SAMPLING

Each round is identified and treated separately.

The average round consists of 20 - 25 tons of broken muck which is removed from the face by a scoop-crete buggy of 1-1/2 tons capacity and deposited in a clean concrete bay. There are 16 such bays.

Muck is removed from the bay and loaded into a hopper above the jawcrusher by means of a Hough Loader, 4 wheel drive, of 3 tons capacity. The jaw crusher is 18" x 24" and discharge is minus 1-1/2 inch.

Following the jaw is a set of rolls, 20" x 30", which reduces the product to 5/8". From the discharge of the rolls a conveyor belt carries the product to the sampling tower.

Here, a Denver Cutter type sampler makes the first quantity reduction, removing 1/120th of the stream and dropping the remainder into a bin from which it is drawn and taken by the Hough Loader, to a warehousing field. Here, each round is identified by number for treatment in future pilot plant work.

The material removed by the sampler, 1/120th or roughly 400 pounds, drops to a 12" gyratory crusher. Here,it is reduced to 5 mesh.

The 5 mesh product is then cut by a vezin sampler in a ratio of 1 - 20.

The larger portion, now approximately 380 pounds, is boxed and identified for bench testing.

The remaining 20 pounds is split 4 ways by means of a Jones riffle. One portion is retained for visual reference and to check for oxidation. The other 3 are available for assay determination.

The plant is completely open-circuit.

Following each round, the plant is shut down and bins, belts, and all units are swept clean.

The products are all completelv identified as to location in the orebody and consist of the following

22 tons of 5/8" on the warehouse field, for pilot plant work.

380 pounds of - 5 mesh, boxed and available for bench scale testing.

4 samples of 5 pounds each, available for assay, visual reference, and oxidization determination.

UNDERGROUND DIAMOND DRILLING

In March 1968, as the underground operation was gradually being phased out, an underground diamond drilling program on a limited basis was adopted by the Highmont management. The drill proposal was a collaboration between Chapman, Wood & Griswold Ltd. and the writer. Reference should be made to item 10 in the Bibliography re this direction.

The original recommendation allowed for 2500' to 3000' of AX drilling in 4 holes. Only two of these holes were completed for a total footage of 900 feet. At this point, financial support recommended the closing off of this program.

Hole #HU-1, a flat boring was collared in the face of the adit drive and on the same bearing. It could be stated to be an extension of the drive on a smaller volume. The 630 foot hole graded 0.265% copper and 0.013% MoS_2 according to one assayer and 0.271% copper and 0.020% MoS_2 by another assayer. Close agreement here on the coppers but some discrepancy between the molybdenite assays. This hole was originally laid out to cut the Bethsaida-Skeena contact. It did not reach its target. The thinking regarding the contact was that mineralization often tends to accumulate at contact planes. The Lornex ore structure lies close to the contact. The Highmont zone also parallels the contact, but would there be an increase in mineralization in the immediate vicinity of this contact? This question, unfortunately, was not answered with this drill hole.

The other hole, HU-4, was spotted at the east end of the P-20 east drift. A flat hole, it was drilled north with the intention to intersect the area of percussion hole P-27. At this horizon, the percussion hole averaged 0.42% copper and 0.261% MoS_2 over 30 feet. In the vicinity of this percussion hole, the diamond drill hole returned results of 0.25% and 0.28% copper according to the two assayers. The molybdenite values were 0.457% and 0.670% MoS_2 at the same locations. However, the molybdenite values were definitely local, conveying the impression that the molybdenum mineralization lies in narrow lenses. The overall hole average for its 270 foot length was 0.21% copper and 0.036% MoS_2 by one assayer and 0.172% copper and 0.045% MoS_2 by the other analyst.

These two holes strengthened the writer's thinking that the molybdenum values lie in narrow lenses with little dissemination between. There is presently insufficient data to make any prediction as to frequency or dimensions of these structures. Further underground excavation and drilling should assist this important facet of operations.

Individual core results from each of the two holes are shown in Appendix VI and VII.

METALLURGY

In May 1967, Britton Research Limited of Vancouver were commissioned to carry out preliminary bench scale flotation tests on large size (NX) diamond drill core, with the view of determining the amenability of the mineralization towards concentration.

The results of these early tests were very encouraging as noted in Chapman, Wood & Griswold's report of July 6, 1967.

In February 1968, Nippon Mining Company requested portions of crushed material from specific locations in the underground workings on which their Tokyo laboratory could run flotation tests. At the same time, Highmont personnel forwarded similar weights from the same underground locations to Britton for similar tests. The results were obtained from both test labs in April 1968.

Both are produced in part or whole as Appendixes to this report. The Nippon Tokyo report is Appendix IV and Britton's report as Appendix V.

In contrast to the early tests, these were a distinct disappointment. Both laboratories agreed closely with one another. Britton Labs reported copper recovery, although down somewhat from the earlier tests, to be in the 85% to 87% range with a concentrate grade in the vicinity of 22% to 28% copper, depending on mill heads. Tokyo reported better recovery - in the 95% range - with a concentrate averaging 25% copper.

In the molybdenum separation, it was apparent that more regrinds would be necessary to improve the recovery of molybdenum and additional methods would be required to eliminate the copper content. Both metallurgists recorded molybdenum separations that carried more than the allowable 0.30% copper. Each laboratory agreed on a molybdenum recovery in the vicinity of 45% to 54%. This is an extremely low recovery, particularly in view of the disappointing underground molybdenum values.

With the confliction between these bulk tests and the earlier diamond drill tests, further tests are a prime requisite. These future tests should, again, be carried on with underground bulk material with a view to a complete study of method and cost to obtain a satisfactory molybdenum separation. The selection of material for these tests should be the prerogative of the metallurgist in consultation with local management.

APPENDIX I

GRADE COMPARISON MAIN XC (101 Samples)

	WILLIAMS	RED MOUNTAIN	BETHLEHEM	COAST ELDRIDGE
Cut-Off	.300% Cu Equiv.	.300% Cu Equiv.	.300% Cu Equiv.	.300% Cu Equiv.
# Samples	33	39	31	34
Grade	.487%	.477%	.526%	.488%
100% Rec	$3.70 @ 38¢ copper	$3.63 @ 38¢ copper	$4.00 @ 38¢ copper	$3.71 @ 38¢ copper
80% Rec	$2.96	$2.90	$3.20	$2.97
Stripping-Ratio	2 waste to 1 ore	1.56 waste to 1 ore	2.23 waste to 1 ore	1.91 waste to 1 ore
Cut-Off	.250% Cu Equiv	.250% Cu Equiv.	.250% Cu Equiv.	.250% Cu.Equiv.
# Samples	39	53	42	42
Grade	.447%	.421%	.460%	.447%
100% Rec	$3.40 @ 38¢ copper	$3.20 @ 38¢ copper	$3.50 @ 38¢ copper	$3.40 @ 38¢ copper
80% Rec	$2.72	$2.56	$2.80	$2.72
Stripping-Ratio	1.56 waste to 1 ore	.923 to 1 ore	1.38 to 1 ore	1.38 to 1 ore
Cut-Off	.200% Cu Equiv.	.200% Cu Equiv.	.200% Cu Equiv.	.200% Cu Equiv.
# Samples	49	65	52	60
Grade	.404%	.384%	.413%	.379%
100% Rec	$3.07 @ 38¢ copper	$2.92 @ 38¢ copper	$3.14 @ 38¢ copper	$2.88 @ 38¢ copper
80% Rec	$2.46	$2.34	$2.51	$2.30
Stripping-Ratio	1 waste to 1 ore	.563 to 1 ore	961 to 1 ore	.695 to 1 ore
Cut-Off	.175% Cu Equiv.	.175% Cu Equiv.	.175% Cu Equiv.	.175% Cu Equiv.
# Samples	62	70	65	65
Grade	.357	.370	.368	.364
100% Rec	$2.71 @ 38¢ copper	$2.81 @ 38¢ copper	$2.80 @ 38¢ copper	$2.77 @ 38¢ copper
80% Rec	$2.17	$2.25	$2.24	$2.22
Stripping-Ratio	.639 waste to 1 ore	.449 to 1 ore	.563 to 1 ore	.563 to 1 ore

NOTE: MoS2 = 2½ Cu.

AVERAGE	CUT-OFF	100% Rec.	80% Rec.
	.300%	$ 3.76	$ 3.01
	.250%	$ 3.38	$ 2.70
	.200%	[illegible]	

APPENDIX II

		CORE			SLUDGE						
D.D. HOLE	ANGLE	Cu.	MoS2	DEPTH	Cu.	MoS2	DEPTH	PERC. HOLE	DEPTH	CU.	MoS2
66-1	-90°	.209	.052	245'	.277	.083	240'	I-20	241'	.437	.105
66-3B	-45°	.208	.028	275'	.360	.072	270'	I-242	284	.170	.035
66-4	-45°	.16[illegible]	.02[illegible]	3[illegible]2'	.354	.041	370'	I-243 A (I-243	392' 380'	.242 .242	.051 .056)
66-5	-45°	.311	[illegible]	[illegible]	.432	-	340'	I-24[illegible] A (I-24[illegible]	[illegible] [illegible]	.309 .192	[illegible] .062)
66-6	-45°	.15[illegible]	[illegible]	192'	.136	[illegible]	180'	I-210	[illegible]	.121	[illegible]
66-7	-90°	.136	[illegible]	2[illegible]'	.1[illegible]	[illegible]	240'	I-[illegible]	2[illegible]'	.245	[illegible]
66-8	-90°	.270	[illegible]	250'	.322	.09[illegible]	240'	I-[illegible]3	240'	.262	.044
66-9	-90°	.220	.021	240'	.323	.086	230'	I-19	230'	.351	.083
66-10	-90°	.374	.010	232'	.454	.041	232'	F-5	234'	.421	.036
66-11	-90°	.287	.140	238'	.323	.214	238'	P-134	235'	.214	.154
AVERAGE		.236	.043		.368	.077				.278	.064

APPENDIX III

DIAMOND DRILL HOLES TO 500' DEPTH

HOLE	ANGLE	DEPTH	CORE		SLUDGE	
66-1	90°	[illegible] / [illegible]	[illegible]	.051	.224	[illegible]
66-2	45°	745' / [illegible]	.212	.042	.305	.045
66-3A	45°	364' / 345	[illegible]	.028	.172	.085
66-3B	45°	735' / 730'	.186	.024	.334	.083
66-4	45°	672' / 660'	.145	.032	.350	.052
66-5	45°	720' / [illegible]	.210	.046	.446	.064
66-6	45°	800' / [illegible]	.209	.043	.235	.058
66-7	90°	556' / 552'	.126	.041	.191	.048
66-8	90°	[illegible] / [illegible]	.217	.019	.432	.110
66-9	90°	[illegible] / [illegible]	[illegible]	.035	.220	.051
66-10	90°	[illegible] / [illegible]	[illegible]	.023	.454	.068
66-11	90°	[illegible] / [illegible]	.213	.103	.279	.142
			[illegible]	.041	.309	.071

APPENDIX IV

THE REPORT ON METALLURGICAL TESTS

of HIGHMONT COPPER - MOLYBDENUM ORE,

PROGRESS REPORT NO.1

Date: March, 1968

Investigation by: T. Itoh, S. Kobayashi

Central Laboratories,

Nippon Mining Co. Ltd.

Tokyo, Japan

INTRODUCTION

The results of concentration tests on composite samples of drill core from D.D. holes 66-12 to 66-15 inclusive were given in Progress Report No. 1, dated November 1, 1967.

The present report gives the results of tests on three samples (A, B and C), which consisted of crushed ore from the current bulk sampling programme and which were received on December 18, 1967. Preliminary tests to determine the optimum grind and rougher flotation conditions were carried out on a composite of equal weights of the three samples.

Instructions for the tests to be carried out were given by Mr. W.G. Hainsworth, P. Eng., Consulting Geologist.

SUMMARY

1. Assays and specific gravities of the individual and composite samples were as follows:

Sample No.	A	B	C	Composite
Our No.	67A	67B	67C	67D
Assays:				
Total Cu	0.22%	0.36%	0.48%	0.36%
Oxide Cu	0.004%	0.004%	0.004%	0.004%
Total Mo	0.008%	0.016%	0.051%	0.025%
Oxide Mo	0.001%	0.001%	0.001%	0.0003%
Total S	0.25%	0.35%	0.51%	0.36%
Specific Gravity	2.67	2.66	2.66	2.65
Cubic feet per ton	12.0	12.0	12.0	12.1

2. The grinding work indices (Bond) were as follows:

Sample No:	67A	67B	67C	67D	67D	67D
Grind (% -200 mesh):	56	56	55	46	56	66
Work Index (KWH/ton):	18.5	14.8	13.7	15.2	15.8	17.6

SUMMARY (cont.)

3. Copper and molybdenum recoveries up to 93% and 84% respectively were obtained in bulk rougher flotation tests on the composite sample.

4. On the individual samples, after grinding to 55-56% minus 200 mesh, the results obtained in bulk rougher flotation were as follows:

Sample No:	67A	67B	67C
Cu recovery %	92.0	87.0	88.6
Cu assay %	5.89	4.75	7.02
Mo recovery %	78.4	72.2	72.3
Mo assay %	0.21	0.18	0.62

5. After cleaning the bulk concentrates 3 times, results were as follows:

Sample No:	67A	67B	67C
Cu recovery %	79.1	76.4	75.0
Cu assay %	28.37	22.82	23.24
Mo recovery %	59.3	53.1	53.8
Mo assay %	0.88	0.74	1.81
Cu lost in cleaning %	12.9	10.6	13.6
Mo lost in cleaning %	19.1	19.1	18.5

6. In the case of sample 67C, cleaning of the bulk concentrate was followed by 6-stage separation of the copper and molybdenum. The final molybdenum concentrate assayed 55.5[illegible]% Mo, 0.75% Cu, 0.03% Pb, 0.35% CaO and 2.37% SiO_2. Apart from copper, the assays were satisfactory.

7. With samples 67A and 67B, the molybdenum assays were too low to permit conducting a complete separation of the copper and molybdenum in the laboratory.

SUMMARY (cont.)

8. The copper concentrates (copper-molybdenum separation tailings) obtained from all three samples had satisfactory copper assays (23-29% Cu).

9. The following results can be expected when treating similar ore in a full-scale mill:

Ore similar to sample	A	B	C	Composite A + B + C (equal weights)
Copper concentrate:				
Weight: % of ore	0.70	1.37	1.72	1.26
Assays: % Cu	28	22	24	24
% Mo	0.21	0.13	0.21	0.18
Cu recovery: %	89	84	86	86
Molybdenum concentrate:				
Weight: % of ore	0.008	0.016	0.053	0.026
Assays: % Mo (min.)	54	54	54	54
% Cu (max.)	0.8	0.8	0.8	0.8
% Pb (max.)	0.04	0.04	0.04	0.04
% CaO (max.)	0.4	0.4	0.4	0.4
Mo recovery: %	52	52	56	55

10. The copper content of the molybdenum concentrates could be reduced by extra cleaning steps; if necessary, the concentrate could be leached with cyanide to remove copper. It is also expected that the lead and lime contents could be reduced, if desired, by leaching the concentrates with hydrochloric acid.

Respectfully submitted,
BRITTON RESEARCH LIMITED

John W. Britton, P. Eng.

John W. Britton, P. Eng.,
Consulting Metallurgist

APPENDIX IV

THE REPORT ON METALLURGICAL TEST
of HIGHMONT COPPER - MOLYBDENUM ORE,
PROGRESS REPORT NO. 2

Date: April , 1968

Investigation by: T. Itoh, S. Kobayashi
Central Laboratories
Nippon Mining Co. Ltd.
Tokyo, Japan

Name	Weight	Assay (%) Cu	Assay (%) Mo	Recovery (%) Cu	Recovery (%) Mo
Mo-Cl-3t	13.9	4.09	1.20	0.28	0.96
Mo-Cl-2t	12.3	[illegible]	[illegible]	1.67	1.76
Mo-Cl-1t	66.3	[illegible]	[illegible]	9.93	0.80
Cu-C	725.5	[illegible]	[illegible]	83.97	2.09
B-Cl-4t	76.0	0.36	0.22	0.32	1.01
B-Cl-3t	77.9	2.62	[illegible]	1.00	2.25
B-Cl-2t	171.0	[illegible]	[illegible]	1.50	4.34
B-Cl-1t	359.0	[illegible]	0.30	1.00	6.22
Total (B.R.C.)	1,558.3	13.036	1.112	100.00	100.00

Calculated assays and recoveries of Cu-concentrates and Mo-concentrates are as follows:

Cu-concentrates (Cu-C, Mo-Cl-1t and Mo-Cl-2t)

Cu	24.15%	95.62% recovery
Mo	0.100%	

Mo-concentrate (Mo-1C and Mo-2C)

Mo	47.10%	72.27% recovery
Cu	0.464%	

Summary

Based on these tests it was estimated that the copper concentrate would assay about 25% Cu with recovery of 85% to 87%, and the molybdenum concentrate would be required more regrindings and cleanings in order to get at least 55% Mo with less Cu assay than 0.3%.

The recovery of Molybdenum concentrate is estimated to be between 45% and 54%.

1st Mo-Cleaning

conc. — Na_2S 3.2 kg/t — tail (MO-Cl-1t)

2nd Mo-Cleaning

conc — tail (Mo-Cl-2t)

Conditioning 5 min.

3rd Mo-Cleaning

conc. — NaCN 640 g/t — tail (Mo-Cl-3t)

4th Mo-Cleaning

conc. — tail (Mo-Cl-4t)

NaCN 970 g/t
Na_2S 1.9 kg/t

5th Mo-Cleaning

conc. — tail (Mo-Cl-5t)

Regrinding

NaCN 970 g/t
Water Glass 200 g/t

6th Mo-Cleaning

Mo Conc. (Mo-1C) — Mo Conc (Mo-2C) — tail (Mo-Cl-6t)

Results of Cu – Mo seperation tests from Bulk Rougher Concentrates are given by following table.

Name	Weight	Assay (%)		Recovery (%)	
		Cu	Mo	Cu	Mo
Mo-1C	14.7	0.29	51.76	0.02	43.89
Mo-2C	11.9	0.63	41.34	0.04	28.89
Mo-Cl-6t	8.4	1.45	9.24	0.06	4.48
Mo-Cl-5t	10.4	0.93	1.55	0.05	0.56
Mo-Cl-4t	11.0	1.95	5.14	0.10	3.26

continued

Cu-Mo Seperation Test on Sample C

Flow sheet of Cu-Mo Seperation test on Sample C is as follows:

Feed
50% -200 mesh
Kerosene 50 g/t
Potash Amyle Xan. 50 g/t
DFW [illegible] [illegible] g/t
Pine Oil 10 g/t

Bulk Flotation 20 min.

Bulk Rougher Concentrate — tail

Potash Amyle Xan. 64 g/t

Bulk 1st Cleaning
conc. — tail (B-Cl-1t)

Bulk 2nd Cleaning
conc. — tail (B-Cl-2t)

Bulk 3rd Cleaning
conc. — tail (B-Cl-3t)

Bulk 4th Cleaning
conc. — tail (B-Cl-4t)

Active Carbon 190 g/t
$Ca(OH)_2$ 640 g/t

Conditioning

Filter and Washing

Na_2S 0.4 kg/t
15 min. conditioning

Cu-Mo Seperation Flotation 30 min.
conc. — tail (Cu-C)

Na_2S 3.2 kg/t

Conditioning 15 min.

Results of Cu-Mo Seperation test from Bulk Rougher Concentrates are given by following table:

Name	Weight (g)	Assay (%) Cu	Assay (%) Mo	Recovery (%) Cu	Recovery (%) Mo
Mo-1C	9.7	0.56	45.93	0.02	48.68
Mo-2C	1.9	0.29	45.75	0.003	9.50
Mo-cl-4t	1.3	1.06	19.13	0.007	2.72
Mo-cl-3t	6.4	3.33	16.92	0.09	11.82
Mo-cl-2t	9.8	9.09	3.39	0.38	3.63
Mo-cl-1t	23.1	29.21	1.55	2.87	3.91
Cu-c	855.0	25.37	0.10	92.32	9.34
B-cl-4t	104.5	5.05	0.26	2.25	2.97
B-cl-3t	186.0	0.74	0.06	0.59	1.22
B-cl-2t	195.0	0.90	0.11	0.75	2.34
B-cl-1t	393.0	0.43	0.09	0.72	3.87
Total (B.R.C.)	1785.7	13.157	0.513	100.00	100.00

Calculated assays and recoveries of Cu-concentrates and Mo-concentrates are as follows:

Cu-Concentrate (Cu-C and Mo-cl-1t)

Cu	25.47%	95.19% recovery
Mo	0.138%	—

Mo-Concentrate (Mo-1C and Mo-2C)

Mo	45.90%	58.17% recovery
Cu	0.516%	—

Bulk 4th Cleaning

conc.

tail (B-Cl-4t)

Active Carbon 163 g/t
$Ca(OH)_2$ 560 g/t

Conditioning

Filter and Washing

Na_2S 5.6 kg/t

Conditioning 20 min.

Cu-Mo Seperation Flotation 30 min.

conc.

tail (Cu-CO

Na_2S 1.7 kg/t
15 min. conditioning

1st Mo Cleaning

conc.

tail (Mo-Cl-1t)

Regrinding 5 min.

Na_2S 1.1 kg/t
NaCN 560 g/t

2nd Mo Cleaning

conc.

tail (Mo-Cl-2t)

Na_2S 1.1 kg/t
NaCN 560 g/t

3rd Mo Cleaning

conc.

tail (Mo-Cl-3t)

Regrinding

NaCN 560 g/t
Water Glass 100 g/t

4th Mo Cleaning

Mo-Conc.
(Mo-1C)

Mo-Conc.
(Mo-2C)

Tail
(Mo-cl-4t)

(2) 100 g/t collectors

Name	Weight (g)	Assay (%) Cu	Assay (%) Mo	Recovery (%) Cu	Recovery (%) Mo
1C	21.3	15.[illegible]	1.31	79.4	63.9
2C	8.2	4.45	0.43	3.6	9.0
3C	8.6	1.35	0.35	2.5	6.9
4C	8.4	[illegible]	0.15	2.2	2.9
tail	945.0	0.033	0.008	7.3	17.3
feed	991.5	0.430	0.0440	100.0	100.0

Calculated assays and recoveries of bulk rougher concentrates are 8.505% assay, 92.7% recovery in Cu, and 0.749% assay, 82.7% recovery in Mo.

Cu - Mo Seperation Test on Sample B

Flow sheet of Cu-Mo Seperation test on Sample B is as follows:

Feed
- 50% -200 mesh
- Kerosene 50 g/t
- Potash Amyle Xan. 50 g/t
- DF #250 20 g/t
- Pine Oil 10 g/t

Bulk Flotation — 20 min.

Bulk Rougher Concentrate — tail

Potash Amyle Xan. 56 g/t

Bulk 1st Cleaning

conc. — tail (B-Cl-1t)

Bulk 2nd Cleaning

conc. — tail (B-Cl-2t)

Regrinding — 10 min.

Bulk 3rd Cleaning

conc. — tail (B-Cl-3t)

Calculated assays and recoveries of bulk rougher concentrates are 10.908% assay, 92.3% recovery in Cu, and 0.752% assay, 75.8% recovery in Mo.

Cu – Mo Bulk Flotation Tests by changing amounts of Collectors

Flow sheet of these flotation tests is shown as follows:

Feed

1.0 Kg (Sample C)
H_2O 1.0 l
$Ca(OH)_2$ 1.0 Kg/t
DF #250 20 g/t

Grinding

(50% –200 mesh)
H_2O 2.3 l
Kerosene 50 g/t and 100 g/t
Potash Amyle Xan. 50 g/t and 100 g/t
Pine Oil 10 g/t
DF #250 10 g/t

Flotation

1C 2C 3C 4C tail

(1) 50 g/t collectors

Name	Weight (g)	Assay (%) Cu	Assay (%) Mo	Recovery (%) Cu	Recovery (%) Mo
1C	15.9	17.43	1.25	65.2	46.8
2C	8.1	10.20	1.20	19.4	22.9
3C	5.7	3.78	0.63	5.1	8.5
4C	5.6	1.88	0.29	2.5	3.8
tail	956.0	0.035	0.008	7.8	18.0
Feed	991.0	0.430	0.0423	100.0	100.0

Calculated assays and recoveries of bulk rougher concentrates are 11.123% assay, 92.2% recovery in Cu, and 0.986% assay, 82.0% recovery in Mo.

The results are given in following tables:

(1) Sample A, 50% -200 mesh.

Name	Weight (g)	Assay (%) Cu	Assay (%) Mo	Recovery (%) Cu	Recovery (%) Mo
1C	9.5	13.49	0.24	63.3	30.0
2C	3.9	6.54	0.18	13.6	9.2
3C	3.7	4.01	0.13	7.9	6.3
4C	3.0	1.82	0.03	2.9	3.2
tail	974.0	0.014	0.004	7.3	51.3
feed	994.1	0.139	0.0076	100.0	100.0

Calculated assays and recoveries of bulk rougher concentrates are 8.655% assay, 92.7% recovery in Cu, and 0.184% assay, 48.7% recovery in Mo.

(2) Sample B, 50% -200 mesh

Name	Weight (g)	Assay (%) Cu	Assay (%) Mo.	Recovery (%) Cu	Recovery (%) Mo
1C	15.0	14.14	0.32	67.3	40.9
2C	5.3	7.77	0.28	13.1	12.6
3C	4.3	5.12	0.24	7.0	8.8
4C	3.8	3.67	0.15	3.2	4.9
tail	963.0	0.031	0.004	9.4	32.8
feed	991.4	0.318	0.0118	100.0	100.0

Calculated assays and recoveries of bulk rougher concentrates are 10.051% assay, 90.6% recovery in Cu, and 0.278% assay, 67.2% recovery in Mo.

(3) Sample C, 50% -200 mesh

Name	Weight (g)	Assay (%) Cu	Assay (%) Mo	Recovery (%) Cu	Recovery (%) Mo
1C	16.3	14.29	0.46	59.4	26.1
2C	7.7	9.64	0.70	19.0	18.8
3C	6.2	5.68	0.66	9.0	14.3
4C	3.7	3.64	0.58	3.5	7.5
tail	956.0	0.037	0.010	9.1	33.3
feed	989.9	0.394	0.0290	100.0	100.0

WORK INDEX OF ORE

Sample No.	Work Index	Grinding (min.)	-200 mesh (%)
A	21.4	20	40.0
"	(F_{80} = 790 μ)	25	49.0
"	(P_{80} = 117)	30	56.0
		35	64.0
B	17.9	15	37.8
"	(F_{80} = 750)	20	44.7
"	(P_{80} = 90.5)	25	54.0
"		30	63.5
C	15.0	15	40.5
"	(F_{80} = 750)	20	46.5
"	(P_{80} = 115)	25	59.0
"		30	66.5

Cu - Mo BULK FLOTATION TESTS

Flow sheet of bulk flotation tests is as follows:

Feed

1.0 Kg (Sample A, B, and C)
H_2O 1.0 l p.d. 50%
Ca $(OH)_2$ 1.0 Kg/t
Pine oil 20 g/t

Grinding

(50% -200 mesh)
H_2O 2.3 l p.d. 30.3%
tpp PH 11.0 - 11.2
Isopropyl Xan. 50 g/t
Areo-Float #25 10 g/t
Kerosene 50 g/t
Pine Oil 10 g/t
Dow-Froth #250 10 g/t

Conditioning 5 min.

Bulk-Flotation 5 min. x 4

1C 2C 3C 4C tail

INTRODUCTION

This report describes the metallurgical tests and assays carried out on samples of Copper-Molybdenum ore by bulk materials.

ASSAYS OF HEAD SAMPLES

The direct assays of the composite head samples used for the various metallurgical tests were as follows:

Composite Sample No.	A		B		C	
	Cu %	Mo %	Cu %	Mo%	Cu%	Mo%
based on Nippon	0.192	0.008	0.336	0.019	0.414	0.052
" " Coast Eldridge	0.21	0.005	0.33	0.010	0.44	0.040
" " Williams	0.15	Tr	0.32	0.021	0.47	0.006

SPECTRAL ANALYSIS OF HEAD SAMPLES

K	Sr	Ca	Co	Ag	Ti	Na	Cu
5	3	>5	2	2	>5	>5	>5
Mo	V	Al	Si	Fe	Ga	Pb	Mn
>5	4	>5	>5	>5	3	2	3
Mg	P	Ba	Sn				
>5	2	1	1				

0 : nil
1 : very poor
2 : poor
3 : clear
4 : rich
5 : very rich

APPENDIX V

AN INVESTIGATION OF

SAMPLES OF COPPER-MOLYBDENUM ORE

submitted by

HIGHMONT MINING CORPORATION LTD.

Progress Report No. 2

(3 Pages reproduced of 33)

WSG

Date: April 18, 1968

Project No.: B143

Investigation by: John W. Britton, B.Sc., A.R.S.M., P.Eng.,
Consulting Metallurgist,
Britton Research Limited,
1612 West 3rd Avenue,
Vancouver 9, B.C.

RESULTS OF BULK ROUGHER FLOTATION and Cu-Mo SEPERATION FLOTATIONS

These results are given by following table:

Name	Weight (g)	Assay (%)		Recovery (%)		
		Cu	Mo	Distribution	Cu	Mo
Mo-conc	22.0	0.56	49.15	0.04	0.06	49.48
Mo-m	2.5	2.89	7.49	0.01	0.04	0.86
Mo-cl-3 tail	8.0	1.78	32.24	0.02	0.07	11.80
Mo-cl-2 tail	10.0	6.68	2.99	0.02	0.32	1.37
Mo-cl-1 tail	27.0	20.14	1.81	0.05	2.62	2.24
Cu-conc	700.00	24.65	0.12	1.40	83.17	3.84
B-cl-4 tail	207.0	0.61	0.10	0.42	0.61	0.95
B-cl-3 tail	193.0	1.15	0.22	0.39	1.07	1.94
B-cl-2 tail	624.0	0.45	0.11	1.26	1.36	3.14
B-cl-1 tail	3357.0	0.13	0.05	6.76	2.10	7.68
B.R. Tail	44515.0	0.040	0.0082	89.63	8.58	16.70
Feed	49668.5	0.418	0.0495	100.00	100.00	100.00

Summary

(1) Calculated assays and recoveries of Bulk Rougher Concentrate are as follows:

Name	Weight(g)	Assay(%)		Recovery (%)		
		Cu	Mo	Dist.	Cu	Mo
B.R. Conc.	5,150.5	3.682	0.353	10.37	91.42	83.30
B.R. Tail	44,515.0	0.040	0.0082	89.63	8.58	16.70
Feed (calc.)	49,668.5	0.418	0.0495	100.00	100.00	100.00

Direct Assay of Feed

	Cu	Mo
Nippon	0.414	0.052
Coast Eldridge	0.44	0.040

(2) Calculated assays and recoveries of final Cu-Concentrates and Mo-Concentrates are as follows:

Cu-Concentrates (Cu-Conc. and Mo-Cl-1 tail)

Cu 24.48% 85.79% recovery

Mo 0.183%

Mo-Concentrate (Mo-Conc)

Mo 49.15% 49.48% recovery

Cu 0.56%

Bulk 2nd Cleaning

conc. — tail (B-Cl-2 tail)

Bulk 3rd Cleaning

cono. — tail (B-Cl-3 tail)

Grinding 20 min.

Bulk 4th Cleaning

conc. — tail (B-Cl-4 tail)

Conditioning 10 min.

Filter and Washing

Na_2S 200 g/t
20 min.
PH 13.0

Cu-Mo Seperation Flotation 10 min.

conc. — tail (Cu-Conc.)

Na_2S 60 g/t

1st Mo Cleaning

conc. — tail (Mo-Cl-1 tail)

Grinding 3 min.

Na_2S 40 g/t
NaCN 20 g/t

2nd Mo Cleaning

conc. — tail (Mo-Cl-2 tail)

Grinding 3 min.

4th Mo Cleaning

Mo conc. (Mo Conc.) — tail (Mo-m)

INTRODUCTION

This report describes the results of metallurgical re-test and assays carried out on Sample C of Copper - Molybdenum ore by bulk materials.

FLOW SHEET of RE-TEST ON SAMPLE C

Feed

Lime	500 g/t
Pine Oil	20 g/t
Kerosene	50 g/t

Grinding 21.5 min. (50% -200 mesh) PH 9.8

Iso Propyl Xan.	50 g/t
Potash Amyle Xan.	50 g/t
Aero-Float #25	20 g/t
Kerosene	50 g/t
Pine Oil #10	10 g/t
Dow-Froth #250	50 g/t

Bulk Flotation 15 min.

conc.

tail

PH 9.5
Na_2S 100 g/t
K.A. Xan. 50 g/t

Bulk Flotation 10 min.

conc.

tail
(B.R. Tail)

Bulk Rougher Conc.

Water glass	11 g/t
Keresene	80 g/t
Potash Amyle Xan.	6 g/t

Bulk 1st Cleaning

conc.

tail (B-Cl-1 tail)

APPENDIX V1

DIAMOND DRILLING
HU-1

FOOTAGE	WILLIAMS Cu.	WILLIAMS MoS2	COAST ELDRIDGE Cu.	COAST ELDRIDGE MoS2	COAST ELDRIDGE Ag.	SLUDGE Cu.	SLUDGE MoS2
0 - 10	.15	TR	.15	.004		.10	.005
20	.25	TR	.20	.012		.45	.026
30	.50	TR	.42	.006		.75	.003
40	.15	TR	.33	.005		.37	.011
50	.15	.010	.18	.009		.25	.030
60	.10	TR	.07	.003		.52	.104
70	.15	TR	.13	.004		.27	.010
80	.07	TR	.07	.002		.25	.009
90	.15	.013	.11	.006		.27	.007
100	.20	.007	.14	.007		.20	TR
100- 110	.30	.007	.23	.008		.95	.056
120	.17	.008	.19	.004	Tr.	.35	.008
130	.17	.026	.19	.041		.25	.030
140	.20	TR	.18	.005		.30	.030
150	.25	TR	.31	.006		.30	.034
160	.32	TR	.33	.009		.30	.021
170	.25	.017	.27	.027	Tr.	.30	.052
180	.32	.178	.32	.129		.35	.152
190	.09	.005	.10	.012		.32	.130
200	.10	.008	.08	.022		.20	.082
200- 210	.07	.005	.08	.009	Tr.	.35	.052
220	.17	TR	.17	.011		.25	.030
230	.17	.013	.19	.019		.25	.030
240	.25	.008	.28	.015		.25	.026
250	.15	.005	.13	.024		.25	.034
260	.37	.003	.34	.011	Tr.	.27	.026
270	.10	.030	.20	.066		.47	.052
280	.25	.007	.21	.015		.40	.043
290	.30	.009	.30	.017		.40	.052
300	.20	TR.	.17	.009		.40	.056
300- 310	.35	.005					
320	.25	TR.	.34	.030	0.07	.30	.056
330	.30	.013	.26	.018		.40	.043
340	.17	TR.	.29	.009		.40	.026
350	.17	.013	.19	.009		.38	.083
360	.27	.008	.17	.038		.22	.026
370	.72	.003	.36	.031		.57	.039
380	.70	TR	.68	.013	0.07	.55	.021
390	.35	.008	.78	.010		.55	.021
400	.67	.008	.36	.015		.42	.005
400- 410	.92	.005	.73	.022		.55	.030
420	.65	.003	.90	.025		.75	.067
430	.30	TR	.70	.011	0.04	.37	.013
440	.50	.026	.32	.009		.37	.005
450	.45	.030	.45	.024		.45	.021
460	.45	TR	.36	.036		.30	.013
470	.32	.039	.41	.011		.70	.008
480	.42	TR	.38	.057	Tr.	.70	.021
490	.27	.003	.50	.023		.52	.021
500	.35	.005	.28	.017		.40	.013
500- 510	.30	TR	.48	.015		.37	.010
520	.27	.005	.30	.007		.40	.018
530	.12	TR	.30	.004	0.06	.30	.007
540	.12	TR	.18	.004		.25	.011
550	.20	TR	.13	.010		.25	TR
560	.22	TR	.18	.005		.30	TR
570	.07	TR	.19	.007		.27	.005
580	.15	TR	.09	.002	Tr.	.27	.008
590	.10	.008	.08	.002		.20	TR
600	.07	TR	.14	.017		.15	.008
600- 610	.05	TR	.10	.019		.20	TR
620	.25	.274	.08	.013		.14	.008
630	.10	.005	.22	.209	Tr.	.15	.043
			.09	.015		.15	.030
0-520	.293	.010					
0 - 590	.275	.009					
0 - 630	.265	.013	.271	.020		.373	.029
						.359	.029

APPENDIX V11

DIAMOND DRILLING

HU-4

FOOTAGE	WILLIAMS Cu.	WILLIAMS MoS2	WILLIAMS Ag.	COAST ELDRIDGE Cu.	COAST ELDRIDGE MoS2	COAST ELDRIDGE Ag.	SLUDGE Cu.	SLUDGE MoS2
0 - 10	.20	TR	TR	.23	.006	TR	.35	.008
10 - 20	.20	.008		.15	.013		.30	.005
20 - 30	.20	.005		.17	.007		.32	.003
30 - 40	.05	TR		.06	.002		.17	.005
40 - 50	.09	TR		.17	.007		.37	.017
50 - 60	.30	TR	.15	.30	.011	TR	.42	.034
60 - 70	.30	.021		.25	.024		.50	.228
70 - 80	.40	.205		.44	.206		.55	.252
80 - 90	.20	.014		.18	.019		.40	.117
90 - 100	.07	.005		.05	.004		.17	.034
100 - 110	.07	.005	TR	.07	.002	0.16	.22	.026
110 - 120	.09	.010		.06	.001		.15	.026
120 - 130	.15	.007		.12	.010		.20	.008
130 - 140	.05	.008		.09	.008		.17	.021
140 - 150	.10	.011		.09	.007		.17	.017
150 - 160	.07	.014	TR	.08	.007	0.04	.15	.030
160 - 170	.07	.013		.05	.004		.17	.052
170 - 180	.09	.021		.08	.002	0.06	.20	.113
180 - 190	.25	.457		.28	.670		.50	.030
190 - 200	.42	.009		.24	.026			
200 - 210	.37	TR	.05	.20	.009	0.05	.60	.034
210 - 220	.15	.034		.15	.019		.35	.030
220 - 230	.30	.005		.25	.012		.60	.039
230 - 240	.45	.095		.17	.078		.65	1.543
240 - 250	.70	.021		.31	.049		.67	.260
250 - 260	.30	.007	TR	.34	.017	0.07	.32	.135
260 - 270	.07	TR		.06	.004		.22	.056
AVERAGE:	.21	.036		.172	.045		.34	.120

W. G. HAINSWORTH

CONSULTING GEOLOGIST

CERTIFICATE

I, W.G. Hainsworth of Vancouver, B.C. do hereby certify:

1. That I am a Consulting Geologist residing at 4664 Clovelly Walk, West Vancouver, B.C.;

2. That I am a graduate of the University of Western Ontario, B.Sc.;

3. That I have practiced my profession for 18 years;

4. That I am a member in good standing with the Association of Professional Engineers of British Columbia;

5. I have no interest in the Highmont property. I do not hold any securities in Highmont Mining Corp. Ltd. and I do not expect to receive or acquire an interest.

6. That I have 3000 shares of Torwest Resources (1962) Ltd., which I bought through a member house of the Vancouver Stock Exchange and, further, I have never had and do not expect to have any options on Torwest Resources (1962) Ltd.

7. That the information contained in this report is based on personal knowledge of the property from previous and present supervision of work and examination of maps and data pertaining to the property and the area in general.

W.G. Hainsworth

W.G. Hainsworth, P.Eng.

Vancouver, B.C.
August 8th, 1968.