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REPORT

PATMORE GOLD MINE, KYUQUOT SOUND, B.C.

CLAIMS:

The Filmil group is comprised of 12 full-sized, unsurveyed mineral claims staked 3 across and 4 along the length of the veins, thus thoroughly protecting the ore bodies. Originally located by J.J. Pugh and associates in 1938, these claims are now held in "lifetime" (of the Property) lease by Dr. W.H. Patmore (mining Geologist -- graduate Princeton University). Exhaustive surface exploration combined with detailed panning has shown that most of the favorable geological structure is embraced by these claims. All the surrounding ground is held by various prospectors.

HISTORY:

This group was examined in 1938-40 by numerous field scouts (Pioneer, Premier, Privateer and others) who turned it down because of insufficient surface exposures. Man-O-War mines held the ground under option for two months but dropped their rights without driving any tunnel. A.E. Trites was negotiating for some months but lost his chance through delay. Some surface stripping was carried out by the owner. The present leaser, encouraged by the considerable amount of coarse gold that could be panned below the vein, took over the claims in 1941-42.

LEASE:

The lifetime lease is on a perpetual, gross royalty basis and involves no cash payments on a time schedule. The owner is to get 6% of the gross value of the mineral recovered from the ore (shipped or milled) for the first two years of actual production. Thereafter he gets 7% of the gross. At our present average mill heads ($30 per ton) this royalty amounts to a direct cost of $1.80 per ton. The owner will consider any offer to purchase his royalty share, but, since the completed mining has given him a fair idea of the possible future value of his property, this offer must be attractive. He believes that at least two million dollars worth of ore will be found, giving him, at a gross royalty of 6%, $120,000. Due to the high rate of income taxation on royalties he might settle for less as "return of capital"payments. $35 oz

Any company taking over the leaser's rights would have to satisfy the terms of the present lease. Aside from paying the royalty, this would mean:

(1) Commencing construction of a mill as soon as war conditions will allow (size of mill at leaser's discretion).

(2) Keeping at least 4 men continuously at work as soon as labor procurable.

(3) Keeping the claims in good standing and free of all liens, etc.

The owner agrees to pay for all costs in surveying and crown granting his 12 claims, the cost to be held against his royalty share.

The lease is such that the property may be "frozen" for the duration or until men and machinery are available.

LOCATION:

The claims lie halfway along and on the south side of Deep or Amai Inlet, Kyuquot Sound, Vancouver Island, B.C. and cover both slopes of the 2500' ridge between Deep and Cachalot (or Narrowgut) Inlets. The beach camp is 15 miles by water (mostly sheltered) from the fishing village, supply center and post office of Kyuquot. Chamiss Bay logging camp lies 9 miles distant. This area is just 12 airline miles from the mining town of Zeballos.

ACCESS:

Planes from Vancouver (C.P. Airlines) will stop or call at the beach camp of the mine on request. Their official schedule is weekly (Wednesday in summer and Saturday at other times). The two hour plane trip costs $32 and 40 lbs. of baggage are allowed.

The C.P. Steamers (Maquinna or Norah) stop only at the nearest port of call, Chamiss Bay logging camp, on a variable 10 to 11 day schedule. The entire trip from Vancouver via Nanaimo-Port Alberni or via Victoria costs about $25 including meals, taxis, tips, etc.. A launch must be hired to get from Chamiss Bay (or from neighboring Kyuquot) to Deep Inlet ($7-$10) unless the small mine boat happens to be at Kyuquot where it calls for mail and supplies. Samples may be shipped from Chamiss Bay via the steamer.

TRANSPORTATION FACILITIES:

The lowest showings (elev. 1360') occur less than 1 mile (4500') from tidewater and a good harbor, while those of Zeballos are between 4½ and 9 miles from the beach. Thus, the Patmore mine has the distinct advantage of low cost transportation involving supplies, fuel oil, machinery and concentrates. None of these need be trucked over 1800' and the fuel will be used at the proposed beach power house. At the present, concentrates or ore may be shipped to the Tacoma Smelter for $4 a ton (prewar charge was $5).

TRAMLINE:

This advantage has already been enhanced by the almost complete installation of a 4000' tramline extending from the mine camp (elev. 1200') to a large, spruce log float (150' x 40') secured in a storm-sheltered bay on the south side of Deep Inlet. Note that very little work at a low cost will complete this tram and allow immediate erection of a 10-ton mill or further ore development.

The lower part of the tramline (1600') is only of temporary construction (single reversible, light duty) since it must be replaced by a [illegible]000' gravelled truck road when and if a 25- or 50-ton mill is justified. The upper part of the line (2400') has heavy duty, double reversible, standa[rd] type towers and trestles (cedar pole fabrication) all set on solid bedrock. Three towers have yet to be completed at an approximate cost of $1500. These upper line towers, with double length, yellow cedar saddle bars, are ready for the switch-over from light, single to heavy, double reversible type if the proposed 25- to 50-ton mill is built. The change will require only new large cables ...

WHARF:

At such time as the installation of a large mill it may be necessary to build a dock of creosote pilings with a 100' approach and an 30' head. This could be done cheaply with government aid after the war. However, a wharf is unnecessary until heavy machinery (units over 2 tons) is required in quantity.

ROAD:

The proposed 2000' length of road includes all switchbacks, bends and a turntable at the mill as well as a junction with the proposed dock approach. By comparison with the roads of Zeballos the grade is good and it varies between a maximum of 14° and 0°, averaging under 10°. Two thirds of the timber along the right-of-way is already cut though not removed to the side. Just one cheap 20' bridge (with solid rock approaches) need be built. Due to its shortness it will require only heavy stringers and 3" decking and should not cost over $200. Few culverts are necessary but the inside edge of the road should be well ditched and the surface well gravelled because of excess seasonal rainfalls. Gravel may be had from an adjoining beach and from the nearest creek bed. Contractors estimate that this road can be built for $2.25 to $2.50 per foot or a maximum total of $5000. A large part of this cost will be borne by the government (after the war) according to its pre-war mine-aid plan. A similar "fore and aft" style of temporary logging road could be built for $2500 to $3000 but it would last for only 5 or 6 years.

Neither the dock nor the road is required until the next stage of development indicates that a 25- to 50-ton mill is warranted. The instal ation of a 10-ton mill may be carried out without a road or a wharf as the machinery is light andmay be dismantled into units weighing less than 400 lt

MILL SITE:

There is a good mill site for a 25- or 50-ton mill at a point on the tramline 1600' from the beach. It lies at the foot of a 35° slope which grades off to 15°, thus allowing a choice of a 25° slope for gravity feed through the mill with the corase ore bin built directly behind on a steeper grade. The site is already cleared of trees including all possible windfalls. Creek water need be piped less than 1000' to secure sufficient head at the mill. There is a relatively flat and safe camp site below and within 200' of the mill site. This is partially cleared. Excellent tailin disposal ground and grade is present within 200' of the mill site. A broad nearly flat area at the foot of the mill offers a good turntable for the required truck road connection from mill to beach (2000' road). A proposed crosscut, giving 360' to 480' more backs on the ore, will reduce the tramline length to either 1060' or 1230' instead of 2320' from portal to mill, and will also eliminate a horizontal angle station.

There is a fine mill site for a 10-ton plant within 600' of the present lowest adit and within 450' of the water supply intake. This site is 150' from the mine camp at the tram terminal. It would require an auxiliary tramline 900' in length. This section is cleared for 600', leaving 300' to be finished. A single, central line tower may be needed (to supply clearance) together with the two terminals. The cable should b ¾" or 1" for heavy duty.

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TIMBER:

The trees are mainly cedar although hemlock and balsam are common at lower elevations. Most of the cedar is red and excellent "shake" trees are numerous. All the towers are built of red cedar poles because of its lightness, availability and long life. Yellow cedar (stronger) is used for saddle bars.

Plenty of mine timbers (yellow cedar and hemlock poles) are present close to all the adits, although very little timbering is needed.. Good cabin logs are available at all elevations. There is an abundance of fine firewood (fir and yellow cedar) al all camp sites.

WATER:

For camp purposes the supply of water at the present mine camp is sufficient although meager. If development justifies a 25- or 50-ton mill, the mill and mine camp will be combined at a site 1600' from the beach where there is normally a good supply of camp and mill water. There is insufficient water for anything larger than a 10-ton mill at the mine. A still greater supply of water may be had by piping water 3500' from the main creek of the area (McKay Creek).

POWER:

There is a choice between a diesel-electric and a hydro-electric plant but with either type the cost of power will be low relative to its cost at the Zeballos mines. This difference is due to the high cost of trucking oil and of hauling in power machinery. Zeballos mines paid 8.6 cents per gal. for fuel oil at the beach and between 11 and 12 cents at the mines. Diesel power is developed by the Zeballos mines at a rate of about 1½ cents per kw-hr, but it should cost less than 1 cent at Deep Inlet.

A 50-ton mill and mining plant would require between 200 and 250 H.P. or a maximum of 186 kw. A diesel engine of this size would use about 100,000 gals. of fuel per year, costing $9000. at tidewater or $12,000 at the average Zeballos mine, a relative saving of $3000 or more per year for any beach installation.

Th initial cost of a 186 kw hydro-electric plant plus the 9000' of power line necessary to reach from the Cachalot Inlet power site to the mill would approach $25,000, whereas a similar sized diesel-electric plant with 5000' of power line and 5 fuel oil taks, could be built for $15,000. However, including maintenance, interest (at 5%), amortization (4% over 5 years), government fees and insurance as well as fuel costs, the annual cost of a diesel plant would amount to about $20,000 while the hydro-electric plant could be operated at a cost of $8000 thus saving $12,000 yearly and paying off its higher initial cost in one year.

The hydro-electric site has a useful head of 90' (with a dam 4' high and subtracting 6' for the power house). It lies at the head of the Narrowgut River flats, approximately 5500' from tidewater. 400' of pipe 18" in diameter would be needed to carry the water from the proposed 10' intake tunnel to the power-house. The intake cuts and tunnel could be driven by hand mining for $500. The flats from tidewater are such that a cheap "cat" road could be built to the site in a short time to aid in hauling in the machinery and pipe.

However, there is no storage space for water above the dam site and the available volume would develop 250 H.P. for two thirds of the year only, its output receding to approximately 80 to 100 H.P. during the dry months (sufficient for the mill alone). During abnormally dry years the developed H.P. would be still less for the one to two driest months (volume under 8 cu. ft. per sec.). Thus, an auxiliary diesel plant (150 H.P.) would be required, raising the initial cost of the hydro plant by another $8000 to $10,000. Such a combined hydro-diesel electric unti would save its entire extra initial cost within 5 years.

A diesel-electric power house could be most easily and economically set up alongside the jnction of the proposed truck road and wharf approach, where there is a satisfactory, level, granodiorite foundation, as well as room for 3 large wooden fuel-oil tanks of 3 months or 22,500 gals capacity (3 storage tanks cost $900 before installation). An oil pipe connection with the dock would allow tankers to pump the tanks full without further handling.

The 1600' power line (using the present tramline right-of-way and towers) from a transformer at the power house to one at the mill site could be cheaply installed due to the closely spaced (300') completed towers Two transformers would cost about $1500 and 1600' of line should not exceed $1000. A further 1200' to 1400' of line would be necessary to carry power from the mill up to the proposed crosscut portal where a transformer and an electric compressor would need to be installed. Otherwise, compressed air would have to be piped from the mill to a receiver at the portal.

A 10-ton mill would require a 30 H.P. gas engine at the upper mill site (cost $450). This could be changed later to a diesel oil user by adding a $350 Lauder exhaust attachment which simply "cracks" diesel oil by reason of waste heat, thus lowering operating costs. Fuel would have to be hauled in on the tramline at the rate of 1500 gals. per month (2 drums per day). This would cost $345 per month for gasoline or $150 for diesel oil at a saving of $200 per month. Thus, the choice of burning diesel oil woid result in a saving of $200 per month which would soon pay for the Lauder converter.

PROPOSED CROSSCUT:

If the next stage of development justifies installation of a 25- or 50-ton mill (i.e. sufficient ore is found on the present lowest or 1400' level) it will be advisable to open up the ore for milling by driving a long crosscut from a point several hundred feet below the upper terminal of the tram to a zone approximately beneath the portal of the lowest level.

Tape and Brunton surveys show that a 1000' crosscut will give 360' of "backs" on the ore (95' to 100' extra would have to be allowed for vein dip which averages about 75° away from the proposed portal). By lengthening the crosscut another 160', a further 120' of backs could be gained. Thus a total of 1100' would give 360' of backs or 1260' would add 480' of backs up to the 1400' level. Below these alternative portal sites the slope decreases from 45° to about 30° for a slope distance of 250' and then flatten out to 0° for 300'.

Depending upon the choice, this crosscut would elimiaate either 1090' or 1260' from the total length of the final tramline, reducing it from 2320' to 1230' or 1060' thereby obviating the building of several towers and and angle station as well as discarding two towers and one trestle. A coars ore bin would have to be erected at the portal and another at the mill site. The shortness of the final tram would allow larger tonnages to be hauled, an an increase to $1\frac{1}{4}$" in size of the track cable would give a sufficiently high safety factor to permit haulage of miners from the mill-camp site to the portal. This scheme would entirely eliminate the need for the upper cam

o . Since the rock is a tough granodiorite the crosscut must be driven by machines and an electric compressor. There is a satisfactory grad for hauling a compressor and receiver up the mountain on skids. As an alternative, the compressor could be set up at the mill site and air piped up 1200' to 1400' to a receiver at the portal. This would eliminate skidding of any heavy machinery beyond the truck road, and would remove the need for 1400' of poer line although 1400' of air pipe would be required to take its place.

The crosscut should cost less than $15 per foot or a total of $16,500 to $18,900 depending on the above choice of length. This cost estimation is made relative to mining experience in nearby Zeballos where the same rocks are encountered. For example, Central Zeballos, for slightly ove $15 per ft., drove a 1600' crosscut through very hard, highly silicified limestones and granodiorite. It is possible that one of the numerous dikes which occur in the area may be encountered thus affording cheaper mining. There is also a likelihood that other veins may be intersected.

MINING EQUIPMENT:

This crosscut will require drill steel, 2 drifters, 3000' of track, several ore cars, and air and water line as well as a compressor and receivere. Two or three cars will be sufficient until ore is being trammed through this main haulage way, when an electric locomotive and ten 1-ton cars will be necessary. The preliminary cars, machines, compressor etc. could be purchased for about $10,000.

None of this equipment is needed for the secondary stage of development as this can be completed by hand writing and the use of rubber-tired wheelbarrows. Also, if it is planned to proceed with a 10-ton mill none of the above will be essential for the first few months operation. Afte that, a small compressor, a drifter and two ore cars will be necessary for certain difficult sections found along the vein.

CAMPS:

At present there are two log cabins at widely separated points on the beach. The larger one (20' x 18'), situated near the float, is used for a warehouse and cook house as well as for limited sleeping quarters. Much of the tramline equipment (saddles, cables, bolts, engines and carriage wheels) is stored in one third of this building partitioned off from the kitchen and manager's room. The large attic is used for storing roofing pape kegs of nails etc.. There is also space for several camp cots. The smaller cabin (12' x 14') is built near an adjacent beach where the trail turns away from the shoreline towards the mine. It is used for a packing shelter. Plan call for a 14' x 16' cabin to be constructed close to the larger building. It will form a stopover bunk-house for visitors and personnel. A small loadi shed should be built at the tram terminal. Four men spent 20 days completing the larger cabin -- it was later improved in many ways. Its value would approximate $800 to $900. The smaller cabins should not cost over $200.

There are, as yet, no builders at the mill site camp and it is only partially cleared. Four men cou'd finish this clearing in 1 or 2 days. The site is reasonable flat, spacious, close to the road terminal and mill site, and abundant creek water flows alongside. The proposed camp woul house both miners and millmen as it would require only 3 to 4 minutes to hau the miners to work on the tramline.

The upper (mine) camp (elev. 1200') has two log buildings at present. One has been used for a bunkhouse (4 or 5 men) but it is too small (12' x 14'). Therefore, another bunkhouse is necessary at once. It should be at least 16' x 16' to accomodate 5 men as well as a table and a heater. The smaller cabin can then be used as a drying-house. A 12' x 14' wood shed power saw house should be constructed at this camp. These two buildings should not cost over $500. The other cabin is 16' x 18' and forms a spaciou cookhouse and cook's quarters. It is entirely of log and shake construction and cost about $600 to complete. A large food storage room was later added its front verandah. The mine camp lies 900' by trail from the lowest portal When and if the mine is operated on a 10-ton basis this camp will be the permanent one.

Camp water is temporarily drawn from a seepage well 125' above the cabins but another 450' of 1" pipe (cost $68) would supply the buildings with fresh creek water.

It should be realized that the next stage of development, whether it is installation of a 10-ton mill or whether it is further ore development to the 25- or 50-ton class, requires very little camp constructi This fact should allow rapid achievement of either objective.

REGIONAL GEOLOGY:

The ore-bearing veins and their dikes lie in north-south fractures and shear zones which cut, at steep dips, an almost circular boss of pinkish granodiorite about 6 miles in diameter embracing all of Deep Inle This intrusive, if it does not actually make a surface junction with the sou western contact of the main Zeballos granodiorite (grey), must meet its flan at shallow depths. These granitic rocks are easily distinguished in the fie by the almost totally light grey to blackish (dioritic phases) appearance of the Zeballos batholith and by the abundant areas, small masses and irregular dikelets of pink to red orthoclase feldspar so common in the greenish Deep Inlet intrusive. There is no doubt that both types may carry gold deposits. Apparently, the two are relatively superficial phases (differing in age as well as in composition) of an underlying, much larger, parent batholith, whi during the last stages of its activity, probably gave off the valuable metal-bearing solutiions that filled suitable structural and cooling fractures in localized projecting cupolas formed by its earlier, overlying differentiates and their volcanic host.

A third intrusive mass of irregular, elliptical shape, lies between the center of Narrowgut River valley and Port Eliza Inlet, with its longer axis bearing south-westerly, parralel to Espinosa Inlet. It is much like the Zeballos granodiorite in appearace but is normally coaraer and dark varying slightly to a dioritic type. No gold deposits have yet been located in the Narrowgut stock although it approaches to within 3000' of the souther margin of the mineralized Deep Inlet boss along the north wall of Narrowgut valley. However, it may be of significance that the pink intrusive is only gold-bearing adjacent to this grey stock.

A large proportion of the older roof (or host) rocks of the are

are massive black to green volcanic flows of andesitic to bassitic composition. Local intercalations of impure grey limestones are found in scant quantity. Volcanie breccias and purple tuffs are common in parts of the ar but light-colored, fine-grained felsic tuffs are scarce. A few veins of relatively limited length have been located in the volcanic roof rocks but always close to the border of the granodiorite. These older rocks display very little obvious high grade metamorphism or metasomatism although dikele of epidote, magnetite and orthoclase are often observed.

LOCAL GEOLOGY:

Many blackish-brown to green mafic dikes (probably lamprophyre and relatively few felsic (aplite) dikes transect the pinkish granodiorite a north-south, closely-parallel system. Their maximum concentration, togeth with that of the gold-bearing veins, occurs within a few thousand feet of, a on both sides of, a limited, embayed section of the intrusive-volcanic conta not far from the neighboring Narrowgut stock. Only six dikes or irregular masses of aplite have so far been found and but three of these show any sign of gold mineralization. Most of the lamprophyres and several of the felsic dikes display great persistence over lengths and depths of several thousand feet even when of narrow width. At least six of the mafic dikes carry gold values although in the majority of them the mineralization appears to be limited to sporadic occurrences.

Some of the veins lie directly in the granodiorite but most of these are typically of gash-filled origin and do not give promise of being commercial. Others lie in the volcanics but these, too, seem to be somewhat limited by the unfavorable fracture-sustaining nature of such rocks.

STRUCTURAL CONTROL:

It is obvious that certain structural conditions have had a great influence in determining which fractures would become strongly minerali Apparently, the major sturctural control has been intensive compression expressed as north-south shearing mainly confined to a few dikes which themselves have occupied lines of earlier tensional weakness. There is no doubt that the shearing of the main ore-bearing lamprophyre persists to a length and depth of several thousand feet. Where conditions are favorable, this crushed dike (though often but a few inches wide and commonly branching or coalescing) may be traced, almost uninterruptedly, for the complete width and height of Deep Inlet ridge because its mantled portions have unmistakably determined the sites of erosional gulleys. Thus, the main shear zone has formed the trough for a substantial creek. However, it should be realized that only a small part of the entire length is favorable to gold mineralizati Other structural features have played an important part in localizing ore shoots, but this strong horizontal movement (proven by horizontal striae) appears to be one of the prerequisites in the formation of a commercial ore body insofar as Deep Inlet is concerned. Numerous other dikes contain highly comminuted portions but they are seldom sulphide or gold-bearing. That is, shearing is usually found with the favourable minerals but the converse is fa from true. In a few localities the crushing embraces small sections of the tougher granodiorite or large inclusions within it although, as a whole, the main intrusive mass seems to have acted as the agent transmitting the stress to its component healing dikes, the weak lamprophyres offering the easiest relief to long-term strains developing within the compressed block. The movement was continued for an appreciable interval as is evidenced by conjugat fracturing of the main dike and by the "rubbelized" quartz and "slickensided" pyrite which fill such fractures. This fact has been of prime importance in assessing the low cost of mining and milling operations. A second lamprophyr dike on the Patmore property displays than the extensive comminution of the main ore zone. The brittle aplites characteristically show oblique cross-fracturing with minor strike-shearing. Apparently they are less susceptible to the latter type of stress.

Other structural features that may have offered direct control to ore shoot localization are systems of flatly-dipping (30°±) cross-striking (east-west) dikes that are concentrated along portions of the shear zone, but their relation is uncertain as yet. Rolls in dip or intersections with branch veins may be important factors.

A pronounced association of aplite and lamprophyre occurs along the main ore zone. The two complementary dikes roughly parallel each other for several thousand feet, averaging about 15' to 20' apart where exposed but actually coming together at the deepest point of underground exploration (face of the lowest tunnel). At the junction, a whitish, clayey gouge seam about 1' wide indicates the intensity of motion and accentuates the more rigid nature of the dense, fine-grained aplite, marked only by tiny, sulphide-bearing, cross-fractures. So far, no ore has been located in the lamprophyre north of this contact although some ore is present in the aplite itself along minor shearing. On the other hand, the best ore so far developed has come from the lamprophyre south of the junction, whereas none has been found to date in a similar section of the aplite. Thus, it would appear as though the mineral solutions had switched over from the mafic to the felsic host at this point even though the northern extension of the lamprophyre is moderately sheared. This may be substantiated by the fact that, at the portal of the middle tunnel the vein lies entirely in the granodiorite between the two dikes.

MAXIMUM FAVORABLE RANGE:

From the standpoint of general structure it would seem that the most favorable horizon for exploration would be that portion lying closest to the original roof of the granodioritic cupola, since here would be the maximum weakness and hence the greatest susceptibility to movement. The older, volcanic-sedimentary rocks were originally domed over the gold-bearing block but erosion has removed them from most of it. However, this stripping has taken place in such a way that it may readily be seen that none of the known veins is at any great distance (max. 3000') from the ancient roof. From the discoveries so far made, it seems likely that the favorable portion of the cupola will not exceed a length of 4000', a width of 3 miles and a less-definable vertical range of at least 4000'. The vein distribution is along this contact or near-contact area and does not extend to parts of the granodiorite or volcanics that are far from the site of the original roof-structure. These points are important in regard to the chances of finding further segregations of ore shoots along the vein-shears (which are strong and well-defined in themselves) as drifting proceeds southerly into the ridge. Even though these drifts are now approaching (1600'-2000') the present intrusive-volcanic contact (a flank of the old roof) on the far side of Deep Inlet ridge, a part of the ancient contact (now eroded) was formerly present only a short distance (perhaps 1500' or less) above the adit portals. Thus, in a general way, the entire section through the ridge top to several thousand feet in depth is of structurally favorable nature and the presence of commercial gold values on the south side would appear to substantiate this theory. The exact localization of segregations of ore shoots is further dependent upon secondary details such as cross-faulting and rolls etc. whose influence was superimposed on the major structure. Their importance cannot be fully estimated until development is more advanced.

FAULTING:

A series of pre-ore coss-faults (approx. east-west and flatly-dipping), some of considerable magnitude, may have been the channels of access followed by the rising mineral solutions as they entered the structurally-favorable gold horizon or cupola. These are expressed on the surface as gravel-filled draws and in the creeks as 90° changes in course. They are to be seen in both the intrusive and the older rocks.

So far no sign of fault displacement has been noticed in the ore bodies except as strike-shearing (closely-spaced faulting) in the plane of the vein. This, of course, is a great aid in low-cost development and stoping. A strong cross-fault may be observed in the lowest tunnel where it intersects the two parallel dikes without seeming to disturb either of them. It may be significant that an ore shoot lies in the aplite where this fault zone (3' wide) meets it, although the lamprophyre is barren at the opposite end of the fault except for calcite and chlorite.

In summation, it is extremely fortunate that faulting has occurred along the plane of the vein since the resultant comminution means reduced drilling, blasting and crushing costs. And it is no less fortunate that it has not displaced the ore bodies laterally which also promises low costs in development and stoping. Furthermore, one of the cross-shears has already been used as the site of a cheaply-driven crosscut.

VEINS:

Numerous gold-bearing veins of different types have been located in the favorable horizon. They occur as lenticular gash-shears and tight gash-fissures in the granodiorite; as narrow, irregular fissures in the volcanics and as combined fillings and replacements along shear zones in both aplite and lamprophyre dikes which transect the intrusive rock. Besides these deposits, there are several instances of "knife-blade", cross-fractures (carrying visible gold and massive sulphide) spaced 1" to 12" apart along the length of the dikes. One aplite dike contains disseminated auriferous cube-pyrite, apparently as a replacement streak 8" to 12" wide well within its walls. It is evident that most of the mineralization in the other deposits is of the filled-fissure type with limited replacement. There is one striking example of visible gold and much chlorite occupying the intergranular spaces and tiny slips in a narrow shear in granodiorite. Quartz and sulphides are entirely absent from large sections of the shear.

Nine different veins and mineralized dikes have been located on the Patmore property so far but the majority are far too narrow to be commercial. Unless shearing is prominent and persistent the veins warrant little attention. However, at least four of them display sufficient structural strength to encourage surface stripping with limited drifting. All of the veins are closely-parallel and they lie within 300' to 400' of each other. It seems likely that "blind" deposits will be encountered during crosscutting since many sheared dikes, without surface ore have been discovered between the known veins. The simple gash-fissures are typical open-cavity fillings composed of sugary quartz crystals, pyrite, oxide and scattered crystalline gold. Vugs showing nearly euhedral quartz are common. These veins vary between a knife-blade and 2" in thickness and do not appear to persist for much more than 100' in length. They are not associated with dikes and are not found outside the granodiorite.

Of the more important deposits, a quartz-healed, brecciated and sheared lamprophyre is worthy of more stripping and a short drift because of its structural strength, more abundant quartz and scattered sulphides. It yields interesting gold assays from several of the open cuts which expose it, in generally narrow widths, for a length of 150'. A second showing of importance is a recently-located, highly oxidized, 6" to 10" shear-zone lying in the dark volcanic rocks several hundred feet from the roughly parallel granodiorite contact. The flank of the intrusive plunges toward the deposit. Due to heavy overburden, this vein is poorly exposed by a single open-cut but channel samples so far taken (3) have assayed between $30 across 7" and $95 across 10". Very little sulphide has been seen in the oxide and gouge, and the adjacent volcanic rock is appreciably replaced by auriferous oxidized pyrite. A third vein which gives some promise is a quartz and pyrite filled-fissure that lies in the granodiorite about 250' east of and parallel to the main ore zone. It is relatively narrow (1" to 3") but may be seen at various points for at least 250' in length. Since it dips more steeply than the main vein, it may possibly intersect that deposit at a few hundred feet below its outcrop. Therefore, a relatively short crosscut from the lower level (along one of the cross-dikes) would permit cheap exploration at depth. This vein shows a little visible gold and carries appreciable values ($40 to $70) in 2 composite samples).

More prospects

MAIN VEIN:

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The main vein displays the greatest strength of structure since it lies within a shear-zone that persists for several thousand feet horizontally, as well as vertically, even though it changes its host along the strike. The shearing is confined mainly to the lamprophyre dike but it, or an offshoot, locally enters the granodiorite and, to a minor extent, the aplite. The ferromagnesian lamprophyre is much more continuous than its closely parallel, felsic neighbour and may be seen at intervals cleaving the entire ridge from Deep Inlet to Narrowgut. The aplite varies between 4' and 8' in width while the dark lamprophyre is between 1' and 6', averaging about 3'. The quartz commonly follows one, or sometimes both walls of the mafic dike but it may also be seen at any point across its width or in the enclosing granodiorite. The present work has exposed widths of quartz, oxide and sulphide, ranging from a knife-blade up to a maximum of 36" (including veins on both walls). Barren shearing may occur between ore shoots and vein "pinches". Since the shearing shows a maximum width of 6', further drifting could expose greater widths of quartz.

MAIN VEIN AVERAGES:

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So far, the average width of the ore shoots is slightly over 8" and approximately 40% of the total initial development has been in ore. However, this latter figure cannot be taken as a mine average because of the limited amount of drifting and the inevitable waste footage needed to gain "backs" on the ore shoot zone. During secondary exploration, the average footage in ore should improve markedly. To date, ore shoots, separated by pinches, have been located over a total vertical range of 450' and over a horizontal range of 850'. The average width (8") should also improve because of the remarkable shoot now being developed on the second level and because of the fact that mining results indicate that the top level merely penetrated the uppermost portion of the ore shoot zone. (The ore lies 100' further into the hill on the top level than at the surface above, and both widths and values increase progressively down and in along a direction pitching diagonally into the ridge.) The rich second-level shoot, averaging $100 per ton across 10", displays a width of over 2' for the last 6' of drift. The bottom level is not yet into the main ore shoot zone but it should reach it within 50' to 100'. A strong, wet gouge streak and scattered particles of oxide appear to signal its nearness. Branch or feather-veins are common and they enter the main zone at acute angles.

POSSIBILITIES IN DEPTH:

The maximum depth of any one vein is limited by the 4000' ore more of the favourable cupola-block already discussed but upon this must be super-imposed further restrictions resulting from more localized structural controls of as yet somewhat indeterminate effect. These may include the attitude, concentration and persistency of cross-faulting or of cross-dikes or perhaps simply the extensive depth and obvious strength of the better known main shear-zone. It seems fairly certain that depth possibilities are best in that portion of the block lying under the sloping flank of the volcanic roof toward the far or south side of Deep Inlet ridge. That is, as greater depth is attained, ore finding chances should improve progressively with drifting through the hill toward the roof remnant, since this steeply-plunging structure (50^{o}-70^{o}) apparently allowed very little avenue of escape to the rising mineral solutions (indicated by lack of deposits above the contact). There is little doubt but that the shearing and faulting penetrate well down through that flank, thus offering channels of access. The unknown factors are the temperature-deposition conditions (restricted by distance from the source) and the presence or absence of suitable trapping structures at the lower horizons. Only further exploration will bring out these details. The structural control which determined the position of the present ore shoot zone must first be ascertained and then it may be possible to extrapolate and trace these features or discover similar ones along the shear in the most favorable portion of the cupola-block. That some such secondary structure must have played its part is emphasized by the several hundred foot barren portion of the shear lying above the main ore shoot aggregate. It is obvious that the presence of appreciable gold values (2 oz. across 2"-4") in the same shearing right at the granodiorite-volcanic contact (roof-remnant) on the Narrowgut side of the ridge 1600' distanr, favours a considerable addition to the vein length, Furthermore, e newly discovered, parallel, gold-bearing shear has been located on that same side of the mountain.

More prospecting

MINERALS:

The ore is simple rather than complex and few ore minerals have been identified so far. Of these, pyrite is the most abundant but even the pyrite is relatively scarce because of extensive oxidation. A little sphalerite and still less chalcopyrite, usually oxidized to malachite, have been encountered. Galena, so apparent in other Vancouver Island camps, is entirely lacking but the far less common tellurides are visible in scattered specimens, especially where appreciable coarse gold is to be seen. Field examination of the small specks is not sufficient to allow determination of the exact species but tetradymite and possibly sylvanite are thought to be present. Chemical analysis has substantiated this observation. Mill tests indicate that the amount of telluride in the ore will not affect the recovery of gold. The gangue minerals are mainly quartz with a little chlorite and sericite. Calcite is rare where the ore is commercial.

PRIMARY GOLD:

possible specimen ore

Visible gold is plentiful and it varies from almost invisible specks to grains as large as a small pea. Much of it is crystalline and grinding tests have shown that it is easily divisible and will pass readily through fine screens. The gold lies inside pure white masses of non-porous quartz as well as amongst bunches of chocolate-brown oxide of iron, and its presence in quantity in this state at over 125' in depth would suggest primary, rather than secondary, deposition. With so much oxide showing, even at the deepest face, there is always the question of some secondary enrichment but it should be remembered that there is no comparison between the ease and rate of oxidation of relatively unstable pyrite and the ease and rate of dissolution of almost insoluble gold. Besides, considerable gold may be seen confined to a few inches near and at the surface amongst pure oxide in the wettest and highest part of the ore zone where, apparently, it has not been carried downward with the supposed effect of impoverishing the surface and enriching the vein beneath. Rather, this surface gold is of the extended "wire" type forming strings 1/2" to 2" long of tiny crystals which are never found at depth in the Deep Inlet veins and it undoubtedly has the appearance of secondary gold. It seems possible that highly localized solution and recrystallization has taken place within an extremely thin surface layer of the vein, where optimum conditions (the presence of ferrous sulphate and sulphuric acid for example) could result in the formation of these delicate wires. However, such solutions are highly unstable at the best so that any transfer of gold would be limited. This relation would thus be residual enrichment as the surrounding pyrite has been completely oxidized and rapidly removed, leaving the gold concentrate behind. The finding of tellurides underground would also suggest a primary deposition of most of the precious metal. Visible gold has been noticed in small quantities within the aplite dike away from oxidation and open fracturing alike. In fact, it is sometimes entirely surrounded by the dense greyish aplite unaccompanied by contiguous sulphide or quartz. This hardly indicates redeposition from cold surface waters lacking in penetrative power. Further evidence is contained in the fact that the greatest quantity of visible gold so far discovered lies 125' underground in close association with the tellurides. If this is indicative of secondary gold then it is to be hoped that there is more like it. Relative to the abundance of visible gold in the lamprophyre ore, the amount located in the aplite is practically negligible. A possible explanation lies in the magnesian content of the mafic dike since magnesia is known to be an effective gold precipitant. The quartz itself is locally ground into a rubble cemented by oxides and the pyrite sometimes displays slickensides. Such late movement is especially favorable to the still later primary deposition of gold and it renders the ore very friable.

ORE SHOOTS:

Five ore shoots have been partially outlined so far by drifting. Four of them occur in the sheared lamporphyre while the fifth lies wholly within the aplite. Taken together with the intervening pinches, they cover a horizontal range of 850' and a vertical range of 450'. However, there is a notable barren section of approximately 100' to perhaps 200' lying between the relatively low grade (marginal) aplite shoot and the nearest lamprophyre ore. For the above reasons the aplite shoot has been left out of the average value and tonnage calculations. It is nevertheless indicative of the total range of possible gold mineralization as established to date.

The shoots are separated by pinches containing 1" to 2" of quartz and sulphides which may be high grade or low grade depending upon the amount of oxides or sulphides associated with the quartz. At other places the pinches are composed of completely barren, sheared lamprophyre with a little soft gouge.

SHOOT CONTROL:

Local rolls and swells occur along the shoots and ore widths very rapidly in any direction because of the irregularity of shearing. It is still too early to hazard much of an opinion on ore shoot control but several possibilities have already been noted. Relatively flat-lying cross-dikes, of limited areal distribution, may have offered some trapping effect ... it is also possible that both these dikes and the shoots are concentrated in close association because both are the result of long-continued stresses within the same weak block. However, it is a fact that they do occur together when in any quantity. One of the cross-striking lamprophyres shows a displacement parallel to the strike of the vein and since it does not cut the ore-bearing dike the latter is apparently younger. Therefore it does not seem likely that these older dikes could have dammed the rising solutions.

Other structures may explain the localization of ore shoots but their importance is not yet clearly discernable. One of these is the presence of numerous branch or feather veins striking or dipping into the main zone, as in the case of the high grade east wall vein (almost vertical) of the top tunnel which approaches a junction with the main west wall vein (dipping at 75°). Another structure worthy of attention is the distinct change in attitude of the dike both horizontally and vertically. There is a marked roll between the lowest and the middle levels, and the lamprophyre becomes almost vertical near the face of the top tunnel (a 15° change in dip). This flattening at depth may be accompanied by a decided widening of ore shoots somewhere along the proposed extension of the lowest drift, since it is commonly found that such conditions appear to retain or slow down rising mineral solutions, thus allowing maximum penetration, cooling and deposition. Until all the structures, including cross-faults, are further explored and accurately mapped, their importance in shoot control will remain indefinite and of little value in the search for new ore shoot zones.

... SHOOT AVERAGES:

The shoot lengths so far exposed are 25', 45', 20', 107', and 33' and the average, or the total, may be readily calculated (230' of shoots in 610' of all development, or, eliminating the marginal shoot, 205' in 610'). However, these figures mean little or nothing for several reasons:

(1) Only 2 out of the 5 shoots have been truly elimited by mining. One of these (the aplite shoot, 25' long) is not considered to be a member of the lamprophyre ore zone and the other (33' long) is believed to be on the upper margin of the favorable zone.

(2) Of the oter 3 shoots, one (20' long) is open at the face and shows considerable strength.

(3) The last two (107' and 45' long) were drifted through at one end, but were terminated at their opposite (north) end by erosion since they are both strong at the portal.

(4) Not enough shoots have been drifted on to give a fair average (this holds true in any kind of sampling).

Thus, it is absurd to use the last 3 lengths in calculating a picture of the mine average. If mining has been confined mainly to the ... and front, or outside perimeter, of the ore shoot zone, the ore bodies so ... exposed should obviously be subnormal and the results of maximum penetration of that zone (face of the middle level) may be considered as representative of the trend to better lengths, widths and values.

Even if the true, average, horizontal length were known, there is no set rule, or law, which justifies the assumption that all the axes of any given shoot or shoots are of nearly equal length although it is possible they could be so. On the other hand, most ore shoots are irregular in outline and size, and although there may be a definite axial ratio for any one mine, or even an entire camp, that ratio is almost completely dependent upon structural control of deposition. Such control, through its inherent nature, must determine which axis shall be longest or shortest, as well as define the pitch and rake of the shoots. Therefore, until several raises or stopes have been completed, any assumptions of equal heights or length are only guesswork, and to attemp calculations of possible future tonnage based upon the above subnormal and limited data is wasted labor unless the results are employed to picture minimum prospects. Similar arguments may be advanced concerning the average width (9.1") and values. It would appear that most, if not all of the above averages are minimal and that further drifting on the two lowest levels will bring increases. *

SAMPLING PROCEDURE:

Three sets of channel samples have been taken by, or under the supervision of, competent field scouts, who have examined the Patmore mine during the past six months. This was done with a view to optioning a substantial interest for their principals. Two of the companies are amongst the largest in Canada and the third is an important mine operator in the United States. All have shown a decided interest in the property and two written offers emphasize the mine-making probablities.

The samples were cut both wide and deep (2"-6") so that they were essentielly bulk samples. This practice was followed in an effort to eliminate "erratics". A methodical 2½' spacing was chosen for the shoots and this was increased to 5' along the pinches. This left a mere 2' between the sample cuts and produced, in effect, almost a continuous strip sample. That such a procedure is essential and warranted on narrow high grade veins is demonstrated by the following:

(1) The results of the various sets agree fairly well. As one engineer said, "On this basis (cutting the highs) the assays show a reasonable check. The figures for each shoot check approximately and there is a very close check on the total".

(2) A large sample was roughly quartered in the field after very coarse crushing. The two opposite quarters were combined and sent in for assay. The other two were treated similarly as a planned check. The results showed a one counce (in gold) per ton difference thus stressing the need for large samples or for fine grinding before quartering.

(3) Although considerable coarse, visible gold is present in the vein, remarkable few erratics were obtained.

(4) Mill tests show that over 80% of the total gold is of the free-milling type.

Thus it seems reasoheble that the completed sampling is fairly representative of the ore so far exposed by drifting. However, the calculated weighted average does not limit the chances for a change and the grade of the deepest shoot would appear to indicate an improvement.

WEIGHT-WIDTH AVERAGE VALUES:

All the samples were assayed by G.S. Eldridge and Co. of Vancouver, and certified copies of the results were presented to the lease holder. The mill-feed figure below is based on a 3' stoping width (this may be improved as is discussed under stoping) with 50% rted to waste. That is, the milling width is 18". Since the ore is highly oxidized, 12 cu. ft. is taken to equal 1 ton. This would be reduced to 10 cu. ft. if the sulphides were unoxidized and present in the ratio of 3 parts to 1 of quartz. Following are the weighted averages of the lamprophyre-shoots. All assays over 5 oz. have been cut to that figure.

SHOOT	WIDTH	LENGTH	GRADE	MILL FEED GRADE-18"	VALUE AT $35 GOLD	TONNAGE VERT.FT.
No. 2 Tunnel (outer)	8.8"	45'	1.72	.84	$29.40	5.62
No. 2 Tunnel (inner)	11.4"	20'	2.55	1.62	56.70	2.50
No. 3 Tunnel (outer)	9.1"	107.5'	1.17	.59	20.65	13.43
No. 3 Tunnel (inner)	7.1"	33'	1.50	.59	20.65	4.12
TOTAL	9.1"	205.5'	1.74	.88	$30.77	25.7

still in ore shoot cut by surface

SHOOT	WIDTH	LENGTH	GRADE	MILL FEED GRADE-18"	VALUE AT $35 GOLD	TONNAGE VERT.FT.
No. 2 Tunnel (outer)	8.8"	45'	1.46	.71	$24.85	5.62
No. 2 Tunnel (inner)	11.1"	20'	2.57	1.58	55.30	2.50
No. 3 Tunnel (outer)	8.8"	107.5'	1.21	.59	20.65	13.43
No. 3 Tunnel (inner)	6.1"	33'	1.65	.56	19.60	4.12
TOTAL	8.7"	205.5'	1.72	.83	$29.09	25.7

Averaging the results of the two examining engineers' sampling:

WIDTH	LENGTH	GRADE	MILL FEED GRADE-18"	VALUE AT $35 GOLD
8.9"	205.5'	1.73	.86	$29.93

A composite assay for silver showed approximately 1 oz. per ton therefore total values are about $30.40.

Since the third set of samples was much more limited in scope because of its preliminary nature, it has not been presented in the same manner. The figures may be tabulated as below to give an idea of the gold distribution. The samples were taken <u>at irregularly spaced points</u> thus making it useless to present them as an average:

NO. 2 TUNNEL

SAMPLE #	PLACE	WIDTH	GRADE
1922	Outer shoot (portal)	4"	1.07 oz.
1823	"	5"	3.62
1924	"	9"	1.48
1925	"	4"	.94
1926	"	6"	.96
1927	"	5"	.78
1928	"	3"	1.92
1929	"	5"	4.06
1935	Inner shoot	3"	.58
1936	"	3"	.49
1937	"	4"	1.30
1938	"	5"	4.05
1939	"	3"	.34
1940	"	5.5"	5.03
1956	"	6.5"	1.31
1957	" (face)	32"	.36

NO. 3 TUNNEL

SAMPLE #	PLACE	WIDTH	GRADE
[illegible]1	Outer shoot (portal)	7"	.68
1942	"	2"	3.10
1943	"	11"	5.94
1944	"	11"	1.35
1945	"	8"	.24
1946	"	9"	1.82
1947	"	13"	1.61
1948	"	10"	3.58
1949	Inner shoot	9"	4.72
1950	"	3"	2.28
1951	"	4"	8.52
1952	"	4"	4.49
1953	"	3"	1.14
1954	"	3"	4.22
1955	" (near face)	5"	.63

AVERAGES

TUNNEL	SHOOT	WIDTH	GRADE
2	Outer	5.1"	1.85oz.
2	Inner	7.7"	1.68
[illegible]	Outer	8.8"	2.29
3	Inner	4.4"	3.71

Note: The widths are narrow here because the full width of the vein was not sampled during the preliminary inspection - at a later date the "stri pped" vein was broken through for complete sampling.

The results of sampling the "pinches" are not shown since they would be of little value. It suffices to say that unless oxides or sulphides are present in the narrow sections they are barren of gold. White quartz alone carries very small amounts of the precious metals although here and there, such quartz may show xoarse visible gold if the quartz accompanied by oxides.

SUMMARY OF VALUES:

Thus, it seems that an average millhead value of at least $30.00 will result if 50% of the broken tonnage is sorted to waste either on a picking belt or by careful use of resuing methods. In the latter case it should be possible to keep the values above that figure by holding the stope widths to approximately 30" and taking out cleaner ore before or after the waste. Further, if the present improvement continues (No. 2 tunnel of higher value than No. 3 and its inner shoot better than the outer one), the millheads might easily reach $35.00 per ton.

MINING CONDITIONS AND COSTS:

The prevailing conditions are readily apparent and the resulting costs may be closely calculated by reason of the 610' of drifts and cross-cuts already completed on 3 levels. A summary indicates that low-cost features are prominent and mining expenses should be appreciably less than those met with in Zeballos.

(1) The veins dip from 65° to 90° to the east, averaging about 75°. This fact insures gravity feed (free running ore) and therefore low handling charges in stopes. It also adds to the ease of working where stopes are narrow (more hammer room) thereby permitting a maximum decrease in mining widths (unhindered limit 2') in order to hold up the grade of ore. Further, narrow stopes require less support by timber or fill. Incidentally, in such narrow openings the direct cost of mining by machine is often greater than by hand although the speed (3:1) of maching cutting results in lower overall costs where overhead and management expenses become appreciable.

(2) The vein-bearing dike has walls of solid granodiorite which seldome show signs of slabbing thus rarely needing any timber except for a few stulls. Due to this, the extra work of framing and installing timber is negligible aside from building manways and chutes.

(3) Because of the persistent shearing (which should remain with depth) and the high proportion of oxidation (which should drop off as the limit of penetration by water and oxygen is approached), mining has been very cheap and easy on all three levels. Note that the adits cover a vertical range of about 300' and vary in length from 125' to 225' through aplite, lamprophyre (with and without ore) and granodiorite. The deep oxidation, still strong at the faces, results from a combination of four factors:

(a) Much of the shearing extended throughout the entire period of mineralization (as is evidenced by "slickensided" pyrite and "rubble- quartz") thus offering easy penetration to overlying solutions.

(b) The surface tracing of the shear has long been a natural water trough.

(c) The large water supply is very erratic (rapid run-off) thereby introducing the moisture variation which is most conducive to oxidation processes.

(d) The steep creek draw was protected by its adjoining granitic ridges from deep glacial erosion.

Therefore, at least one, if not both, of the above features should continue to considerable depth and consistent low-cost mining should follow.

The average rate of all the hand contracts completed so far (involving some dense, tough aplite and a small amount of granodiorite) approximates $7 per foot including food, powder, fuse and caps. This figure does not cover the cheaper contracts where the leaser participated in the actual work. Rapid progress was made (2' or more per man-shift) and the miners cleared over $12 per day in wages at all times. One section was driven 100' in 10 days by 3 men using less than 2 boxes of 40% powder (3' per man-shift). According to government statistics, 1' per man-shift (direct cost - $5 per ft.) is an average footage. Thus, the costs could have been still further lowered by some sacrifice in speed. About 75% of the holes were driven in rapid time by using a point-bar and it was seldom necessary to drill more than 3 or 4 holes to a 3' or4' round (4' x 6'). Locally the dike may narrow and then the much tougher granodiorite must be sideswiped. A jackhammer or light drifter should be on hand to speed up this more difficult work.

(4) As may be seen from the above figures, the intense shearing and osidation also result in low powder consuption. The mine records show an average of 12' to 15' per box of powder, mainly 40% gelatin, for 600' (a powder-cost of 45 cents per ft. of drift). In most cases 3 holes were placed vertically one beneath the other and this sufficed to cause a collapse of the entire dike where it was less than 3' to 4' wide. Overbreak of 1' was common.

(5) For the same reasons a large part of the muck was finely broken and therefore easy to shovel. The leaser, with the help of one miner, average 4' per shift for 100' although this meant mucking (with plates) 8 tons or about 50 barrows and wheeling it from 100' to 200' (using smoothly-rolling rubber-tired wheelbarrows but no planks). These conveyors are efficient up to 300' and can be used in relays up to 800' underground but it is probably cheape to purchase track and a car for any tramming over 400'.

As a whole, there is not a single mine in the Zeballos district which can boast similar conditions or results, so the assumption of equally low-cost stoping appears justifiable. Mining charges at the Privateer approximated $4 per ton 1939-40 (these increased during the war). At the Central Zeballos $5.92 was recorded for 1942, a war year. Although post-war costs (mainly labor) may not recede to their pre-war level, it is reasonable to assume some reduction. Knowing that stoping conditions (including poser and transportatio factors) are better at Deep Inlet, it is, therefore, likely that the overall s stoping and development (drifts, raises, and orosscuts) expenses will be under $5 per ton after the effects of the war are readjusted.

To gain some idea of development costs per ton of pure ore for the average Vancouver Island vein of 6" to 8" width, it is necessary to consider the drifting cost per foot. Assuming $6 per ft. (labor, food, powder, use and caps), a reasonable rate as far as the Deep Inlet ore zone is concerned, and an 18" width of mill feed (8" ore and 10" waste), the direct development cost per ton for the drifting (in ore) item alone on 100' lifts would be 48 cents (since 1' drift would develop 12½ tons of millfeed 18" wide for the 100' between two levels). To this must be added other development charges for drift in waste, for crosscuts (with their attendant higher cost ratio) and for raises. Locally, in mining narrow veins it is common to have 2/3 of the total underground footage in waste (very little ore is opened up by much development). Thus, the total direct development charges should be at least 3 times 48 cents or probably slightly over $1.50 per ton of millfeed before stoping. Moreover, it is likely that contract rates can be cut to $5 per ft. since the man-shift average at Deep Inlet lies between 2' and 3' of drift, which affords each miner-shift a liberal $10 to $15 (less $2 for food and supplies). These figures for contract labor are substantiated by the general average of $5-$6 per ft. recorded by the U.S. Bureau of Mines for a large number of similar soft (sheared and oxidized) deposits in Oregon (e.g. Record Mine were development costs were $1.30 and square set stoping $1 per ton). Note that although the Central Zeballos ore is highly sheared, neither it nor any of the other neighboring mines displays the persistent and almost complete oxidation found at Deep Inlet and in parts of Oregon (n.E.). This lower contract price should effect a further decrease in development expense to about $1.50 per ton of millfeed before actual stoping costs (or higher figure per ton of pure ore).

To arrive at the expected total mining cost to be applied against each ton of millfeed, the cost of breaking both ore (a maximum 18" milling width or less if economical) and waste (a maximum of 18" or less if feasible), whether separately or together, must be estimated and then apportioned to the millfeed alone (after sorting). One ton of drift-mined millfeed (18") would cost about $6.20 since .81 tons would result from 1' of 6½' x 4' drift at $5 per ft. but shrinkage-stoped millfeed should not exceed 1/3 of this amount or $2.20 per ton aside from overhead. (At the Central Zeballos, where low-cost shrinkage stopes were used, the figure was $2.38 including approximately 30 cents overhead). Using the resuing method (or its converse) on a similar soft, narrow vein, the stoping charges would be considerably higher but so too would be the mill-heads (providing the ore and the waste would separate or could be sorted to a certain economical limit). The final choice of stoping system will be determined by the practice of giving the maximum profit. Therefore, for purposes of calculating that possible profit, it will be logical to assume 100% use of the Shrinkage method if it is also assumed that millheads will not be higher than the $30 average obtained by sampling to date (stoping 3' and sorting to 18").

Thus, with the above estimated direct costs of about $1.50 and $2.00 per ton for development and shrinkage stoping respectively, the total mining charges, allowing 60 cents per ton overhead as an average figure, on a 50-ton basis, for supervision, engineering, assaying, bookkeeping and social security, would approximate $4.10, to be assigned to each ton of ore reaching the fine ore bin. With resuing (stripping) or its converse, the total might reach $5 but with resultant higher mill heads and an equal if not better profit.

STOPE MINING:

In narrow, steep deposits of a highly sheared nature, such as the main Deep Inlet vein, hand drilling is as cheap as, if not more economical, than machine work insofar as stoping is concerned. This is because narrower stopes may be carried by hand, thus using less powder (more control) and negligible power to deliver a cleaner product at a disproportionate loss in speed. This is especially true in the abnormally soft Deep Inlet ore, However, light stopers or jackhammers will be necessary locally where the shearing diminishes or the dike pinches to less than 32". The fastest cuts are vertical water holes or steep, dry, 2'-3' uppers from which cuttings may fall freely, but where the vein material is under the average width or the waste is too weak to stand heavy powder charges, it may be more economical to use horizontal cuts since they allow greater control of breaking. Where drilling is inescapable, "singlejacking" is preferable to doublejacking as the increase in speed is not proportional to the extra labor. In such soft ores actual drilling is seldom needed because a "bull-prick" or point-drill and at times a hand-auger, will suffice for 75% of the holes. This Point-drill work requires a doublejack (14# to 16#) and the steel is best withdrawn by means of a collar and hand winch when the ore is very "gummy". In similar ore at the Sioux Consolidated mine, Utah, one man was able to break 25 tons per shift. Augers were used almost exclusively in the stopes of the Gleason mine, Oregon, where parallel conditions existed.

Since the Deep Inlet ore is so easily drilled, it should be quite feasible to maintain 2½' stope widths except for local sections intensely sheared across more than 32". Although cases of stoping to maximum widthe of 18" are raported for Mexico and the States, unhindered hand mining is limited to a minimum of 24" and common practice tends towards 32" for drilling by hand and 36" for machine work. The important factor appears to be the softness of the ore and the ease of putting in a round. Where the shearing is greater than 32" excessive dilution may sometimes occur if powder charges are not kept at a minimum, holes drilled horizontally, and a few hitched stulls placed with headboards against the sloughing area. Central Zeballos experienced this trouble because of "blocky" ground (3' stopes). Stope faces will probably be maintained as horizontal or slightly inclined 3'-4' steps with holes placed so that they may break to either of two free faces. If it seems advisable tu use cut and fill methods it is quite possible that muck-handling charges may be lowered by instituting rilled stopes.

STOPING METHODS:

Because of the appreciable differences in cost per ton of millfeed resulting from the application of the numerous stoping systems, the selection of one or the other of them is of highest importance in the economical exploitation of such narrow veins. Since the main Deep Inlet vein is steeply dipping as well as narrow, the choice is fairly well restricted to shrinkage stopes (the cheapest), open stulled or square-set stopes and variations of the cut and fill type (resuing, its converse or full width breaking followed by underground sorting). Open-stulled and underhand systems are not favoured due to local weakness of the back and walls where the dike is blocky or widely sheared. Shrinkage stopes may be used only on the strong-walled wider shoots (since sloughing is pronounced where walls are sheared and broken ore tends to hang up dangerously in irregular narrow openings). At certain mines in Oregon where the back was weak and natural timber was as readily and cheaply procured as it is at Deep Inlet, the square set-open system has proven to be most efficient (e.g. Record mine - stoping costs $1 per ton. There appears to be little doubt that more than one system will be employed at the Patmore mine due to local variations in wall rock character and the amount of shearing.

At least two shoots already drifted on display features that favor cut and fill methods, one of the shoots being firm yet unfrozen and therefore suitable for resuing or "stripping"; the other, soft and free and so best removed ahead of the waste. This choice depends entirely upon which fraction can be stoped cleaner and easier thus leaving the firmer one standing. In either case there must be a free or well-defined junction between ore and waste and it is the degree of freedom which determine the economic limit to which cut and fill may be employed (i.e. the economic and possible limit of underground sorting of intimately mixed ore and waste). Although the Deep Inlet ore breaks fine and certain shoots display ragged margins between ore and sheared dike, extensive sorting of the 1" material is possible and warranted due to the ease of separating the highly contrasting green waste from the red oxide and white quartz of the ore. Where the two are so intimately mixed that they must be broken together and cannot be separated by sorting, the green dike usually carries sufficient values along fracture planes to rank as millfeed but the product tends to be marginal and may not withstand the high cost cut and fill system where the vein is wide enough to create undue handling of the ore (since waste is not mucked in cut and fill the controlling factor is the proportion of waste to ore). The only alternatives for greater widths are to use shrinkage methods in which lower handling costs should increase profits (practically no timbering, no mucking, greater speed and easier daylight sorting after washing but more tramming) or rilled stopes. Because shrinkage is not feasible where the back or walls are too weak to stand or the walls or shoots are too irregular, the latter may offer the higher profit.

In summation, the limited development so far carried out at the Patmore mine indicates that the width and extent of shearing and of ore as well as the degree of possible economical sorting will determine local use of shrinkage or cut and fill but that both systems will be employed in shear zone characteristics favoring the latter. It may be justifiable to institute rilled stopes (gravity-clearing platforms) where the proportion of ore is high rather than shrink with attendant high sloughing and consequent excess dilution.

MILL TESTS:

Two lots (100# each approximately) of average type millfeed were sent out to the Denver Equipment Co. of Colorado and to G.E. Eldridge and Co. of Vancouver. The ore sent to the Denver Testing laboratory was of average type but was not a true weighted average. That sent to Eldridge and Co. was a weight-width average although it included the more heavily-pyritized No. 1 tunnel shoot which is too marginal in grade ($14 to $16) to be milled. Even at that, the millhead average was 1.48 oz. (using a shoot width of about 8.8" with very little waste involved). This figure amounts to approximately $3.85 per ton more than the calculated sample average. Therefore, the mill tests show that at least $31 to $34 may be expected over an 18" width. Since the mill recovery will be over 95%, the gross returns should amount to a minimum of $30 and perhaps even as much as $32. It should be noted that the recovery in a laboratory test is nearly always exceeded by that in a full-size mill run once the operation is proceeding smoothly.

Following is Mr. Eldgridge's summary of results:

"As this ore appears to be very friable and easily crushed, the indicated flow sheet would be a gyratory crusher reducing to ¾", then to rolls reducing to ¼", thence to a jig taking out the free gold in the hutch concentrate which would go direct to the amalgam barrel thus doing away with the usual troublesome tie-up of a large proportion of the gold in the ball mill circuit.

" The overflow from the jig being ground to ¼" will give a better feed to the ball mill, giving it a greater capacity than a coarse crusher feed, and will save considerable wear on ball mill liners and grinding medium. The overflow from the ball mill goes to a drag classifier, from which the fines overflow to the conditioner and then to six flotation cells, and the oversize sands are returned to the ball mill for regrind. The concentrates from the first two cells are sent to the vacuum filter, and the froth from cells Nos. 4, 5 and 6 are returned to the No. 3 cell for recleaning. The concentrate from No. 3 would probably be rich enough to be added to the concentrate from Nos. 1 and 2, and all filtered and shipped together. The flotation tailings would go to a Wilfley Table, and the iron oxide concentrate from this could be either mixed with the flotation concentrates for shipment, or be sent to the amalgam barrel to remove any free gold by finer grinding.

"The tailings from the amalgam barrel should be put over a small Wilfley Table to recover any amalgam or floured mercury in the form of a concentrate, and the table tailings, which would be small in amount, sent to the flotation conditioner to recover any sulphides, fine free gold or amalgam that might get over the cleaning table.

"Test No. 2 indicates that with this oxidized ore, using the above flow sheet, a recovery of Ninety-five per cent (95%) of the gold can be obtained. About 65% of this would be by amalgamation, 21.5% by a flotation concentrate running 8.1 ozs. per ton, and 8.5% by a table concentrate on the tailings running about 1.5 ozs. per ton. This latter may be re-treated in the mill, as shown above, or re-cleaned and shipped with the flotation concentrate.

"The ratio of concentration by flotation is:
Twenty-four (24) tons of ore to one (1) of concentrete.

"The much higher amalgamation recovery (82%) in Test 1 where all the ore was ground with the mercury indicates that in Test 2 condiderable gold was carried over in the Jig tails possibly due to too fast a rate of feed to the Jig. We consider that by means of a very much slower feed to the Jig a higher rate of recovery by amalgamation would be obtained".

The above may be compared with the Denver conclusions given below:

"Following is a resume of the results of each test:

"Test No. 1

Denver Mineral Jig followed by Blanket Table concentration and Amalgamation of the resulting concentrates.

This test produced a Denver Jig Concentrate which assayed 147.14 oz per ton gold and 16.12 oz per ton silver, and which contained 78.35% of the total gold and 37.65% of the total silver. Blanket Table treatment of the Denver Jig tailing described on Page D-3 produced a concentrate which assayed 9.95 Oz Au/ton and 1.93 Oz Ag/ton.

"ratio of concentration by Denver Jig was 27.3 to 1 and by blanket table was 18.2 to 1. The combined ratio of concentration was 10.93 to 1.

"Amalgamation of the Denver Jig Concentrate produced amalgam which containe 75.45% of the total gold and 34.45% of the total silver.

Amalgamation of the blanket table concentrate produced amalgam which contained 4.7% of the total gold and 2.55% of the total silver.

TEST NO. 2

Denver Mineral Jig followed by gravity table concentration and amalgamation of the resulting concentrates.

This test produced a Denver Jig Concentrate with ratio of concentration of 20.6 to 1 which assayed 99.9 oz Au/ton and 10.7 oz Ag/ton and which contained 77.19% of the total gold in the original head ore. Table concentration of the jig tailing produced a concentrate with ratio of concentratio of 28.6 to 1 which assayed 17.08 oz Au/ton and 2.0 Oz Ag/ton and which contained 9.52% of the total gold. Amalgamation of the Denver Jig Concentra produced amalgam containing 73.72% of the total gold in the original head ore. Amalgamation of the table concentrate produced amalgam containing 6.84% of the tdal gold.

"REMARKS AND CONCLUSIONS:
The tests reported indicate that over 80.0% of the gold in ore represented by the sample tested is recoverable in bullion form by treatment with the Denver Mine al Jig and Gravity Table concentration followed by barrel amalgamation of the resulting concentrates.

The grinding time used in the tests compared to that of our standard grind test ores indicates that the ore tested should be classed as being between "Medium" and "Medium-hard" ore to grind.

"The flowsheet recommended for the treatment of this ore to secure maximum gold recovery in bullion form should include the following equipment:

(1) Coarse Ore Bin
(2) Grizzly
(3) Denver Jaw Crusher
(4) Fine Ore Bin
(5) Denver Belt Ore Feeder
(6) Denver Ball Mill
(7) Denver Mineral Jig
(8) Denver Cross-Flow Spiral Classifier
(9) Denver Wilfley Concentration Table
(10) Denver Amalgam Barrel
(11) Denver Mercury Separator
(12) Amalgam Retort

The tests indicate that the concentrating table is preferable to blanket table concentration. However, it should be remarked that a blanket table test on a batch basis is not comparable to the continuous operation of a blanket table in practice. The length of table used in the test was much shorter than would be recommended in practice and the amount of concentrate remaining on the blanket as compared to the quantity of batch used was much greater than would be held by the blanket after a few hours operation.

For this reason it is recommended that the above listed equipment be installed and larger scale blanket tests be conducted when the mill is operating continuously. Such tests would show the ...

stalling a full size blanket table on the tailings.

"Because of the appreciable quantity of pyrite in the ore, maximum recovery will be secured by making a ratio of concentration of about 12 to 1. However, the results of the milling operation may show that the ratio of concentration can be raised to 20 or 25 to 1, by carrying heavier bedding in the Denver Jig, and by cutting higher grade table concentrate without too great a loss of amalgamable gold.

"The screen analysis of the tailing from test No. 1 indicates that the tailing losses occur heaviest in the minus 200 mesh portion and that considerable additional recovery of gold may be expected when flotation is installed".

MILLING COSTS:

These costs are more difficult to determine but the results of the above mill tests, used in conjunction with figures on actual practice employing a similar flowsheet on an equal tonnage, will give an approximation. The Central Zeballos mill could concentrate Deep Inlet ore quite efficiently with very minor changes. Furthermore, although this Zeballos ore is unoxidized, it is partially sheared somewhat like that of the Patmore mine and presents no major difference in metallurgical conditions. The plant was designed for 35 tons per day but relatively few alterations have allowed an increase to 50 tons per day. Since the Deep Inlet ore shoots are at least the equal (in tons per vertical foot as well as values) of those developed at the Central Zeballos and other neighboring mines, a minimum rate of 50 tons daily has been chosen as a base for discussion. Of six Zeballos properties, only one milled at less than this figure.

Following is a comparison of vital cost factors:

(1) Power supply - Fuel costs 8.6 cents per gal. at the beach where the proposed Deep Inlet diesel-electric plant will be located and Central Zeballos pays 11.1 cents at its mine plant. One year's fuel bill would amount to between $12,000 and $16,000 since 80,000 to 100,000 gals. of oil would be needed to develop the required 200 to 250 H.P. (185 kw.) The mill alone is driven by a 75 kva generator (about 100 H.P.) so that it would consume about 2/5 of the fuel or $4,700 to $6,400 worth. Mill-power costs about 63 cents per ton milled or ¼ of the total milling expense. By comparison it should not exceed 46 cents at Deep Inlet. Note that three secondary factors enter into this figure. One is the cost of hauling in the initial machinery and replacemtns to the Central Zeballos property (14 miles of freighting, including the return trip). A second is the already completed power line right-of-way and towers at Deep Inlet, and a third is the ease of crushing and grinding the ore at the latter place. If a hydro-electric plant were constructed there would be an appreciable further saving as discussed earlier (see Power).

(2) Maintenance and operation of crushing-grinding-classifying circuit - The mine-proven and lab-tested friability of the ore so far developed at the Patmore property is certain to result in low repair and replacement expenses (especially for ball charges and linings). Sorting, crushing and grinding costs exceed $1 per ton at Central Zeballos.

(3) Bullion Recovery - At Deep Inlet this will amount to over 80 (possibly 85% when continuous treatment has been sufficiently tested and varied). Central Zeballos recovers 43.26%, Privateer removes 64% (without cyaniding) and Mount Zeballos averages close to 55%. Thus, the Deep Inlet ore may be classed as typically "free-milling" and this fact should aid in [illegible]ting costs by means of decreasing grinding time and increasing tonnage per H.P. utilized, as well as lowering flotation expense (no oversliming of fine gold).

(4) Ratio of Concentration - Central Zeballos has a ratio of 20.65 (tons of ore) to 1 (ton of concentrates). G.S. Eldridge and Co. indicate a ratio of 25:1 for the Patmore ore. This means that there should be no more than two tons of concentrates per day to be hauled 2000' to the tidewater and shipped to Tacoma (60 tons per month at $3 to $4 per ton). Central Zeballos must truck 2 1/5 tons per day for a distance of 7 miles and for this reason their concentrate expense runs over 20 cents per ton milled while marketing exceeds $1.

(5) Mill and Mill-camp Supply - To a certain extent the cost of chemicals and of food supply will be lower at Deep Inlet due to the short haul.

(6) Labor - This item, which is the major single factor in cost of milling, amounted to $1.02 per ton at the Central Zeballos mill during 1940 (March), a war year. It seems possible that even the first post-war year should see some reduction in this cost. It is a least certain that labor charges will not be higher.

A review of total milling cost factors would appear to indicate that the Deep Inlet ore may be concentrated at a slightly lower rate than that attained at the Central Zeballos plant ($2.681). The power factor alone should cut this to 2.51 and a combination of more favourable milling characteristics and cheap transportation will almost certainly lower the total to a maximum of $2.25 per ton milled. Unfortunately, no comparison may be made with the results at the Privateer mine as more costly cyanidation is employed there.

OTHER EXPENSES:

(1) Tramming - Initially, the tramline may be 2320' long but after driving the proposed crosscut it would not be over 1230' and possibly only 1060'. Tramming 2400' by aerial jigback at the rate of 66.3 tons daily, costs about 47 cents per ton (milled) at the Central Zeballos mine. Therefore, after completion of the long crosscut, the cost at Deep Inlet should be somewhat less than 40 cents per ton.

(2) Tailings Disposal - This cost will be negligible because ther is ample disposal room with sort transportation of the tails required (gravity flow).

(3) Marketing - Aside from the lower tonnage of concentrates to be shipped and the shorter truck haul, marketing costs at Deep Inlet should be comparable with those at Central Zeballos ($1.03) or say 75 cents per ton milled. If mining proves up sufficient ore reserves and tests are effective, it may possibly be profitable to cyanide the concentrates (due partly to the small daily tonnage) and thus eliminate most of the marketing expense. Naturally, the cost of cyaniding (per ton of ore milled), allowing for a retirement fund on such costly equipment (10%), interest on the

investment, (5%) for operation and for chemicals, could not exceed the marketing cost of say 75 cents per ton if it were to be profitable.

(4) Plant Overhead - (Depreciation, extra staff) - By way comparison, the only advantage would be that of transportation of repair and replacement parts. Office and staff charges would be held at a minimum by maintaining an office and staff only at the mine. Directors and consultants (insofar as fees are concerned) would be unnecessary. Thus, the over-head should not amount to more than 75 cents per ton milled (87 cents at Central).

(5) Royalty Charge - At the two year rate of 6% the royalt cost would amount to $1.80 per ton of $30 ore milled.

(6) Insurance - This expense could be held to about 15 cen /ton.

(7) Mineral Tax - At 2% this would amount to approximately cents per ton milled.

(8) Amortization - 15% yearly on a proposed total developm and equipment cost of $100,000 would equal $15,000 annually (18,000 tons) or 83 cents per ton milled.

SUMMARY OF TOTAL EXPECTED COSTS:

1.	Stoping	$2.00
2.	Development	1.50
3.	Mine overhead (assaying, book-keeping, engineering, superivision, social security, and maintenance	.60
4.	Milling	2.25
5.	Tramming	.40
6.	Marketing	.75
7.	Mill and power plant overhead (excluding mine)	.50
8.	Royalty charge	1.80
9.	Insurance	.15
10.	Amortization	.83
11.	General	.22
	TOTAL	$11.00

FINAL PROFIT:

Before taxes the profit would be $19 (assuming $30 recovery per ton). The depletion allowance (at 33 1/3%) amounts to $6.33 per ton thus leaving $12.66 per ton taxable at the present excessively high war-time rate of 40%. It should be stressed that such a rate cannot possibly persist after the war has ceased if mining is to survive as a major Canadian industry. However, using this figure (40%) the tax will amount to $5.06 per ton milled. The mineral tax at 2% would be about 38 cents thus making a combined tax of $5.44. Therefore, the final profit to be made on ore of the present average grade could be expected to be better than $13.56 per ton of ore milled (at the rate of 50 tons per day). Of course, this figure depends upon the assumption of efficient operation and field management. Montly earnings at the proposed tonnage could reach a minimum of $20,340 or about $240,000 annually. On the basis of a 40-60 division of profits this would give the minority holder at least $96,000 yearly.

[illegible]

As explained previously, the present equipment includes everything necessary to complete the <u>secondary development</u> from the 10-ton to the 25- or 50-ton stage. The machinery and tools have all been well looked after and they are essential to the work. If they were not already on the property, any company taking over the Patmore lease would be forced to acquire a similar outfit. Its value at the beach and mine camps exceeds $5,000. Following is a nearly complet list:

1.	Main tram hoist and gas engine (used)	$500.00
2.	Auxiliary tram hoist and gas engine (new)	400.00
3.	Assay office crusher (Chipmunk - new)	
4.	Assay office pulverizer (Braun - new)..............	400.00
5.	Assay office power plant (gase engine - new).......	70.00
6.	Two nearly new Beebee winches	230.00
7.	Cable - various sizes (mostly new)	1000.00
8.	Boat - 14.5' - and 3.4 H.P. engine (used)	400.00
9.	Boat - 12' - and 1 H.P. engine (used)	100.00
10.	Cables saddles, bolts etc. (new)	150.00
11.	Carriage wheels - 2 sets (new)	50.00
12.	3 rubber-tired barrows (2 used)	75.00
13.	At least 400' of mining steel	60.00
14.	150' of galv. water pipe and connections	25.00
15.	Two anvils and forges	50.00
16.	Picks, shovels, scrapers, hammers and mucking plates for 3 faces	50.00
17.	Blacksmith's tools	25.00
18.	Carpenter's tools and coal	100.00
19.	6 Crosscut saws and wedges	60.00
20.	12 kegs of nails	75.00
21.	2 entire camp cooking outfits	250.00
22.	2 entire camp sleeping outfits	50.00
23.	Heater 1...	10.00
24.	8 cases powder, fuse and caps	70.00
25.	Boom chains	20.00
26.	Eye bolts (split with wedges)	16.00
27.	25 iron sheets	40.00
28.	Lumber (new)	120.00
29.	300 drift bolts	50.00
30.	6 rolls of roofing paper	20.00
31.	8 spare windows (lights)	10.00
32.	Drag saw (used)	60.00
33.	Circular saw blade and mandrel	60.00
34.	Food supplies (good shape)	150.00
35.	Machinist's tools	54.00
36.	Other camp equipment (axes, ropes, 100 ore sacks, carbide, canvas etc.)	1200.00
		$5000.00

It should be noted that much of this equipment has already been <u>packed</u> up the mountain thus its value is enhanced since most of it is very heavy material (back-packing -$300). In 6 months the leaser packed 6 tons of equipment and supplies from the beach to the mine camp.

DEVELOPMENT COMPLETED TO DATE (INITIAL):

The initial development included the following items with an approximate value of $50,000 (embracing cost of labor, supplies, compensation, maintenance, supervision, engineering, and interest on investment)

1. 610' of crosscut and drift (4' x 6.5').
2. Blacksmith shops and powder houses.
3. Beach camp (2 cabins).
4. Mine camp (2 cabins).
5. 150' x 40' float and freight shed.
6. 4500' tramline and millsite-clearing.
7. 1000' pipeline (lower camp).

Value of this construction may be computed as follows:

1.	Leaser's salary (3 yrs. at minimum engineering figure	$10,800
2.	Bookkeeper-cook's wages (3 yrs.)	3,600
3.	Leaser's helper's wages ($1\frac{1}{2}$ yrs. with overtime	3,780
4.	Wages of two working partners (6months)	2,520
5.	Leaser's cash investment	1,000
6.	Other partners' cash investments	13,950
7.	Cash loan	1,650
8.	Value of original camping equipment used to start this work	400
9.	Value of materials in float and shed	4,000
10.	Value of materials in tramline	9,000
11.	Value of materials in 4 cabins	1,000
12.	Interest on the above labor-cash investment at 5% (approx.)	2,400
	TOTAL investment and value	$54,100
	Present equipment and supplies	5,000
	TOTAL COST OF INITIAL DEVELOPMENT	$49,100
	ACTUAL CASH SPENT TO DATE	$16,700

SECONDARY DEVELOPMENT:

The present equipment is such that with very few repairs and replacements (such as hammer handles) underground work can be initiated a soon as the temporary tramline and camps are completed. The 800' to 1000' of drift necessary to prove up enough ore for a 25- or 50-ton mill will cost about $10,000. It may be driven by hand steel and the muck may be removed by wheelbarrow (rubber-tired). The lowest level requires another 500' to 600' of drifting in order to explore the vertical continuity of the zone of shoots exposed in the two upper levels. The face will then be about 800' from the portal or about the economic limit of relay wheelbarrow tramming. It might pay to build a lightweight rubber-tired dump truck to avoid the expense of bringing in rails and a mine car until the larger tonnage is proven. The middle level should not need over 300' to 400' of secondary development drift - its length would then be 425' to 525'. The cost of tramming will increase the total cost per foot as each consecutive 100' of advance is attained. It will require an extra trammer for each 200' of advance if wheelbarrow are used.

The necessary tramline construction may be completed for out $2000 (3 towers to be built, the saddles bolted down, and cables chored and strung up). Four men could finish the work in two to three nths. One of the towers (the highest) is already half erected and the her two are short break-over and terminal structures (one of them being trestle or rail section). The cables are light and can be hauled into sition easily by means of a rope and winch. The mainline hoist has not et been taken up to the terminal site. Cables, saddles, eye bolts, carriage wheels and gas hoist have already been purchased.

The buildings needed are few. A 14' x 16' cabin and a tram erminal freight shed should be constructed at the beach camp ($400). At he mine camp a new bunkhouse (16' x 16') and a woodshed (12' x 14') are ssential ($520). A pipe-line has been planned to supply the upper camp ith running water ($80 installed). A small, storage powder house ($200) shoul also be erected near the mine before any more powder is brought in rom the beach. Construction of a mine storage ore bin will require about $800.

Thus, the total secondary expense should be about as follows:

(1)	Completion of temporary tramline for use in hauling camp supplies and powder	$2,000
(2)	Completion of camps (4 small buildings and water system)................	1,000
(3)	Driving of between 800' and 1000' of prospect-size drift by hand (2 levels)..............	10,000
(4)	Workmen's compensation charges	1,000
(5)	Construction of powder house and ore bin at No. 2 level portal	1,000
	TOTAL	$15,000

Note: These estimates include the cost of supplies (food, powder, fuse, caps, carbide, coal, nails, roofing, pipe, etc.).

It should be emphasized that the $10,000 for drifting will not be needed if the mine is opened up by the present leaser as a 10-ton proposition, However, the tramline, camp and ore bin expenses ($4000) would have to be met in either case. All of the above cost estimates are based upon experience at this camp and therfore should be fairly accurate.

FINAL PRE/MILLING DEVELOPMENT:

Providing that sufficient ore is opened up or is indicated by the proposed secondary development (800' to 1000' of drift) it is then planned to conclude the pre-milling development as follows:

(1)	Drive a 1000' to 1200' haulage crosscut to gain 400' more backs (see Proposed Crosscut)	$15,000
(2)	Complete the road (2000') and wharf program with govt. aid (as already outlined)	2,500
(3)	Buy and install a 25- or 50-ton mill of simple design (with camp alongside)	30,000
(4)	Convert the temporary tramline into a short permanent line with ore bins (as already discussed	6,000
(5)	Buy and install a diesel-electric plant with [illegible] power-line (see Power)	15,000

(6) Purchase mine cars, locomotive, compressor (but not power), steel, rails, drifters, stopers, etc. 12,000
(7) Drive a 400' raise from the crosscut to the present lowest drift 4,000
(8) Open up initial stopes 3,000

TOTAL (Expense of final development). $85,000

EXPENSE TOTAL including secondary development $100,000

COST OF MILL (INSTALLED):

The $30,000 estimate is based upon the complete cost (up to actual operation) of the almost exactly similar mill now being used at the Central Zeballos mine. This plant, with all electrical equipment, and entirely new except for the ball mill and crusher, cost slightly less than $25,000 including the labor, supplies and equipment up to the first day's run. It was designed for 35 tons but is now able to treat about 51 tons per 24-hour day. With the much more favorable location at Deep Inlet (2000' from the beach) it seems quite likely that the $30,000 figure may be decreased even if the ball mill is purchased new. Of course the cost will be appreciably lower if the proven ore is only sufficient to allow operation of a 25-ton mill.

AMORTIZATION:

The above estimated total development and mill cost expenditure of $100,000 could be retired at 15% (maximum government allowance) over a period of 7 years. Naturally, if a 25-ton plant is planned, the total figure would not exceed $70,000 and this amount could be retired in 5 years. Any company taking over the present lease will be allowed the return of its development expense at the 15% rate - as an expense item to be withdrawn before division of net profits. Any cash payments in lieu of a interest will not be considered in the same light. These must be returned only out of the payer's share of net profits.

PROPOSED DEAL:

This will be on the above basis with the leaser retaining a 60% interest in the net profits but giving operating control to the minority interest (40%) Purchaser who must satisfy the terms of the present lease. A certain amount of cash to cover the value of present usable equipment and leaser's back salary, must be paid upon signing of any deal and further payments due after the secondary development is finished will be decided by negotiation. If the operating party does not intend to proceed when the secondary development has been completed, it shall not retain any equity in return for its outlay.

Thus it may be seen that any company which takes over the lease has to gamble only the cash-down payment and the $15,000 for secondary development. When that amount has been spent the final size of the mill may be determined as the ore will be proven, if it is present at all. If proven, there will be no further gamble as to the return of the entire investment. There are relatively few prospects with the ore showings and block of the Patmore mine that can be proven up at such a low initial expenditure.

(Signed) W.H. Patmore
(mining geologist)