880646

Bell and Granisle porphyry copper-gold mines, Babine region, west-central British Columbia

G.E. DIROM
Consulting Geological Engineer, North Vancouver, British Columbia

M.P. DITTRICK
Consulting Geologist, Whitehorse, Yukon

D.R. McARTHUR
Noranda Mining and Exploration Inc., Bell Mine Division, Granisle, British Columbia

P.L. OGRYZLO
Consulting Geologist, Vancouver, British Columbia

A.J. PARDOE
Consulting Geologist, Quick, British Columbia

P.G. STOTHART
Brunswick Mining and Smelting Corporation Ltd., Bathurst, New Brunswick

ABSTRACT

The Bell and Granisle open pit mines exploited two porphyry copper-gold deposits in the Babine Lake region of central British Columbia. The Bell mine produced 303 277 tonnes copper, 12 794 kg gold and 27 813 kg silver from 77.2 million tonnes of ore, while the Granisle mine produced 214 300 tonnes copper, 6833 kg gold and 69 753 kg silver from 52.7 million tonnes of ore.

The deposits are associated with intrusive rocks of a Tertiary continental magmatic arc known as the Babine Igneous Suite. This suite consists of the remnants of volcanic edifices constructed on rocks of the Stikine terrane during Eocene time. Tertiary extension and transtensional faulting resulted in the formation of a series of northwesterly trending grabens in the Babine Lake region. The most prominent of these structures is the Morrison Graben which is bounded on the east by the Morrison Fault and its offset, the Newman Fault. Dikes and plugs of intermediate to felsic calc-alkaline porphyritic intrusive rocks were emplaced along these faults with extrusive equivalents preserved in downdrop basins as flows, debris flows, hornblende crystal tuff and piles of poorly consolidated volcanic rubble. Volcanism was locally explosive, with coarse breccias plugging volcanic vents.

The Bell deposit is a classic high-level porphyry copper-gold deposit with symmetrical zones of biotite-magnetite and propylitic alteration overprinted by pervasive quartz-sericite alteration. The principal sulphides are chalcopyrite and pyrite occurring as disseminations, fracture fillings and in an intensively developed quartz stockwork. Symmetry was subsequently disrupted by explosion and collapse, resulting in the partial destruction of the upper part of the southeastern quadrant of the deposit.

The Granisle deposit also has well developed biotite-magnetite/propylitic alteration zoning, but less extensive development of a pervasive quartz-sericite overprint. The principal sulphides are chalcopyrite, bornite and pyrite. The Granisle deposit appears to be exposed at a lower level than the Bell deposit and may represent the root zone of a porphyry system.

Operating practices at Bell and Granisle are described with emphasis on their evolution in response to technological and economic change.

An extensive examination of the Bell deposit, beginning in 1988, identified a potential open pit resource of 70.4 million tonnes grading 0.44% Cu and 0.20 g/t Au at a strip ratio of 1.9:1. Economic considerations prevented development of this resource at the time of the mine closure in 1992.

Introduction

The past-producing Bell and Granisle porphyry copper-gold mines operated by Noranda Minerals Inc. (now Noranda Mining and Exploration Inc.) are situated in the Babine Lake region of west-central British Columbia, some 65 km east of Smithers (Fig. 1). Bell mine (latitude 55°01′N, longitude 126°14′W), is on Newman Peninsula and operated from 1972 to 1992. Granisle mine (latitude 54°57′N, longitude 126°08′W) is on McDonald Island, 8 km to the southeast of Bell mine, and operated from 1966 to 1982.

Topography in the region is subdued with maximum elevations at both mines being 825 m above sea level (asl) or 110 m above the mean level of Babine Lake. Prior to development, the Bell deposit was covered by 3 m to 30 m of lake bottom sediments and glacial till, while the surface expression of the Granisle deposit was a small rocky knoll referred to by Emmens (1914) as the "bare hill".

The Bell and Granisle mines, together with a number of other porphyry Cu ± Au prospects in the Babine region, are hosted by small Eocene intrusions of hornblende-biotite-plagioclase porphyry known locally as "BFP". These intrusions were emplaced as part of a continental magmatic arc along north-northwest trending faults and associated north and northeast trending fault structures that developed during the latter stages of a major period of transtensional block faulting (Carson et al., 1976; Richards, 1988).

History

Newman Peninsula and McDonald Island (Fig. 2) are named after Charles Newman and H.J. McDonald, two prospectors who were active in the region during the early 1900s. Prior to 1914, Newman drove several short adits 800 m west of the current Bell open

CUDDY, A.S. and KESLER, S.E., 19‾ 'Jold in the Granisle and Bell Copper porphyry copper deposits, B .a Columbia. *In* Precious Metals in the Northern Cordillera. *Edited by* A.A. Levinson. Special Publication number 10 of the Association of Exploration Geochemists, p. 139-155.

DIROM, G.E., DITTRICK, M.P., McARTHUR, D.R., OGRYZLO, P.L., PARDOE, A.J. and STOTHART, P.G., 1995. Bell and Granisle porphyry copper-gold mines, Babine region, west-central British Columbia. *In* Porphyry Deposits of the Northwestern Cordillera of North America. *Edited by* T.G. Schroeter. Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46.

EMMENS, N.W., 1914, Babine Lake. *In* British Columbia Department of Mines, Annual Report, p. K112-K114.

FAHRNI, K.C., KIM, H., KLEIN, G.H. and CARTER, N.C., 1976. Granisle. *In* Porphyry Deposits of the Canadian Cordillera. *Edited by* A. Sutherland Brown. Canadian Institute of Mining and Metallurgy, Special Volume 15, p. 239-244.

JAMBOR, J.L., 1974. Trace element variations in porphyry copper deposits, Babine Lake area. B.C. Geological Survey of Canada, Paper 74-9.

KIRKHAM, R.V., 1971. Intermineral intrusions and their bearing on the origin of porphyry copper and molybdenum deposits. Economic Geology, 66, p. 1244-1249.

LEVINSON, A.A. and CARTER, N.C., 1979. Glacial overburden profile sampling for porphyry copper exploration, Babine Lake area, British Columbia. Western Miner, 52, Number 5, p. 19-31.

MACKIE, J., 1972. Report of geology, magnetometer and IP surveys, Add, Brian, Val Mineral claims, Mt. Horetzky. British Columbia Ministry of Energy, Mines and Petroleum Resources, Assessment Report 3870.

MALOTT, M.J., 1989. Fireweed (093M151). *In* Exploration in British Columbia 1988. British Columbia Ministry of Energy, Mines and Petroleum Resources, p. B127-131.

McMILLAN, W.J., 1 Porphyry deposits in the Canadian Cordillera. *In* Ore Deposits, Tectonics and Metallogeny in the Canadian Cordillera. British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1991-4.

NEWELL, J.M., CARTER, N.C. and SUTHERLAND BROWN, A., 1995. Porphyry Deposits of the Northwestern Cordillera: A Retrospective. *In* Porphyry Deposits of the Northwestern Cordillera of North America. *Edited by* T.G. Schroeter. Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46.

OGRYZLO, P.L., DIROM, G.E. and STOTHART, P.G., 1995. Morrison - Hearne Hill copper-gold deposits, Babine region, west-central British Columbia. *In* Porphyry Deposits of the Northwestern Cordillera of North America. *Edited by* T.G. Schroeter. Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46.

RICHARDS, T.A., 1990. Geology and mineral deposits of the Hazelton Map area (93M), British Columbia. Geological Survey of Canada, Open File 2322.

SHEETS, R.W. and NESBITT, B., 1994. Differences in biotite and apatite mineral chemistry from barren and mineralized porphyritic intrusions of the Babine Lake area, west-central B.C. *In* Final Program with Abstracts. Geological Association of Canada - Mineralogical Association of Canada Annual Meeting, Waterloo, p. A102.

WILSON, J.W.J., KESLER, S.E., CLOKE, P.L. and KELLY, W.C., 1980. Fluid inclusion geochemistry of the Granisle and Bell Porphyry Copper Deposits, British Columbia. Economic Geology, 75, p. 45-61.

ZALUSKI, G., NESBITT, B. and MUEHLENBACHS, K., 1994. Hydrothermal alteration and stable isotope systematics of the Babine porphyry Cu deposits, British Columbia: implications for fluid evolution of porphyry systems. Economic Geology, 89, p. 1518-1541.

Photograph taken by Grant Webb looking south along Newman Peninsula with Bell in foreground and Granisle in background. Grant Webb flew bush planes throughout northern Canada and Alaska and recorded the landscapes he saw with his camera. He lived the bush pilot's dream with a passion and combined it superbly with photography to bring to others images of a world accessible only by air. On January 13, 1993, while captain of a DC-3 cargo plane, Grant was killed in a crash moments after takeoff from Bronson Creek, British Columbia. The photographs Grant created during his flying career are now in the care of his wife Debbie and daughters Ashley and Amber in Smithers, British Columbia, Canada.

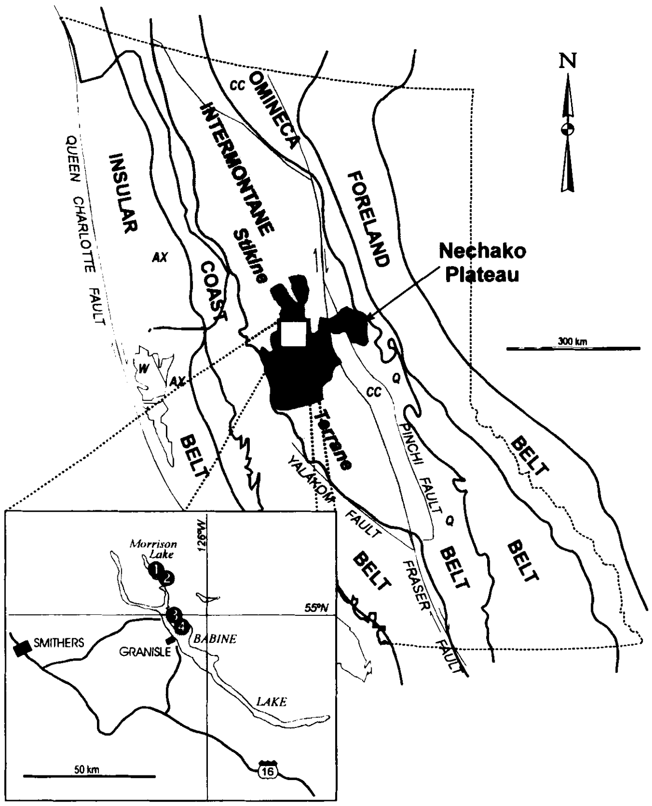


FIGURE 1. Location map with principal tectono-stratigraphic belts and terrane boundaries. Modified from Gabrielse and Yorath (1989). The principal terranes are AX (Alexander); CC (Cache Creek); Q (Quesnellia); W (Wrangellia). Stikine terrane is in light stipple (modified from Richards, 1988). Inset shows location of the major deposits of the Babine porphyry copper district. 1. Morrison. 2. Hearne Hill. 3. Bell mine. 4. Granisle mine.

pit to test shear and vein related gold, silver, lead, zinc and copper mineralization. During the same period, Newman and McDonald were also associated with the development of a number of exploration workings that tested disseminated and breccia related copper mineralization and gold, silver, lead, zinc and copper veins within and adjacent to what is now the Granisle open pit (Emmens, 1914).

In 1929, The Consolidated Mining & Smelting Company of Canada, Limited drilled five diamond drill holes in the "bare hill" region at Granisle and outlined 7.2 million tonnes grading 0.8% Cu, 0.3 g/t Au and 5 g/t Ag (Dolmage, 1943). The property lay dormant until 1945-1946 when four holes totalling 675 m were drilled.

The Granby Consolidated Mining Smelting and Power Company Limited acquired the Granisle property in 1955 and conducted a drilling program consisting of vertical holes spaced 60 m apart which tested the Granisle deposit to a depth of 90 m. Subsequent infill drilling led to a feasibility study in April, 1964 based on 7765 m of drilling in 66 vertical drill holes. This feasibility study indicated a mineable open pit reserve of 20.6 million tonnes grading 0.53% Cu to a depth of 146 m (652 m asl) using a cutoff grade of 0.3% Cu and a waste to ore strip ratio of 0.15:1.00. Construction was begun in 1965 and production, at 4500 tonnes per day, was achieved in 1966 (Parliament, 1964; Fahrni et al., 1976).

Production from the Granisle mine during its operating life totalled 52.7 million tonnes averaging 0.47% Cu with an average waste to ore ratio of 1.37:1. Cutoff grade varied between 0.20% and 0.35% Cu depending on economic conditions. Recovered metals totalled 214 300 tonnes copper, 6833 kg gold and 69 753 kg silver.

In 1962, Noranda Exploration Company, Limited (now Noranda Mining and Exploration Inc.) staked eleven claims on Newman Peninsula covering the original Newman workings on the shore of Babine Lake and 'ucted limited pace and compass controlled geological, electromagnetic and magnetic surveys (Dirom, 1962; O'Keefe, 1993). Strong electromagnetic anomalies were detected extending east from the original Newman workings. These anomalies and associated weak magnetic correlations led to the staking of additional claims and follow-up geological, electromagnetic, magnetic, silt and soil surveys over grids established in late 1962 and early 1963 (Dirom, 1964).

Three diamond drill holes were drilled in July, 1963 to test selected electromagnetic and copper soil anomalies. The first hole intersected strong pyrite mineralization (10% to locally massive pyrite) within what was later recognized as the pyrite halo associated with the Bell orebody. The next two holes intersected strongly altered, partially oxidized sedimentary rocks and feldspar porphyry containing disseminated and fracture filling copper mineralization with erratic but significant grades up to 0.82% Cu over 1.52 m. The fourth hole, a vertical hole drilled early in 1964 to test a −350 mV self-potential anomaly 90 m west of DDH 3, averaged 0.94% Cu from bedrock at 4 m to end of hole at 62 m. These holes, collectively, were the discovery holes at Bell and set the stage for the ultimate development of the Bell copper-gold deposit.

Although many people contributed to the development of the Bell mine, Noranda's history of discovery in the Babine region during the period August, 1962 to June, 1964 is generally attributed to the efforts of Noranda's Norpex Group (Newell et al., this volume; O'Keefe, 1993). The mine was appropriately named in honour of A.M. (Archie) Bell, General Manager of Noranda Exploration Company, Limited at the time of discovery.

Drilling and exploration at Bell during 1964 to 1968 led to a 1969 feasibility study which indicated a mineral resource of 116 million tonnes averaging 0.48% Cu. Contained within this mineral resource were mineable open pit reserves of 42 million tonnes grading 0.50% Cu to a depth of 470 m using a cutoff grade of 0.30% Cu. The waste to ore strip ratio was calculated at 0.52:1, after removal of some 5.5 million tonnes of overburden (Hall and Kraft, 1969). Gold and silver grades were estimated at 0.35 and 1.0 g/t, respectively, with molybdenum averaging less than 0.01%. Initial gold, silver and molybdenum estimates were based on metallurgical test results as systematic assaying for these elements had not been done.

Production from the Bell mine during its operating life totalled 77.2 million tonnes averaging 0.47% Cu with an average waste to ore ratio of 0.98:1. The cutoff grade varied between 0.25% and 0.35% Cu depending on economic conditions. Total recovered metals were 303 277 tonnes copper, 12 794 kg gold and 27 813 kg silver.

Regional Geology

The Babine region is situated in the Intermontane Belt of west-central British Columbia (Fig. 1). It is underlain by late Paleozoic - Mesozoic volcanic and sedimentary rocks of the Intermontane Superterrane (Gabrielse and Yorath, 1989; Wheeler and McFeely, 1991), whose boundaries coincide approximately with those of the Intermontane Belt. The Intermontane Superterrane comprises a number of accreted terranes which amalgamated prior to being welded to the North American craton in Late Jurassic to Early Cretaceous time. The largest of the accreted terranes of the Intermontane Belt is the Stikine terrane.

The tectonic history of the Babine region has three principal phases, progressing from a system of island arcs in the Late Triassic to Early Jurassic, through a middle to late Mesozoic post-orogenic molasse sequence and culminating with a continental magmatic arc in the Late Cretaceous and Early Tertiary (Monger et al., 1972). The Stikine terrane evolved as a collage of island arcs with an internal stratigraphic record independent from that of the North American craton prior to amalgamation. Alkaline and calc-alkaline volcanic, volcaniclastic and intercalated sedimentary rocks accumulated as subaqueous and emergent volcanic piles from late Paleozoic to early Mesozoic time.

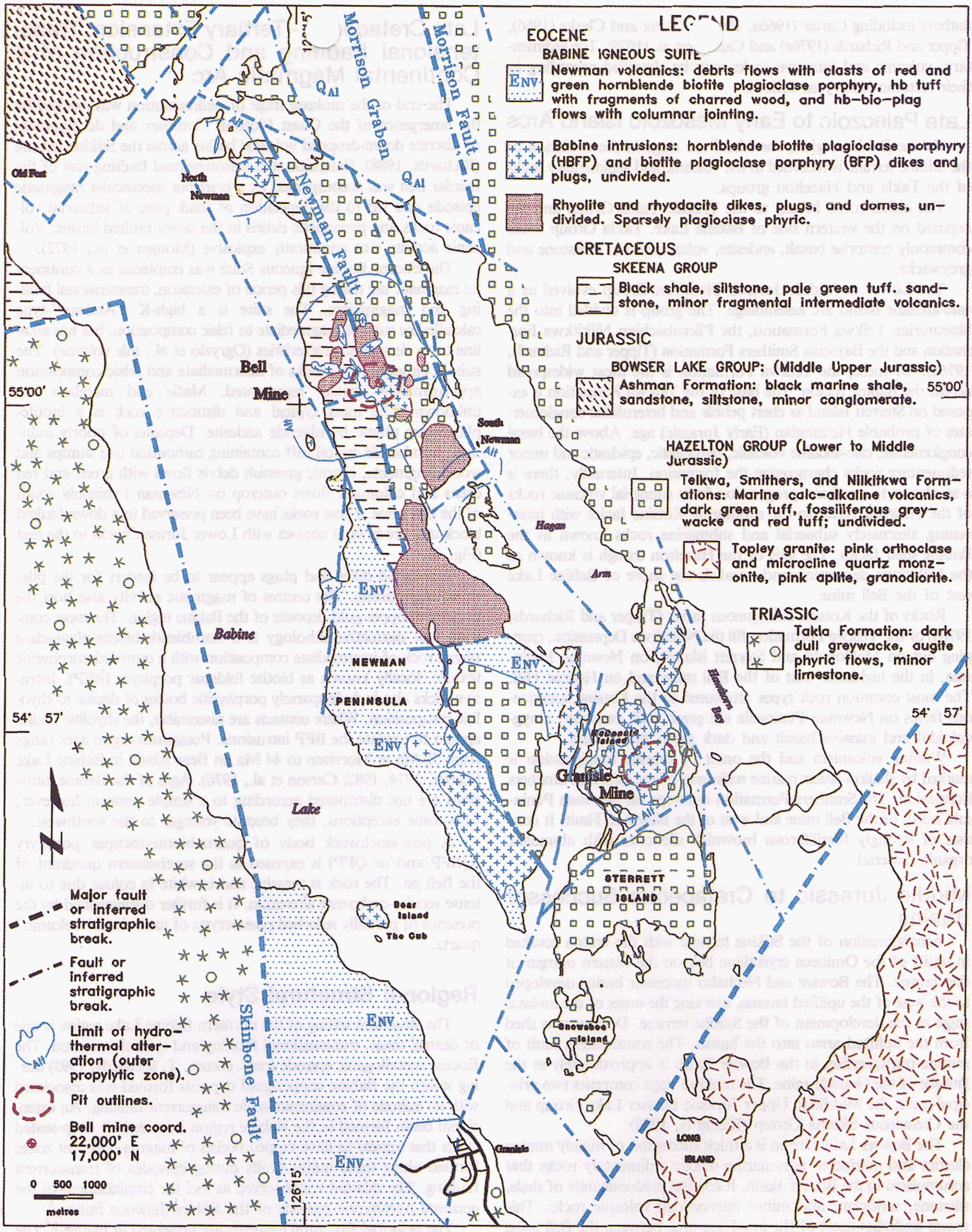


FIGURE 2. Geology of northern Babine Lake region including Newman Peninsula and McDonald Island. Compiled from Carter (1966 a,b), Carter and Clarke (1966), Tipper and Richards (1976b), Carson et al. (1976), Granby and Noranda pit mapping, exploration files, and mapping by the authors.

Amalgamation of the Stikine terrane with the North American continent was followed by accumulation of molasse-type sedimentary rocks in late Middle Jurassic to Early Cretaceous successor basins. Late Cretaceous to Early Tertiary magmatism led to the construction of Andean-type continental volcanic piles on the Stikine terrane. The porphyry copper-gold deposits of the region are hosted by epizonal intrusions associated with the Tertiary magmatic arc.

The Babine area geology has been described by numerous authors including Carter (1966a, 196‸ Carter and Clarke (1966), Tipper and Richards (1976a) and Car‸, et al. (1976). The sedimentary, volcanic and intrusive rocks may be classified according to their tectonic history as follows.

Late Paleozoic to Early Mesozoic Island Arcs

In the Babine region, the island arc stage in the evolution of the Stikine terrane is recorded in the volcanic and sedimentary rocks of the Takla and Hazelton groups.

The oldest rocks belong to the Triassic Takla Group and are exposed on the western side of Babine Lake. Takla Group rocks commonly comprise basalt, andesite, volcaniclastics, limestone and greywacke.

The Lower to Middle Jurassic Hazelton Group evolved as a calc-alkaline island arc assemblage. The group is divided into the Sinemurian Telkwa Formation, the Pliensbachian Nilkitkwa Formation and the Bajocian Smithers Formation (Tipper and Richards, 1976a). Of these, the Telkwa Formation is the most widespread across the Babine region. The base of the Telkwa Formation is exposed on Sterrett Island as chert pebble and heterolithic conglomerates of probable Hettangian (Early Jurassic) age. Above the basal conglomerate, calc-alkaline volcanic, pyroclastic, epiclastic and minor sedimentary rocks characterize the formation. Internally, there is a southwest to northeast progression from subaerial volcanic rocks of the Howson facies to the submarine Kotsine facies with intervening alternately subaerial and submarine rocks known as the Babine shelf facies. The axis of the Hazelton trough is known as the Nilkitkwa depression and parallels the shore of Babine Lake east of the Bell mine.

Rocks of the Kotsine subaqueous facies (Tipper and Richards, 1976a) of the Telkwa Formation fill the Nilkitkwa Depression, cropping out on McDonald and Sterrett islands, on Newman Peninsula, in the highlands east of the Bell mine and on Hearne Hill. The most common rock types attributed to the Kotsine subaqueous facies on Newman Peninsula are green tuff, vesicular, amygdaloidal and massive basalt and dark green pyroclastics.

Waning volcanism and the onset of marine transgression is marked by shallow-water marine sediments of the Bajocian Smithers Formation. The Smithers Formation outcrops on Newman Peninsula north of the Bell mine and west of the Newman Fault. It consists of strongly fossiliferous brownish siltstones with abundant organic material.

Middle Jurassic to Cretaceous Successor Basins

Amalgamation of the Stikine terrane with the craton resulted in uplift of the Omineca crystalline belt on the eastern margin of the terrane. The Bowser and Nechako successor basins developed to the west of the uplifted terrane, marking the onset of the molasse stage in the development of the Stikine terrane. Detritus was shed from the uplifted areas into the basins. The southeastern limit of marine transgression in the Bowser Basin is approximately at the latitude of the Granisle mine. The molasse stage comprises two principal units, the Middle to Upper Jurassic Bowser Lake Group and the Cretaceous Skeena Group (Richards, 1988).

The Bowser Lake Group is a thick succession of mainly marine clastics and marine to non-marine deltaic sedimentary rocks that accumulated in the Bowser Basin. It consists predominantly of shale, siltstone, sandstone and minor interbedded volcanic rocks. This group underlies most of the low-lying area between the Bell mine and Morrison Lake.

The Middle Cretaceous Skeena Group comprises interbedded marine and non-marine sedimentary rocks together with basic and intermediate flows and breccias. It is regionally widespread and occurs on the western side of Newman Peninsula and in the Bell pit where its faulted contact with the Hazelton Group is obscured by Eocene intrusions. Black shale with wisps of sandstone, pale green tuff, minor bentonite and brownish siltstone characterize Skeena Group rocks on Newman Peninsula.

Late Cretace‸ - Tertiary Extension, Transtensional Fau‸ting and Construction of a Continental Magmatic Arc

The end of the molasse stage of sedimentation was marked by the emergence of the Coast Plutonic Complex and development of discrete down-dropped volcanic basins across the Stikine terrane (Richards, 1988). Extension and transtensional faulting east of the Insular Belt was accompanied by a brief but spectacular magmatic episode that led to the deposition of thick piles of subaerial volcanic rocks and pyroclastic debris in the down-faulted basins. Volcanic activity was commonly explosive (Monger et al., 1972).

The Eocene Babine Igneous Suite was emplaced as a continental magmatic arc during this period of extension, transtensional faulting and magmatism. The suite is a high-K "Andean"-type calc-alkaline suite of intermediate to felsic composition, but has alkaline trace element characteristics (Ogryzlo et al., this volume). The suite is bimodal, with rocks of intermediate and felsic composition approximately equally represented. Mafic end members are unknown. The most typical and distinctive rock is a biotite-plagioclase phyric hornblende andesite. Deposits of poorly indurated hornblende crystal tuff containing carbonized tree stumps and wood fragments, chaotic greenish debris flows with green and red clasts and columnar flows outcrop on Newman Peninsula south of the Bell mine. These rocks have been preserved in a down-faulted block and are in fault contact with Lower Jurassic rocks to the east (Fig. 2).

Associated dikes and plugs appear to be feeders for the piles of volcanic debris. The centres of magmatic activity also host the porphyry copper-gold deposits of the Babine region. The most common and distinctive lithology is a hornblende-biotite-plagioclase phyric rock of intermediate composition with a crowded porphyritic texture, locally known as biotite feldspar porphyry (BFP). Intrusive rocks also include sparsely porphyritic bodies of dacitic to rhyolitic composition. Where contacts are discernible, the rhyolitic bodies appear to predate the BFP intrusions. Potassium-argon ages range from 52 Ma at Morrison to 44 Ma on Bear Island in Babine Lake (Carter, 1974, 1982; Carson et al., 1976). Ages of the Babine intrusions are not distributed according to a simple pattern; however, with some exceptions, they become younger to the southwest.

A post-stockwork body of quartz-biotite-feldspar porphyry (QBFP and/or QFP) is exposed in the southeastern quadrant of the Bell pit. The rock is massive and is white in colour due to intense sericite-carbonate alteration. It is further distinguished by the presence of partially resorbed phenocrysts of unstrained "volcanic" quartz.

Regional Structural Style

The structural setting of the northern Babine Lake region is one of dextral shear, transtensional faulting and crustal extension. The Eocene metallogenic episode (pers. comm. T. Richards, 1993) during which the Babine copper-gold deposits formed was associated with an episode of continental-scale transcurrent faulting. An extensional basin formed in the Babine region bounded by deep-seated faults that appear to have tapped bodies of magma. Dilatant zones formed along these major faults during episodes of transcurrent faulting. The dilatant zones served as loci for emplacement of the epizonal porphyritic plutons of the Babine Igneous Suite.

The principal structural elements are presented in Figure 2. The Late Cretaceous-Early Tertiary northeast-southwest directed extension (pers. comm., T. Richards, 1993) formed a series of northwest trending grabens. The most prominent is Morrison Graben which extends for approximately 35 km from the northern end of Hagan Arm (Fig. 2) to the northern end of Morrison Lake and ranges from 2 km to 4 km in width. Rocks of the Middle to Upper Jurassic Ashman Formation of the Bowser Lake Group underlie much of the graben and are bounded to the east and west by uplifted horst blocks formed by older rocks of the Lower to

Middle Jurassic Hazelton Group. The Mc   n Fault, which forms the eastern boundary of the graben, can raced as a prominent northwesterly linear marked by fault scarps over a distance of approximately 100 km. It appears to be offset at the north end of Hagan Arm by a dextral transcurrent fault toward the centre of Newman Peninsula, where its continuation is known as the Newman Fault (Carter, 1974). The Newman Fault is "stitched" by a series of porphyritic felsic and intermediate hypabyssal plutons of the Eocene Babine intrusions. It terminates at its southern end against the uplifted block of basal Jurassic rocks which underlie McDonald and Sterrett islands (Fig. 2).

At Bell, multiple, upwardly diverging bodies of rhyolite, dacite and biotite feldspar porphyry (BFP) were emplaced across the trace of the Newman Fault. At Granisle, a BFP dike trending 025° cuts across McDonald Island and is bracketed by two major northwesterly trending faults. The orientation of the dike suggests that it may occupy a tensional feature created by dextral shear between the coupled faults.

Many of the more significant porphyry copper-gold deposits in the Babine region are associated with the Morrison Graben and even more closely with the Morrison and Newman faults which bound it on the east. The Granisle and Bell mines (Fig. 2) lie close to the trace of the Newman Fault while the Hearne Hill deposit is located on the scarp of Morrison Fault, overlooking the Morrison deposit in the Morrison Graben (Fig. 1). Porphyry prospects of lesser importance within the Morrison Graben include South Newman, North Newman and Sparrowhawk. Porphyry prospects are also known to be associated with parallel to subparallel graben and fault structures immediately east and west of the Morrison Graben.

A final episode of extension superposed a rift on the pre-existing northwesterly trending horst and graben structures. The main body of Babine Lake and parts of Newman Peninsula are underlain by a north trending down-dropped basin filled with Eocene volcanic rocks. The northwest arm of the rift is underlain by down-faulted Cretaceous sedimentary rocks and the northeast arm is a failed rift connecting Babine Lake to Morrison Graben (Fig. 2).

Geology of the Bell Porphyry Copper-Gold Deposit

Introduction

The Bell deposit is a classic symmetrically zoned porphyry copper-gold deposit spatially associated with multiple phase subvolcanic intrusions of the Babine Igneous Suite. Symmetry of mineralization and hydrothermal alteration has been significantly modified and disrupted by explosion and collapse events and by the emplacement of a postmineral subvolcanic mass of quartz ± biotite-feldspar porphyry (QBFP/QFP).

The deposit is situated on the central portion of Newman Peninsula (Fig. 3). The peninsula is dissected by the northwest trending Newman Fault. This structure marks the faulted contact between Cretaceous Skeena Group and Eocene rocks to the west and Lower Jurassic Hazelton Group rocks to the east. The Eocene Babine Intrusions which host the copper-gold mineralization were emplaced at the intersection of this fault with an east-northeast trending fault that cuts across the peninsula.

The Bell orebody is crescent shaped in plan between bedrock surface (~750 m) and the 580 m elevation, with the arc open to the southeast. Below the 580 m elevation, the shape is annular. The vertical extent of the Bell mineralization is unknown; however, it probably extends well below the deepest drill hole which terminated in material grading 0.55% Cu at an elevation of 150 m. The deposit straddles the trace of Newman Fault, with most of the ore lying immediately west of the fault. The northeast limb of the deposit extends 500 m to the east of the fault.

The deposit is surrounded by a broad pyrite halo extending radially outward some 1100 m. Rocks within the pyrite annulus contain an average of 10% ite as disseminations, stringers and fracture fillings.

A zone of hydrothermal alteration extends outward over a radius of some 1500 m from the centre of the deposit affecting virtually all rocks within the central Newman Peninsula. Rocks within the Bell open pit are often so intensely altered that identification of the original rock is difficult.

Hydrothermal alteration in the porphyry copper deposits of the Babine region developed first as an inner core of potassic alteration mantled by a shell of propylitic alteration. This pattern of central potassic (biotite-magnetite) and peripheral propylitic (chlorite-carbonate) alteration is best preserved in the Morrison and Hearne Hill deposits. A moderately to intensely developed sericite-carbonate alteration overlaps the potassic/propylitic zone at the Bell and Granisle deposits. In addition, a well developed, quartz-sericite stockwork occurs peripheral to the biotite-magnetite core in the Bell deposit. Carson et al. (1976) concluded that the sericite-carbonate and quartz-sericite record later stages of alteration which overprinted the earlier chlorite-carbonate and biotite assemblages.

Description of Hostrocks

Hazelton Group

Rocks assigned to the Telkwa Formation of the Lower Jurassic Hazelton Group occur east of Newman Fault and are the oldest exposed at Bell. Unaltered rocks occur on the northern and southern parts of Newman Peninsula and along its eastern shore. The dominantly marine succession consists of green aquagene tuff and tuffaceous argillite overlying massive light green volcanic flows, amygdaloidal basalt and green volcaniclastics.

Within the Bell deposit, Hazelton Group rocks have experienced weak to intense alteration. Diamond drilling and pit wall mapping have shown them to be predominantly tuffs with minor intercalated siltstone. Alteration is dominantly sericitic to weak quartz-sericitic resulting in an aphanitic grey-buff coloured rock known as "buff tuff" in mine terminology. Zones of medium to dark brown-grey biotite-altered tuff occur in the central northeast portion of the Bell pit. The unit is cut by a fine network of hairline fractures commonly displaying sericitic alteration envelopes. Sulphide mineralization occurs as disseminations and in stockworks. Copper grades vary from trace to over 1%.

Skeena Group

Lower to Middle Cretaceous Skeena Group rocks are found west of Newman Fault and consist of interbedded tuff, siltstone, argillite and shale. Rare beds of pale green, swelling bentonite were observed in diamond drill core.

Skeena Group rocks adjacent to and within the pit display varying intensities of alteration with sericite-carbonate alteration predominant. In areas with intense alteration, original textures have been destroyed. Where rock type could not be distinguished, the rocks were classified as undivided sedimentary rocks. Rocks are typically very fine grained and buff-yellow-grey to dark grey in colour. Sedimentary rocks generally display a fine hairline stockwork containing up to 3% disseminated sulphide mineralization.

Chlorite-carbonate altered rocks are exposed on the west and northwest side of the pit. The fine-grained rocks are pale to dark green with buff sericitic and brown biotitized sections. The dark green colour is imparted by smectite clays in the siltstones after hornfelsing or hydrothermal alteration has removed the darker organic matter. Hairline stockwork veinlets locally display sericitic alteration envelopes. Sulphides occur as disseminations and fine fracture coatings. Exposures of Skeena Group rocks on the upper benches of the Bell pit were baked to a brittle dark brown biotite hornfels representing early developed biotite.

Babine Igneous Suite

Subvolcanic intrusions of Eocene rhyolite, rhyodacite and biotite feldspar porphyry (BFP) were emplaced across the trace of the

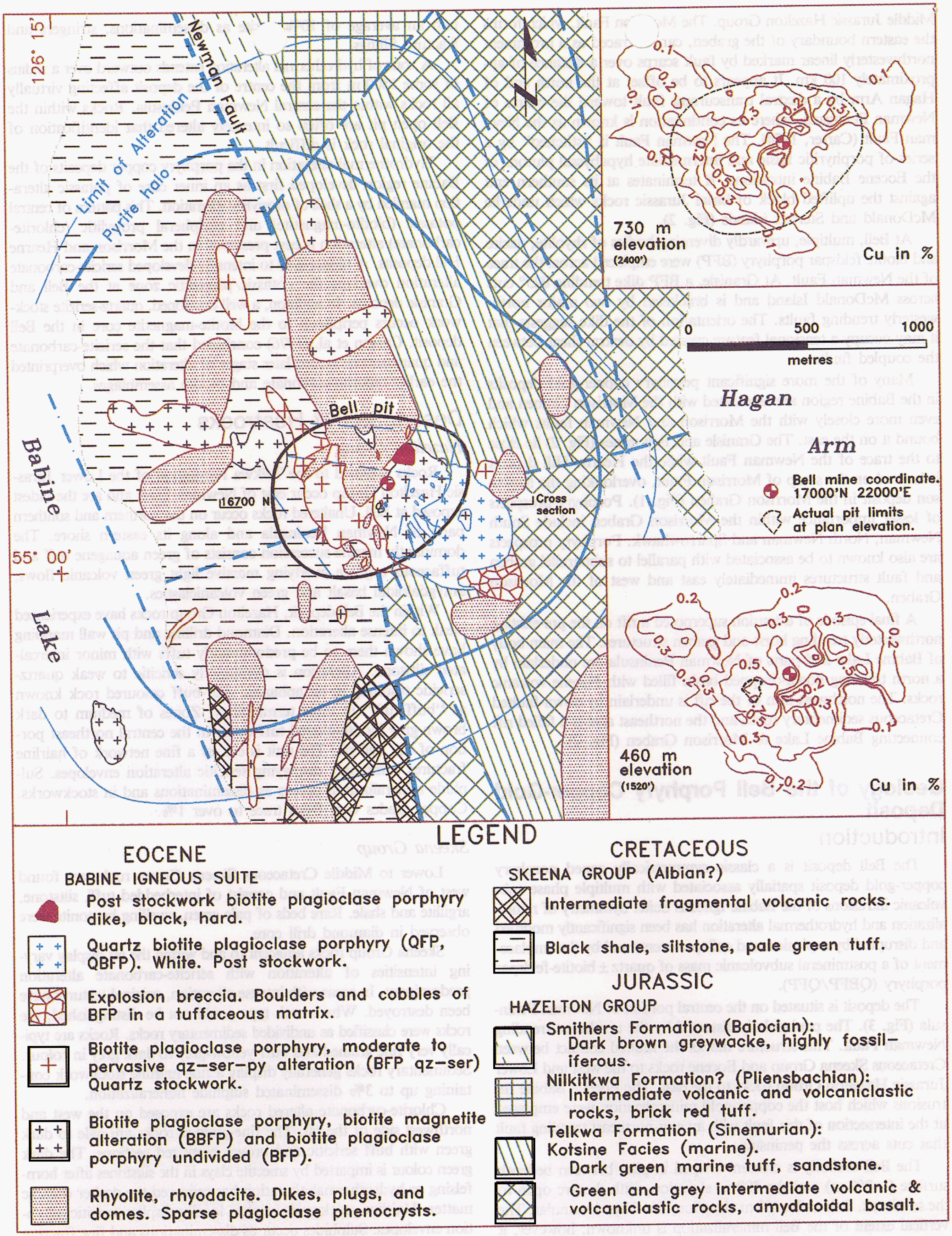


FIGURE 3. Geology of the Bell porphyry copper-gold deposit. Compiled from Carson et al. (1976), company diamond drill maps and mapping by the authors. Insets show distribution of copper grades at the 730 m and 460 m elevations.

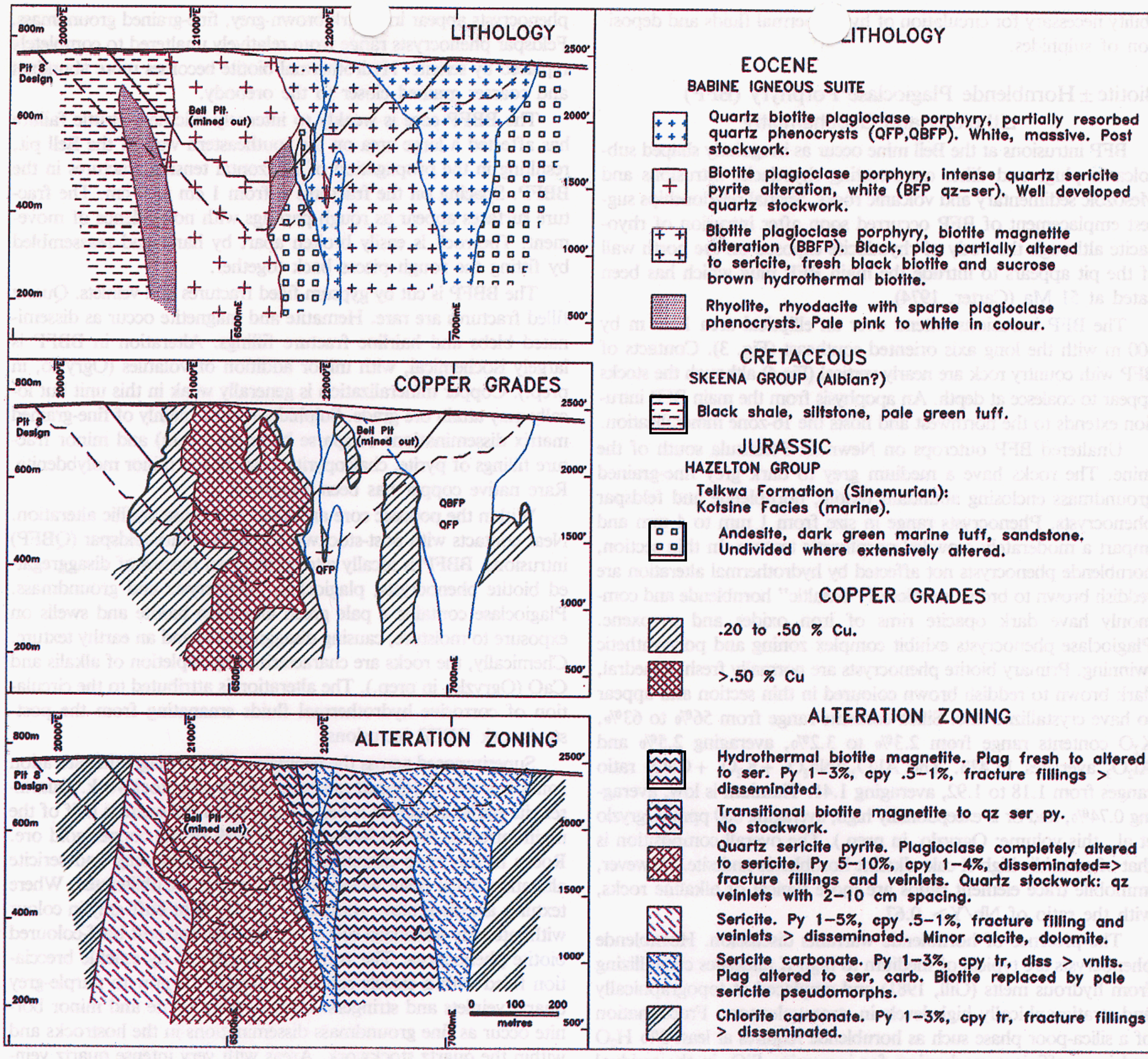


FIGURE 4. Bell mine section 16700 North showing lithology, copper distribution and alteration zoning. Compiled from diamond drill logs, pit mapping and blast hole sampling.

Newman Fault on Newman Peninsula. Extrusive equivalents are preserved in a down-faulted block 2 km south of the Bell mine, indicating the intrusive complex once formed part of a volcanic pile. The felsic rocks appear to occupy a dilatant zone formed along the Newman Fault. Successive pulses of BFP intrusions may indicate a magmatic centre which migrated across the peninsula from west to east. The final intrusive pulse was preceded by an explosion, possibly laterally directed, that destroyed a part of the southeastern portion of the orebody.

Rhyodacite

Eocene rhyodacite occurs as large masses exposed across the middle of the Newman Peninsula. Rocks of dacitic to rhyolitic composition are grouped as undivided rhyodacite (Fig. 3). Unaltered specimens contain approximately 70% SiO_2. In general, they consist of a very pale pink, greyish-green, buff or white aphanitic groundmass with scattered indistinct white feldspar phenocrysts (<2 mm). Textures in altered rhyodacite range from earthy and soft in sericite-carbonate altered rocks to vitreous and hard in quartz-sericite altered rocks. The rocks are usually brittle with broken surfaces being sharp and angular. Weathered exposures on the north wall of the Bell pit are encrusted with bright yellow jarosite, white melanterite and blue cuprian melanterite (Carson et al., 1976). The growth of sulphates may be related to the underlying whole-rock composition; the highly siliceous rocks are low in CaO, MgO and other acid consuming constituents.

Rhyodacite outside the alteration halo may be either brecciated or massive; some breccia fragments display flow banding. Closer to the orebody, sericite alteration, hairline fractures and stockwork development increase. Fine-grained sulphide disseminations predominate with minor sulphide mineralization within the fractures and stockwork.

Rhyodacites and adjacent rocks within the ore zone exhibit intense quartz-sericite alteration. Feldspar phenocrysts are scattered and indistinct, almost blending into the groundmass. The unit is cut by a fine, intense stockwork of grey to purple-grey quartz veinlets and sulphide stringers containing chalcopyrite and pyrite as blebs and disseminations. Finely-divided sulphide disseminations are also common in the matrix. Lesser amounts of hematite and magnetite occur as hairline fracture fillings. Roughly 30% of the Bell orebody is contained in rhyodacite; pervasive crackle brecciation of the siliceous and brittle rock provided the high porosity and permeability necessary for circulation of hy ıermal fluids and deposition of sulphides.

Biotite ± Hornblende Plagioclase Porphyry (BFP)

Lithology and Geochemistry

BFP intrusions at the Bell mine occur as irregularly shaped subvolcanic plugs and dikes cross-cutting rhyodacite intrusions and Mesozoic sedimentary and volcanic rocks. Spatial relationships suggest emplacement of BFP occurred soon after intrusion of rhyodacite although the body of rhyodacite exposed on the north wall of the pit appears to intrude the main BFP plug which has been dated at 51 Ma (Carter, 1974).

The BFP intrusions extend over an elliptical area 1200 m by 600 m with the long axis oriented southeast (Fig. 3). Contacts of BFP with country rock are nearly vertical (Fig.4) although the stocks appear to coalesce at depth. An apophysis from the main BFP intrusion extends to the northwest and hosts the 16-zone mineralization.

Unaltered BFP outcrops on Newman Peninsula south of the mine. The rocks have a medium grey to dark grey fine-grained groundmass enclosing abundant biotite, hornblende and feldspar phenocrysts. Phenocrysts range in size from 1 mm to 4 mm and impart a moderately crowded porphyritic texture. In thin section, hornblende phenocrysts not affected by hydrothermal alteration are reddish brown to brown needles of "basaltic" hornblende and commonly have dark opacite rims of iron oxides and pyroxene. Plagioclase phenocrysts exhibit complex zoning and polysynthetic twinning. Primary biotite phenocrysts are normally fresh, euhedral, dark brown to reddish brown coloured in thin section and appear to have crystallized late. Silica contents range from 56% to 63%, K_2O contents range from 2.3% to 3.2%, averaging 2.5% and Al_2O_3 averages 15.4%. The $Al_2O_3/(Na_2O + K_2O + CaO)$ ratio ranges from 1.18 to 1.92, averaging 1.41. Titanium is low, averaging 0.74%, and Sr is exceptionally high, averaging 880 ppm (Ogryzlo et al., this volume; Ogryzlo, in prep.). The overall composition is that of an acidic high-K calc-alkaline hornblende andesite. However, immobile trace element ratios are more typical of alkaline rocks, with the ratio of $Nb/Y > 0.67$.

The presence of hornblende warrants discussion. Hornblende phenocrysts are typical of medium to high-K andesites crystallizing from hydrous melts (Gill, 1981) and emplaced at topographically and stratigraphically high levels in stratovolcanoes. Fractionation of a silica-poor phase such as hornblende requires at least 3% H_2O and is an efficient mechanism for increasing SiO_2 in the residual melt. Mineralogy and whole-rock chemistry, particularly enrichment in incompatible elements such as K and Sr, are characteristic of andesite emplaced in a continental magmatic arc at a considerable distance from a subduction zone, having traversed a considerable thickness of continental crust.

Alteration in BFP

Alteration zoning is best displayed by progressive mineralogical changes in the BFP. Hornblende is least stable and is the first mineral to show the effects of hydrothermal alteration. Chlorite and apple green epidote replace hornblende on the fringes of the alteration halo and mark the outer limits of propylitic alteration. Calcic zones in plagioclase are altered to concentric shells of carbonate and clay minerals. In the Bell pit, chlorite altered BFP occurs largely on the northwest edge with minor patches occurring throughout the deposit. Chlorite tends to impart a dark green-grey cast to the rocks. Biotite and pale green-cream coloured feldspar phenocrysts are replaced by sericite, carbonate and greenish clay minerals. Hornblende phenocrysts are replaced by chlorite. Late fractures and veinlets filled with carbonate are common. Sulphides occur as fine disseminations, as veinlets and within the carbonate fracture fillings.

Biotite altered BFP, defined as BBFP, occurs in the central part of the main intrusion and represents the potassic core of the deposit. Abundant black primary (magmatic) biotite, sucrose brown hydrothermal biotite, magnetite and cream to white feldspar phenocrysts appear in ırk brown-grey, fine-grained groundmass. Feldspar phenocrysts range from relatively unaltered to completely replaced by sericite. Hydrothermal biotite becomes more abundant and coarser grained closer to the orebody.

The BBFP core is weakly to intensely silicified. Brittle failure has affected a large area on the southeastern wall of the Bell pit, resulting in the propagation of horizontal tension fractures in the BBFP. Spacing on the fractures is from 1 cm to 4 cm. The fracture surfaces appear as rough partings with no evidence of movement. The rock is easily broken apart by hand and reassembled by fitting the rough pieces back together.

The BBFP is cut by gypsum filled fractures and veinlets. Quartz filled fractures are rare. Hematite and magnetite occur as disseminated blebs and hairline fracture fillings. Alteration in BBFP is largely isochemical, with minor addition of volatiles (Ogryzlo, in prep.). Copper mineralization is generally weak in this unit but locally may attain ore grade. Sulphides consist mainly of fine-grained matrix disseminations or sparse blebs (to 5 mm) and minor fracture fillings of pyrite, chalcopyrite, bornite and minor molybdenite. Rare native copper has been identified.

Within the potassic core are zones of intense argillic alteration. Near contacts with post-stockwork quartz-biotite-feldspar (QBFP) intrusions, BBFP is locally altered to green masses of disaggregated biotite phenocrysts, plagioclase phenocrysts and groundmass. Plagioclase contains a pale green alteration biotite and swells on exposure to moisture, causing disaggregation and an earthy texture. Chemically, the rocks are characterized by depletion of alkalis and CaO (Ogryzlo, in prep.). The alteration is attributed to the circulation of corrosive hydrothermal fluids emanating from the post-stockwork QBFP intrusions.

Superimposed across the propylitic/biotite-magnetite alteration zones is a well developed quartz-sericite-pyrite stockwork. Quartz-sericite-pyrite alteration is best developed in the western half of the main intrusion and includes roughly 50% of the copper-gold ore. Rocks in this zone exhibit pervasive quartz flooding and sericite alteration to the point where original textures are obliterated. Where textures are discernible, rocks are typically light buff-grey in colour with intensely sericitized soft white feldspar and soft buff-coloured biotite phenocrysts. The zone is characterized by crackle brecciation filled with a closely spaced stockwork of grey to purple-grey quartz veinlets and stringers. Chalcopyrite, pyrite and minor bornite occur as fine groundmass disseminations in the hostrocks and within the quartz stockwork. Areas with very intense quartz veining have mosaic breccia textures in which unrotated angular fragments may be reassembled into their original positions.

The quartz stockwork does not extend into the central core of the deposit which remains as a zone of intense biotite-magnetite alteration with only weak copper mineralization. The transition from quartz-sericite-pyrite altered BFP to the central zone of biotite altered BBFP is striking. Copper grades decrease inward from >0.50% Cu to <0.20% Cu over a distance of 10 m to 20 m, stockwork quartz veining disappears and the colour of the rocks changes from the brilliant white of the quartz-sericite zone through a pale green transitional zone to dark grey or black in the biotite-magnetite zone.

Quartz ± Biotite-Feldspar Porphyry (QBFP and QFP)

A post-mineralization intrusion of quartz-biotite-feldspar porphyry (QBFP) replaces the southeastern quadrant of the orebody above the 580 m elevation. Quartz-biotite-feldspar porphyry (QBFP) is distinguished from quartz feldspar porphyry (QFP) by the presence of discernible relics of primary biotite. Both rock types are considered as undivided QFP in the following discussion.

Exposures of QFP on the southeast wall of the pit are bright white in colour and massive in appearance. In cross-section, the main QFP body appears to have a narrow feeder dike at depth that flares upward to form a "champagne-glass" shape (Fig. 4). A number of smaller QFP dikes occur throughout the southeast

FIGURE 5. Post-stockwork QBFP/QFP dikes (light grey) intruding shattered core BBFP (dark grey). Note baked selvages along contacts and "sugar cube" jointing in BBFP.

and central east portion of the pit, as outlined by diamond drilling and pit wall mapping (Fig. 5).

The QFP is similar in appearance to intensely sericitized BFP but the two may be distinguished both texturally and mineralogically. Most notable is the absence of fracturing and stockwork development in the QFP. Chalcopyrite is present in QFP only in minor amounts as disseminations and very rarely as fracture fillings. Quartz feldspar porphyry is characterized by the presence of phenocrysts of partially resorbed unstrained "volcanic" quartz as observed in thin section. It has a more fine-grained, crowded porphyritic texture than BFP and lacks black primary biotite phenocrysts. Biotite has been intensely sericitized resulting in buff-coloured, soft, pearly and platy altered phenocrysts. Pervasive sericite-carbonate alteration has developed in feldspar to produce white to cream-coloured altered phenocrysts with a waxy to earthy texture in a light buff-grey groundmass. Weak gypsum veinlets are developed locally. Pyrite occurs as fine-grained disseminations from 1% to 3%. Chemically, hydrothermal alteration in QFP is characterized by the introduction of volatiles. Loss on ignition ranges from 8% to 12%. Depletion of alkalis, particularly Na_2O, and highly variable alumina content (15% to 24%) indicates that leaching by corrosive hydrothermal fluids has also affected QFP. Alumina enrichment indicates that Al_2O_3 has remained as a refractory phase under hydrothermal alteration as other elements were selectively removed.

Breccia

In addition to the crackle and mosaic breccias in the quartz-sericite-pyrite stockwork zone, pebble dike, collapse and explosion breccias have been observed at Bell.

FIGURE 6. Northeast collapse breccia. Note slabs of QFP with rinds of black chalcedony "floating" in rhyodacite rubble. Bell mine, 2340 bench.

Pebble dikes were mapped in the upper parts of the deposit and are narrow (1 m to 5 m) structures with sheeted vertical contacts, filled with rounded pebble-sized clasts. The breccias are cemented with pyrite and quartz. Open spaces are abundant. These features have been attributed to late-stage venting of steam and gas from the intrusion.

A prominent collapse breccia is exposed on the northeast wall of the pit along the northern contact of the QFP. The breccia is characterized by angular metre-long slabs of QFP supported by smaller clasts of tuff containing mineralized stockwork veinlets (Fig. 6). Clasts of QFP and tuff are encased with millimetre thick rinds of black chalcedony and pyrite. The breccia appears to have formed from slabs and blocks of QFP caving into a void partially filled with stockwork mineralized rubble. Fluid inclusions in the chalcedony cement are vapour filled, indicating that cementation coincided with the movement of steam through the breccia. The breccia is similar in appearance and mode of formation to mineralized breccias found elsewhere in the Babine region, particularly on Hearne Hill (Ogryzlo et al., this volume).

Explosion breccias have been recorded from diamond drilling southeast of the QFP intrusion and are poorly exposed at the base of the till on the south wall of the pit. There, the breccia consists of rounded 30 cm to 60 cm boulders of BFP in a poorly consolidated matrix with some boulders exhibiting exfoliated surfaces.

Mineralization at Bell

The core of the Bell orebody, as defined by the 0.30% Cu contour, is crescent shaped in plan between bedrock surface and 580 m elevation with the crescent open to the southeast. The northeast limb is separated from the main orebody by a barren late-stage BBFP dike referred to as the northeast waste horst. The 16-zone, a low-grade extension of the main orebody, lies northwest of the main orebody which it joins at depth. Below 580 m elevation, the main orebody becomes essentially annular in plan although grades to the southeast tend to be lower than those to the north and west (Fig. 3). The decrease in grade in the southeast quadrant may reflect structural offset caused by collapse and faulting. The low-grade core of the annulus is approximately 100 m by 200 m in size and trends 055°. It coincides with the potassic zone as represented by the core BBFP phase.

FIGURE 7a. BFP with intense quartz-sericite-pyrite alteration and veining.

FIGURE 7b. Rhyodacite with intense quartz-sericite-pyrite alteration and veining.

The principal copper mineral at Bell is chalcopyrite with minor bornite and chalcocite. Gold and silver were also recovered in concentrate. Non-economic sulphides include pyrite with minor pyrrhotite, marcasite, molybdenite, sphalerite and galena.

Mineralization occurs in all rock types but the majority of the copper-gold ore lies within pervasively quartz-sericite altered rhyodacite and BFP (Figs. 7a, 7b). In this zone, chalcopyrite and pyrite occur as fine disseminations, fracture fillings and within the intense quartz stockwork cutting the rhyodacite and BFP. The average copper head grade over the life of the Bell operation was 0.47% Cu.

The distribution and grade of gold mineralization at Bell is difficult to quantify. Drill core was not systematically assayed for gold until after 1988 and gold occurs in quantities near assay detection limit. A 1992 check-assay program covering over 500 composite samples indicated significant differences between gold values obtained using commercial fire assay methods and those obtained using in-house atomic absorption methods. The differences in some sample pairs exceeded 100% and reflect both sampling problems and laboratory bias. The Bell mill production records indicate recovered gold averaged 0.16 g/t over the life of the operation. The average head grade and recovery factor cannot be stated with certainty, but are estimated at 0.26 g/t Au and 62%, respectively. Regardless of absolute values, there is a close correlation between copper and gold at Bell with higher gold grades associated with higher copper grades. Under the scanning electron microscope, gold has been observed as discrete 50 micron particles in embayments in chalcopyrite and pyrite grains.

The distribution and grade of silver mineralization is also not fully understood. Systematic silver assaying was not done on drill core until after 1988. Some zoning is evident on the basis of blast hole assays. Production records indicate average recovered silver was 0.36 g/t from ore estimated to have a head grade of 1 g/t.

Molybdenite occurs as fracture fillings, disseminations and small blebs. It tends to be slightly more scattered spatially than copper, with higher concentrations internal to high-grade copper zones. Molybdenite also occurs outside the orebody which averages 0.005% Mo. Traces of bornite occur throughout the orebody as fine disseminations and with chalcopyrite as fracture coatings. A distinct bornite zone does not exist. Marcasite and pyrrhotite, although of minor importance, occur throughout the deposit as rare blebs and disseminations. Chalcocite occurs mostly in the supergene zone replacing chalcopyrite and coating pyrite. Traces of primary chalcocite were found in drill core throughout the orebody. Minor galena and sphalerite occur in late quartz-carbonate veins that cut the deposit and surrounding rocks.

Geology of the Granisle Porphyry Copper-Gold Deposit

The Granisle deposit is an asymmetrically zoned classic porphyry copper-gold deposit spatially associated with multiple phase dike-like intrusions of the Babine Igneous Suite but exposed at a deeper stratigraphic level than the Bell deposit. Dikes of biotite feldspar porphyry (BFP) of the Tertiary continental Babine intrusions were emplaced in island arc rocks of the Jurassic Hazelton Group. Uplift by regional block faulting has exposed the base of the Hazelton Group and the root of the porphyry copper-gold system.

Hazelton Group on McDonald and Sterrett Islands

McDonald Island is largely underlain by marine volcanic and sedimentary rocks, included in the Kotsine subaqueous facies of the Sinemurian Telkwa Formation of the Lower Jurassic Hazelton Group (Tipper and Richards, 1976a). The volcanic and sedimentary succession has been divided (Fahrni et al., 1976) into a sedimentary/epiclastic member and a volcanic member. The central and eastern parts of the island are underlain by green to purple intermediate tuff and breccia with intercalated chert pebble conglomerate. These rocks strike northerly, dip moderately to the west and are apparently overlain in the western part of the island by massive and amygdaloidal andesitic flows and thinly bedded shales (Fahrni et al., 1976). Some of the oldest rocks exposed in the area are on Sterrett Island where the base of the Hazelton Group is marked by well-bedded fossiliferous shale, siltstone and conglomerate. Clasts within the basal conglomerate correlate elsewhere with the Triassic Takla and Permian Asitka groups which are inferred to underlie the Jurassic succession (Richards, 1988). It is possible that the chert clasts in the conglomerate exposed on McDonald Island are also derived from the Permian Asitka Group.

Porphyritic, amygdaloidal and fragmental andesite and intermediate ash tuff of the Hazelton Group are exposed in the Granisle pit. On the east side of the pit, intensely altered Hazelton rocks occur as inclusions and roof pendants in the quartz diorite. These rocks include strongly biotitized tuffs, minor flows and breccias. Hazelton Group volcanic and volcaniclastic rocks are generally dark grey, weak to moderately silicified and may contain magnetite. Less altered Hazelton rocks occur peripheral to the ore zone. The moderately soft, less altered rocks are multicoloured to pale yellow and exhibit weak to moderate sericite/clay alteration. Minor felsic flows consisting of scattered feldspar and quartz phenocrysts in a fine-grained pale yellow groundmass occur locally on the northwest side of the deposit.

Babine Intrusions on McDonald Island

Copper mineralization is associated with a series of Tertiary porphyritic intrusions that cut Lower Juras Iazelton rocks in the central part of McDonald Island (Fig. 8). Although several intrusive pulses may have occurred at the Granisle deposit (Kirkham, 1971), two main porphyry phases are identified. The two units have the same bulk composition but are differentiated by texture, intensity of alteration and cross-cutting relationships. The first intrusive stage is a northeasterly trending elliptical plug (Fahrni et al., 1976) of quartz diorite microporphyry. The northwestern margin of the quartz diorite plug is cut by a more coarse-grained biotite feldspar porphyry dike striking 055°. Smaller dikes and irregular masses of the later biotite feldspar porphyry also cut the older plug.

K-Ar age determinations on four biotite samples collected in and near the Granisle orebody yielded a mean age of 51.2 Ma ± 2 Ma (Carter, 1972). Three of these samples came from biotite feldspar porphyry (both mineralized and unmineralized) and the other from a quartz-chalcopyrite-bornite-apatite vein.

Quartz Diorite Microporphyry

Quartz diorite microporphyry is exposed on McDonald Island as an elliptical intrusive body with dimensions of 500 m by 300 m in plan. The rock is composed of 1 mm phenocrysts of zoned andesine feldspar and phenocrysts of biotite in a fine-grained to aphanitic groundmass of quartz, plagioclase and biotite (Fahrni et al., 1976). Primary magmatic amphibole grains have been completely altered to fine masses of secondary biotite (Carter, 1972). Hydrothermal biotite, disseminated in the groundmass, imparts a dark grey to black colour depending upon the amount of secondary biotite present. The feldspar phenocrysts, although locally euhedral, tend to be subhedral and somewhat indistinct, with crystal edges merging with the groundmass.

Although the quartz diorite generally has sharp contacts with the younger biotite feldspar porphyry, contacts with older units, mapped as porphyritic andesite, are indistinct or gradational. Because the lithologies are similar in appearance, it has been suggested that the quartz diorite may have formed in part either by recrystallization or by post-intrusion hydrothermal metasomatism of andesite (Fahrni, 1967). Alternatively, the andesite may represent a synvolcanic intrusion (Piteau and Martin, 1977).

Cross-cutting relationships indicate that the quartz diorite microporphyry predates the biotite feldspar porphyry. No age dates have been determined for the quartz diorite; hence, the degree to which it predates the biotite feldspar porphyry is uncertain.

Biotite Feldspar Porphyry (BFP)

Characteristically the BFP at the Granisle mine is a crowded porphyry, with phenocrysts of euhedral, relatively fresh plagioclase 2 mm to 5 mm in diameter, and 1 mm to 2 mm flakes and books of fresh brown biotite. Phenocrysts comprise 35% to 50% of the rock, with feldspar more abundant than biotite. The groundmass is a fine-grained to aphanitic mixture of quartz, plagioclase, biotite, potash feldspar and apatite (Fahrni et al., 1976) which varies from light to dark grey in colour. Outside the pit area the BFP is a uniform grey colour and contains minor hornblende phenocrysts in addition to the plagioclase and biotite phenocrysts.

Several phases of biotite feldspar porphyry overlap the period of mineralization. The most prominent of these is a northeast trending dike that cuts the northwest edge of the quartz diorite plug. The dike is light to dark grey and ranges in composition from quartz diorite to granodiorite depending on the amount of potash feldspar present, most of which is secondary (Carter, 1972). The dike averages roughly 120 m in thickness, increasing to 200 m thick in the pit.

Alteration in the Granisle Deposit

Biotite-Magnetite (Potassic) Alteration

A crudely oval zone of potassic (biotite-magnetite) alteration is coincident with, but of greater areal extent than the copper orebody (Fahrni et al., 1976). Within this zone, finely disseminated hydrothermal biotite colc Hazelton rocks and the quartz diorite microporphyry dark brown-grey to black. The rocks are weakly to intensely silicified and contain variable amounts of disseminated magnetite. Gypsum veinlets, where present, generally occur in biotite-altered rock. Phenocrysts in the microporphyry (especially biotite) tend to be obscured by alteration. Primary magmatic hornblende is replaced by fine-grained aggregates of secondary biotite. Magnetite tends to be most strongly developed near borders of the ore zone and occurs both as fine stringers and rims on biotite phenocrysts (Fahrni, 1967).

The younger BFP appears fresh in hand specimen and phenocrysts are essentially unaltered; however, the groundmass contains sparse to abundant light to dark brown secondary biotite uniformly distributed throughout the rock. Secondary potassium feldspar is also present within the ore zone but is of limited extent and detectable only by petrographic staining (Fahrni et al., 1976). It is fine-grained in the groundmass and also occurs as thin envelopes enclosing veinlets and fractures.

Carbonate-Sericite-(Quartz)-Pyrite Alteration

The central biotite/potassic alteration zone grades outward to a carbonate-sericite-quartz-pyrite zone which forms a partial ring around the deposit (Carson and Jambor, 1974). In this zone, intrusive and volcanic rocks weather to a bleached buff colour, mafic minerals are altered to a mixture of sericite and carbonate, and plagioclase phenocrysts are variably sericitized. Pyrite and marcasite are abundant, comprising 3% to 5% of the rock by volume and occurring as both disseminations and stringers. Although coarse-grained BFP locally bears ore grade mineralization, the sericite-carbonate zone is generally classified as waste.

Carbonate-sericite-pyrite altered rocks are exposed on the northwestern wall of the Granisle pit and crop out southwest of the pit. This alteration assemblage weathers with a bright yellow jarosite stain on exposed surfaces. Where marcasite is present, weathering is rapid, with the rock breaking down within the space of a few years to a yellowish mass of clay minerals.

Dacite and other felsic units, described in earlier publications, have been reclassified as carbonate-sericite altered rock. Some of these units, however, may be equivalent to the late-stage quartz-biotite-feldspar porphyry (QBFP/QFP) unit identified at the Bell mine.

Chlorite-Carbonate-Epidote (Propylitic) Alteration

Minor propylitic (chlorite-carbonate-epidote) alteration is exposed on the periphery of the pit and locally within blocks of waste in the main orebody. In the pit, weak propylitic alteration is indicated by a green tinge to the groundmass and/or feldspar phenocrysts accompanied by variable degrees of carbonate fracture filling.

Rocks outside the pyrite halo show varying degrees of propylitic alteration, with chlorite, carbonate and epidote as common constituents (Carter, 1972).

Structure in the Granisle Deposit

McDonald Island is bracketed to the southwest and to the northeast by two northwesterly trending block faults (Fig. 8). One crosses the west side of the McDonald Island and the other extends along the channel separating the island from the east shore of Babine Lake (Carter, 1972).

Between these block faults, a complementary set of faults striking predominantly 025° is exposed in the pit, with the most prominent faults dipping 57° to 75° east and secondary faults dipping steeply to the east and west. The major faults contain thick sections of gouge and breccia and have slickensides developed on the fault surfaces. Although displacement is often ambiguous, local offset relations show left-lateral displacement (Piteau and Martin, 1977). Copper bench-assay plans also suggest some vertical offset occurred. Dominant joint patterns are subparallel to the main fault patterns. The structural pattern indicates that the BFP intrusions at Granisle

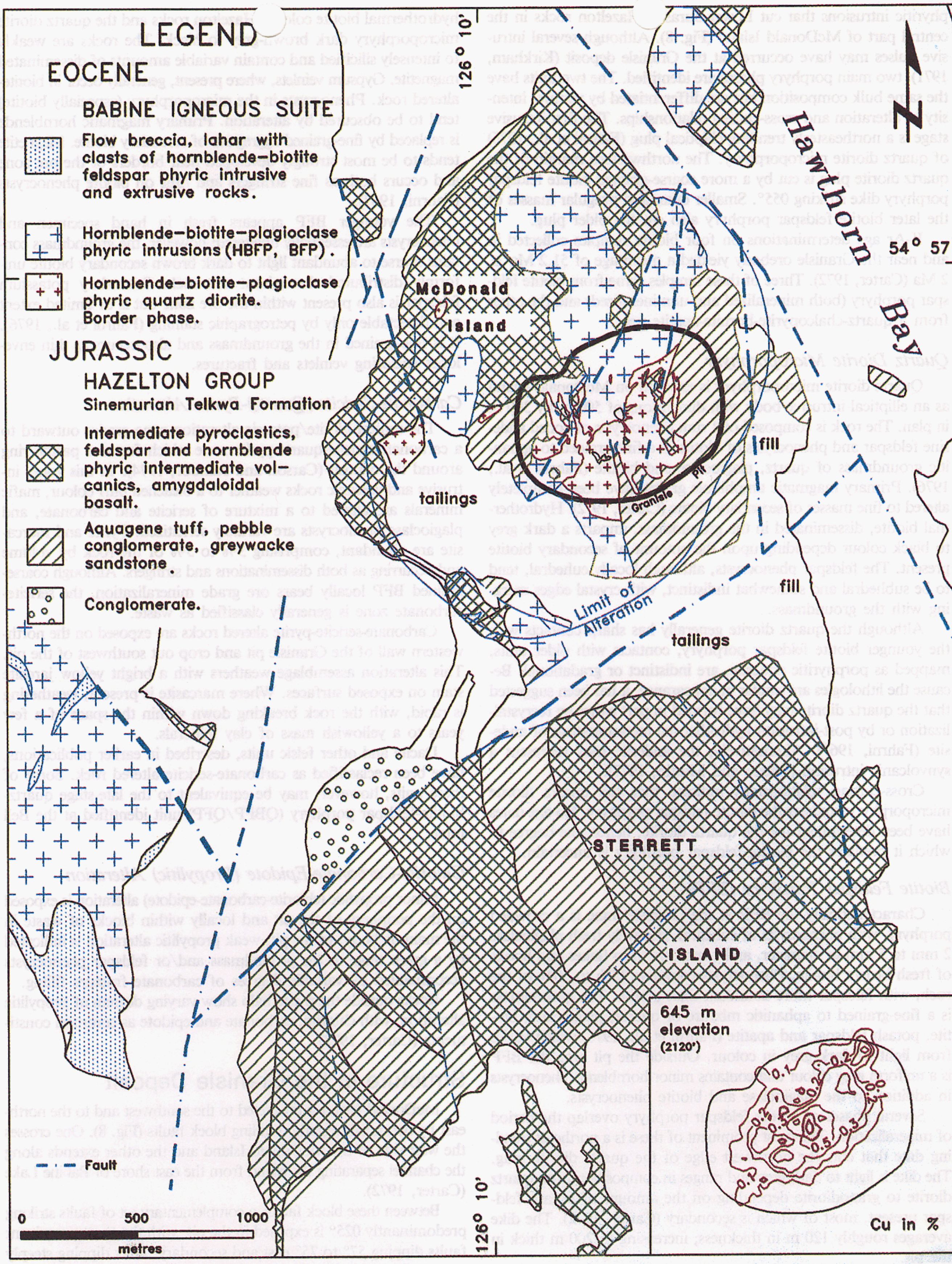


FIGURE 8. Geology of the Granisle porphyry copper-gold deposit. Compiled from Granisle Copper pit mapping and mapping by the authors. Inset compiled from diamond drilling and blast hole sampling shows distribution of copper grades at the 645 m elevation.

were emplaced in a transtensional dilatan .ıe between two right-stepping transverse faults subjected to dextral shear.

A north-northeast striking zone of quartz stockwork cuts across the main zone of biotite-magnetite (potassic) alteration. Within this stockwork, 60% of the stringers are oriented 285°/80°E; 30%, 010°/80°E; and 10% are horizontal (Fahrni, 1967).

Mineralization and Ore Zone

The major portion of the Granisle deposit is contained within an elliptical zone of pervasive biotite-magnetite (potassic) alteration with its long axis orientated northeast. The limits of mineralization, as defined by the 0.10% Cu isopleth, form a northeast trending elliptical body 800 m long by 450 m wide (Fig. 8). Symmetry of copper grade distribution is disrupted by two pipe-like bodies of waste. These waste zones form continuous vertical structures which flare slightly at depth. The zones are elliptical in plan with long axes roughly oriented 075° and 350°. The shape and distribution of these internal bodies of waste suggest that mining operations may have been nearing the bottom of the copper mineralized portion of the porphyry system at the time of closure in 1982.

The principal sulphide minerals in the Granisle ore zone are chalcopyrite, bornite and lesser amounts of pyrite. Visible sulphide content in the ore zone is low, generally comprising less than 3% of the rock by volume. Chalcopyrite is most widespread, occurring as disseminations both in the quartz stockwork and in the groundmass. Bornite is most concentrated in the southern half of the ore zone and was more abundant in the upper portions of the orebody (Fahrni, 1967). The average head grade over the life of the Granisle operation was 0.47% Cu.

Gold occurs as grains of electrum ranging in composition from 81.5% to 88.5% Au, with the balance being silver (Cuddy and Kesler, 1982). The grains of electrum are 10 to 60 microns across and are found in grains and on selvages of bornite. Recovered gold grade at Granisle was less than that of Bell, averaging 0.13 g/t milled. Annual production records indicate a significant decrease in recovered gold grade with depth. Average head grade has been estimated at 0.2 g/t.

The average recovered silver grade at Granisle was 1.3 g/t and, like gold, appeared to decrease with depth. The average head at Granisle has been estimated at 2 g/t Ag.

Molybdenite, galena and sphalerite occur in small amounts in the Granisle deposit. Drill core and blast hole sampling in 1977 and 1978 indicated that molybdenum was distributed in a more or less annular zone surrounding the high-grade copper zone. Average grade within the 0.005% Mo isopleth was estimated to be 0.010%. Rising prices for molybdenum in 1978 to 1980 encouraged the installation of a molybdenum recovery circuit which was completed in 1980. Recovery of molybdenum was abandoned shortly thereafter due to metallurgical difficulties arising from the erratic distribution of molybdenite within the deposit and the significant drop in price after 1980.

Although ore may be present in any of the mine lithologies, it is more dominant in the quartz diorite microporphyry. Much of the ore grade material is associated with a quartz stockwork that parallels the main axis of the biotite-magnetite (potassic) zone; however, ore grades also occur in rocks containing only disseminated copper minerals and lacking the quartz stockwork.

Fluid History, Temperature and Depth of Emplacement of the Babine Porphyry Copper-Gold Deposits

Strongly saline magmatic-hydrothermal brines were instrumental in the early evolution of the Babine porphyry copper-gold deposits. Magmatic waters were essential to the development of the central potassic biotite-magnetite-chalcopyrite core zones and peripheral propylitic (chlorite-carbonate) zones at Bell, Granisle and Morrison. Increasing dilution of magmatic-hydrothermal brines with meteoric waters led to the development of sericite-carbonate-pyrite

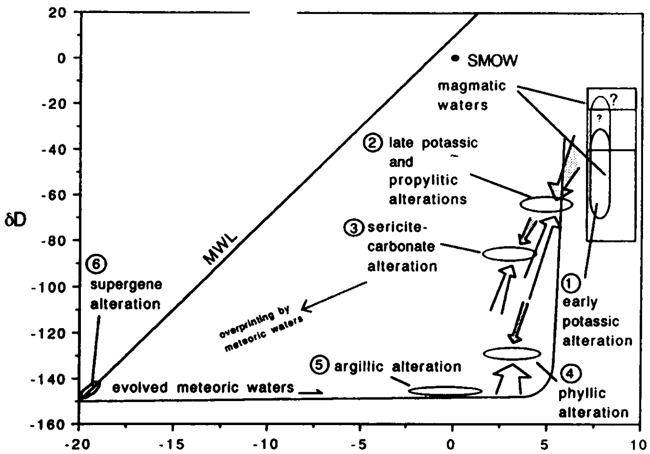


FIGURE 9. Fluid sources and hydrothermal evolution of the Babine porphyry copper deposits (from Zaluski, 1992).

alteration at both Bell and Granisle and an overprint of quartz-sericite-pyrite stockwork at Bell (Wilson et al., 1980).

At 55° North latitude, Granisle and Bell are the most northerly porphyry copper mines in the world. The stable heavy isotopes of hydrogen and oxygen (D and ^{18}O) in rain and snow become progressively more depleted in relation to Standard Mean Ocean Water with increasing latitude and increasing altitude. Depletions are recognized by low or negative values for δD and $\delta^{18}O$, with more negative values indicating greater depletion. In addition, meteoric waters are significantly more depleted near the west coast of North America. For these reasons, meteoric waters in the north-central Cordillera have δD and $\delta^{18}O$ values of approximately -150 and -20 per mil, respectively, among the lowest on the continent. Pristine magmatic waters are less depleted than meteoric waters in the northern Cordillera. Isotopic values are in a relatively narrow range of -50 to -86 for δD and $+5$ to $+10$ for $\delta^{18}O$ (Guilbert and Park, 1986). The isotopic contrast between meteoric and magmatic waters for the Babine deposits will, therefore, be greater than for most other deposits in North America. This contrast makes the Babine deposits particularly amenable to studies involving the mixing of magmatic and meteoric waters in hydrothermal alteration.

Calculated isotopic values of waters in equilibrium with biotite at 541°C in the biotite-magnetite (potassic) zone at Bell have a mean δD_{H_2O} of -86 per mil and a mean $\delta^{18}O_{H_2O}$ of 7.2 per mil (Zaluski, 1992), indicating they were primarily of magmatic origin. Indicated compositions for waters in the outer propylitic zone yield a δD_{H_2O} of -112 per mil and $\delta^{18}O_{H_2O}$ of 7.4 per mil. The lower value for δD in the propylitic zone suggests depletion of D from mixing of magmatic waters with a lesser component of isotopically evolved lighter meteoric waters. Nonetheless, magmatic waters appear to have dominated the hydrothermal system during the formation of the initial central potassic and peripheral propylitic alteration zones.

Strongly saline potassium-enriched chloride brines of mainly magmatic origin were responsible for potassium alteration and copper-gold mineralization at Granisle. Isotopic compositions for waters in equilibrium with biotite in the biotite-magnetite (potassic) zone at Granisle were similar to those at Bell. Calculations of fluid compositions at 541°C yield δD_{H_2O} ranging from -47 to -95 per mil and $\delta^{18}O_{H_2O}$ of 7.1 to 8.2 per mil. Temperatures, however, were higher at Granisle than at Bell. Potassic alteration and ore at Granisle formed at temperatures ranging from 400°C to over 800°C (Wilson et al., 1980) from potassium-enriched fluids (K/Na >0.2) having a mean salinity of 64.7%. Fluid inclusions contain liquid + vapour + halite + sylvite ± an unidentified K-Fe-Cl phase, in contrast with inclusions from the Bell deposit which lack sylvite. Isotopic data from the propylitic zone also indicate predominantly magmatic waters.

Mixing of magmatic waters w a component of evolved meteoric waters appears to have been responsible for sericite-carbonate alteration at both Bell and Granisle (Zaluski, 1992; Fig. 9).

Mixing of isotopically lighter meteoric waters with strongly saline magmatic waters under acidic reducing conditions was responsible for the quartz-sericite (phyllic) overprint at the Bell deposit. Waters in equilibrium with mineral separates in the quartz-sericite (phyllic) overprint at Bell are isotopically lighter than magmatic waters (Zaluski, 1992). Calculated fluid compositions at 400°C range between 1.8 and 4.6 per mil for $\delta^{18}O_{H_2O}$ and between -101 and -130 per mil for δD_{H_2O} in the quartz-sericite zone. Fluids trapped in inclusions in the quartz-sericite stockwork at Bell have mean salinities of 66.1% NaCl equivalent, K/Na <0.2, and homogenize by halite disappearance after disappearance of vapour between 289°C and 619°C (Wilson et al., 1980). Vapour-rich inclusions are most abundant, with a lesser number of inclusions filled with liquid + vapour + halite or liquid + vapour + halite + an unidentified K-Fe-Cl phase. Estimates for minimum pressures at which the stockwork was developed range between 50 bars to 100 bars, which translates into a minimum depth of emplacement of 150 m to 300 m for the Bell deposit (op. cit.). Formation of the quartz-sericite stockwork appears to have involved a mixture of magmatic and meteoric fluids (Zaluski, 1992), with an increase in salinity from evolution of vapour during boiling. Acidic reducing conditions prevailed during deposition of the quartz-sericite stockwork. Calculated maximum pH for aqueous fluids in the quartz-sericite stockwork was 3.8, assuming K/Na <0.2. The rocks acquired a bleached white colouration as silicates were altered to white sericite and oxides were reduced to sulphides.

Postulated Genesis of the Bell and Granisle Deposits

Emplacement of the Babine Intrusions

An easterly-dipping subduction zone developed along the western margin of the northern Cordillera in Late Cretaceous to Early Tertiary time. Low degrees of partial melting (around 10%) of mantle lithosphere at depth beneath the Stikine terrane led to the formation of a hydrous copper-gold bearing alkaline melt which rose to underplate the Stikine terrane. Back arc extension and transtensional faulting in the Babine region tapped the melt which then rose along the deep-seated fractures. Assimilation of the island arc rocks of the Stikine terrane and fractionation of hornblende from the ascending hydrous melt led to the evolution of a bimodal calc-alkaline suite of hornblende andesites and rhyodacites known as the Babine Igneous Suite. Fluctuating water pressures from the repeated breaching of magma chambers led to the development of complex zoning patterns in plagioclase. Copper partitioned selectively into residual phases facilitated by high $Al_2O_3/(Na_2O + K_2O + CaO)$ ratios (Feiss, 1978). The Morrison Fault and Morrison Graben, together with the offset Newman Fault, served as a locus for the emplacement of the intrusions. Multiple magmatic pulses formed an Eocene stratovolcano at Bell at the intersection of northwesterly trending normal faults and easterly trending cross faults (Figs. 10a, 10b). Similarly, multiple intrusions were emplaced as dikes in a dilatant zone between two northwesterly trending faults at Granisle.

Formation of the Bell and Granisle Zoned Porphyry Copper-Gold Deposits

Circulating magmatic-hydrothermal waters led to the development of zoned porphyry copper-gold deposits at Bell (Fig. 10c) and Granisle with central biotite-magnetite chalcopyrite mineralization mantled by propylitic haloes of chlorite and carbonate. At Bell, pervasive brittle failure shattered the BFP and rhyodacite in the western part of the deposit, caused either by release of energy upon second boiling and decompression of fluids released from crystallizing magma (Burnham, 1985) or from an influx of cooler meteoric waters. Circulation corrosive mixed meteoric and magmatic hydrothermal brines leached copper and gold from the biotite-magnetite zone. Precipitation of sulphides from strongly saline chloride brines increased $[H^+]$ and $[Cl^-]$ ion concentrations and lowered pH. Acidic, reducing, copper- and gold-enriched brines circulated through the shattered highly permeable BFP and rhyodacite subsequently bleaching and altering the rocks. Chalcopyrite and electrum were deposited with pyrite in quartz stockwork that healed the fractures in the bleached and broken rocks.

Bell Deposit: Explosion, Collapse and Intrusion of QFP

As magmatic and hydrothermal activity was waning in the Bell deposit after the formation of the quartz stockwork, continued influx of water (possibly of meteoric origin) interacted with hot rock or magma. A subsequent phreatomagmatic blast removed the upper part of the southeastern quadrant of the orebody and produced a crater which was partially filled by coarse vent breccias (Figs. 10d, 10e). In the core of the deposit, gravity-induced vertical tensional stresses caused by removal of support led to brittle failure in the central BBFP. Sheeted horizontal tension fractures propagated through the BBFP with spacings of 1 cm to 4 cm.

A final pulse of magmatic activity backfilled the crater produced by the explosion with an upward-flaring intrusion of quartz ± biotite-feldspar porphyry (QFP) (Fig. 10e). Massive QFP is exposed in the Bell pit where it intrudes shattered BBFP (Fig. 3). The porphyry system lacked sufficient energy in its waning stages to induce crackle brecciation in the QFP intrusion. However, hydrothermal activity continued. Hydrothermal fluids associated with the QFP caused pervasive internal sericite-carbonate-pyrite alteration and were responsible for the development of patches of argillic overprinting in BBFP. The late-stage fluids were sufficiently corrosive to form a solution cavity along the northern contact of the QFP which was filled by the northeast collapse breccia (Fig. 4). Narrow vents bored by a final episode of gas streaming were partially filled with pebble breccias. Continued subsidence due to removal of support from gas venting or magma withdrawal led to formation of collapse structures bracketed by inward-dipping normal faults (Fig. 10f).

The deeper Granisle deposit, undisturbed by similar subvolcanic events, was preserved as a classic zoned porphyry copper-gold deposit. The biotite-magnetite-chalcopyrite-bornite assemblage in the orebody was emplaced at a depth which was near the lower limits of meteoric water circulation. A small component of deeply circulating hydrothermal fluids, however, lead to the development of an incomplete halo of carbonate-pyrite-sericite alteration.

Structural Adjustment

Movement along major regional structures led to uplift of the McDonald Island block. The base of the Hazelton Group and the roots of the Granisle deposit were exhumed. Unroofing was most probably from erosion; however, the deposit may also have been dismembered by faulting in a manner similar to the Morrison - Hearne Hill deposits (Ogryzlo et al., this volume). The Bell deposit, in contrast, remained relatively protected from post-Eocene erosion in a down-dropped fault block. Oxidation and weathering of the upper portions of the deposit produced a blanket of supergene enrichment. Quaternary glaciation scoured out the subsidence-shattered BBFP core of the Bell deposit, forming a central crater-like depression. Up to 40 m of varved glaciolacustrine clays were deposited in the depression, along with spruce, fir and dwarf birch trees and at least one woolly mammoth. Radiocarbon dates of 42 900 ± 1860 yr B.P. and 43 800 ± 1830 yr B.P. were obtained on wood associated with the mammoth bones which were dated at 34 000 ± 690 yr B.P. (Harington et al., 1974). The ore deposits remained thus until their discovery, with the exposed Granisle deposit eroding into the Babine Valley and Bell protected under its blanket of impervious clay.

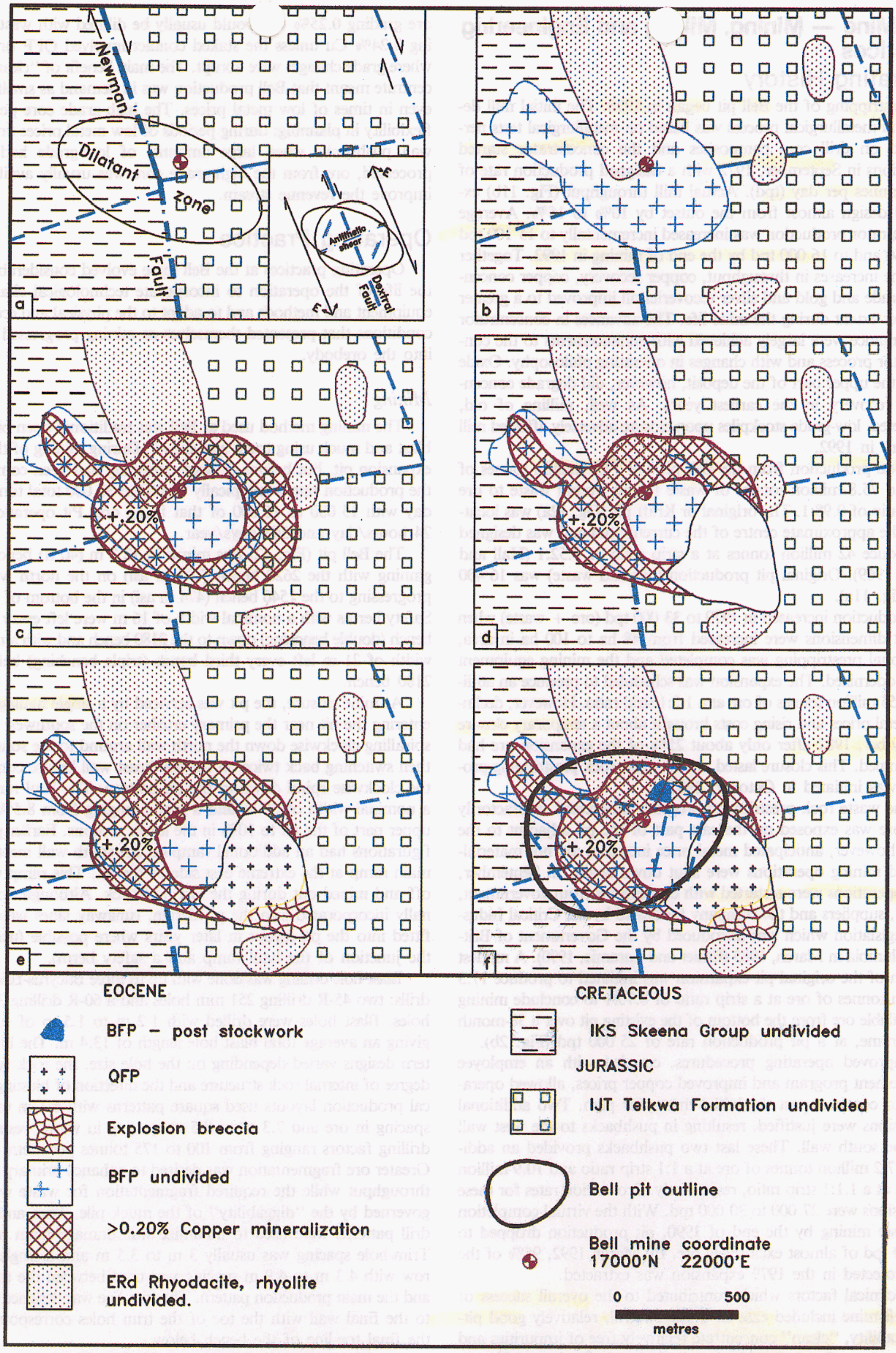


FIGURE 10. Proposed structural model for the Bell porphyry copper-gold deposit. a: Faulting — extension accompanied by dextral shear along Newman Fault creates a dilatant zone. Stress diagram modified from Dennis (1987). b: Intrusion — emplacement of Eocene BFP and rhyodacite in the dilatant zone and along the trace of Newman Fault. c: Mineralization — formation of a zoned annular porphyry copper-gold deposit. d: Blast — explosive reaction from the interaction of meteoric fluids with magma removes southeastern quadrant of the deposit. e: QFP intrusion — the vent is backfilled with explosion breccia and post-stockwork QFP. f: Subsidence and collapse — inward-dipping normal faults bracket the deposit. Emplacement of post-stockwork BBFP dike.

Bell Mine — Mining, Mil. ȷ and Engineering Practices

Operating History

Prestripping of the Bell pit began in 1970. The initial mill design and metallurgical process was based on metallurgical tests performed on drill core composites and the concentrator started operations in September, 1972 with a designed production rate of 9100 tonnes per day (tpd). Actual mill throughput (Fig. 11b) exceeded design almost from the outset by 10% to 15%. Average concentrator production was increased incrementally to 15 100 tpd by 1980 and to 16 000 tpd by the end of mining in 1992. Together with the increases in throughput, copper recovery, copper concentrate grade and gold and silver recoveries all improved to a greater or lesser extent during the mine life. The advances in concentrator performance were largely achieved with enhancements to the concentrator process and with changes in operating philosophy. Oxide ore in the upper part of the deposit, however, did degrade concentrator recovery in the earliest years. As well, milling of old, weathered, low-grade stockpiles upon closure adversely affected mill recovery in 1992.

Total production from the Bell pit was 77.2 million tonnes of ore and 75.8 million tonnes of waste for an overall waste to ore strip ratio of 0.98:1. The original or Kraft pit (Fig. 12a) was located in the approximate centre of the current void and was designed to produce 42 million tonnes at a strip ratio of 0.52:1 (Hall and Kraft, 1969). Original pit production (ore and waste) was 16 400 tpd (Fig. 11a).

Production increased in 1979 to 33 000 tpd (ore + waste) when the pit dimensions were increased from 48 ha to 100 ha in area, additional prestripping was completed and the mining equipment was modernized. The expansion was scheduled to produce an additional 55 million tonnes of ore at a 1.2:1 strip ratio. However, declining metal prices and rising costs brought about a temporary closure in October, 1982 after only about 25% of the expansion ore had been mined. This closure lasted until a waste rock prestripping program was initiated in October, 1983.

The waste rock prestripping program progressed satisfactorily until ore was exposed on the east part of the pit adjacent to the QFP, however, anticipated metal price increases did not materialize and mining operations were shut down again. In September, 1985 operations were restarted with concessions from government, labour, suppliers and the company facilitated by the Critical Industries legislation which was introduced by the Government of British Columbia in March, 1985 (Ethier and Laramie, 1990). A modest version of the original pit expansion was initiated to produce 17.3 million tonnes of ore at a strip ratio of 0.75:1 to conclude mining of available ore from the bottom of the existing pit over a 38-month time frame, at a pit production rate of 25 000 tpd (Fig.12b).

Improved operating procedures, coupled with an employee involvement program and improved copper prices, allowed operations to continue past the initial three-year plan. Two additional expansions were justified, resulting in pushbacks to the west wall and the south wall. These last two pushbacks provided an additional 7.2 million tonnes of ore at a 1:1 strip ratio and 10.9 million tonnes at a 1.1:1 strip ratio, respectively. Production rates for these expansions were 27 000 to 30 000 tpd. With the virtual completion of waste mining by the end of 1990, pit production dropped to 16 500 tpd of almost exclusively ore. By March 1992, 95% of the ore projected in the 1979 expansion was extracted.

Technical factors which contributed to the overall success of the Bell mine included ease of grade control, relatively good pit-wall stability, "clean" concentrate relatively free of impurities and the presence of a core of higher grade mineralization within the deposit. With gold accounting for 20% to 35% of the mine revenue, the income stream was also partially buffered against economic cycles. The homogenous nature of the ore meant that sampling and daily planning were relatively simple. Dilution was seldom a problem: if dilution did occur across a staked waste/ore contact, ore grading 0.25% ‹ ‹ould usually be diluted with waste grading 0.24% Cu unless the staked contact involved QFP or BBFP where grade changes were abrupt. The main benefit of "clean" concentrate meant that Bell production was in demand as smelter feed even in times of low metal prices. The high-grade core permitted flexibility in planning: during periods of low metal prices or during wall pushbacks when large amounts of low-grade had to be processed, ore from the high-grade core was usually available to improve the revenue stream.

Operating Practice

Operating practices at the Bell mine evolved considerably over the life of the operation to incorporate technological changes in equipment and methods and to adapt to the physical and economic conditions that presented themselves as mining progressed deeper into the orebody.

Mining

The mining method used at Bell was traditional open pit: drill, blast and muck using truck and shovel. During mining of the final expansion pit, but before all waste stripping had been completed, the production rate was typically 27 000 to 30 000 total tonnes per day with 15 000 to 16 000 of that being ore. Pit operations ran 24 hours/day and 365 days/year.

The Bell pit (Fig. 12) was mined in 12.2 m (40 ft) benches beginning with the 2620 bench (798 m asl) on the north wall and progressing to the 1540 bench (474 m asl) in the bottom of the pit. Safety berms with a nominal width of 16 m were left every second bench (double benching) down to the 2180 bench and with a nominal width of 21 m left every third bench (triple benching) below the 2180 bench.

At mine closure, the pit was serviced by a single haulage ramp entering the pit near the primary crusher on the southwest corner, spiralling clockwise down the north wall around to the south wall, then switching back twice on the southeast wall before continuing the clockwise spiral down to the pit bottom. The haul ramp had a nominal width of 30 m and a grade ranging from 8.5% in the upper part of the pit to 10% in the lower sections. Earlier pit configurations had an additional ramp on the south wall meeting the main ramp at the extreme east side of the pit. This ramp was cut off and mined out during the last pushback. Although not originally incorporated into the pit design, runaway lanes were retrofitted into the pit layout in later years where possible (usually at the junction of the haul ramp and a safety berm).

Blast hole drilling was done with up to three Bucyrus-Erie rotary drills: two 45-R drilling 251 mm holes and a 60-R drilling 311 mm holes. Blast holes were drilled with 1.2 m to 1.5 m of subgrade giving an average total blast hole length of 13.4 m. The blast pattern designs varied depending on the hole size, the rock type, the degree of internal rock structure and the direction of blasting. Typical production layouts used square patterns with 6.1 m to 7.3 m spacing in ore and 7.3 m to 8.5 m spacing in waste, resulting in drilling factors ranging from 100 to 175 tonnes per metre drilled. Greater ore fragmentation was desired to enhance primary crusher throughput while the required fragmentation for waste was only governed by the "diggability" of the muck pile. Trim and buffer drill patterns were used to minimize wall damage from blasting. Trim-hole spacing was usually 3 m to 3.5 m and a single buffer row with 4.3 m to 4.9 m spacing was used between the trim row and the main production pattern. The trim line was designed parallel to the final wall with the toe of the trim holes corresponding to the final toe-line of the bench below.

Blasting was primarily done using bulk ANFO at powder factors averaging 0.25 kg/t for ore and 0.20 kg/t for waste. Damp holes were blasted using plastic hole liners where possible and pumpable bulk emulsion was used in extreme water inflow conditions (common in the pit bottom). Emulsion averaged 10% to 15% of total explosive consumption. Production holes were loaded with 5

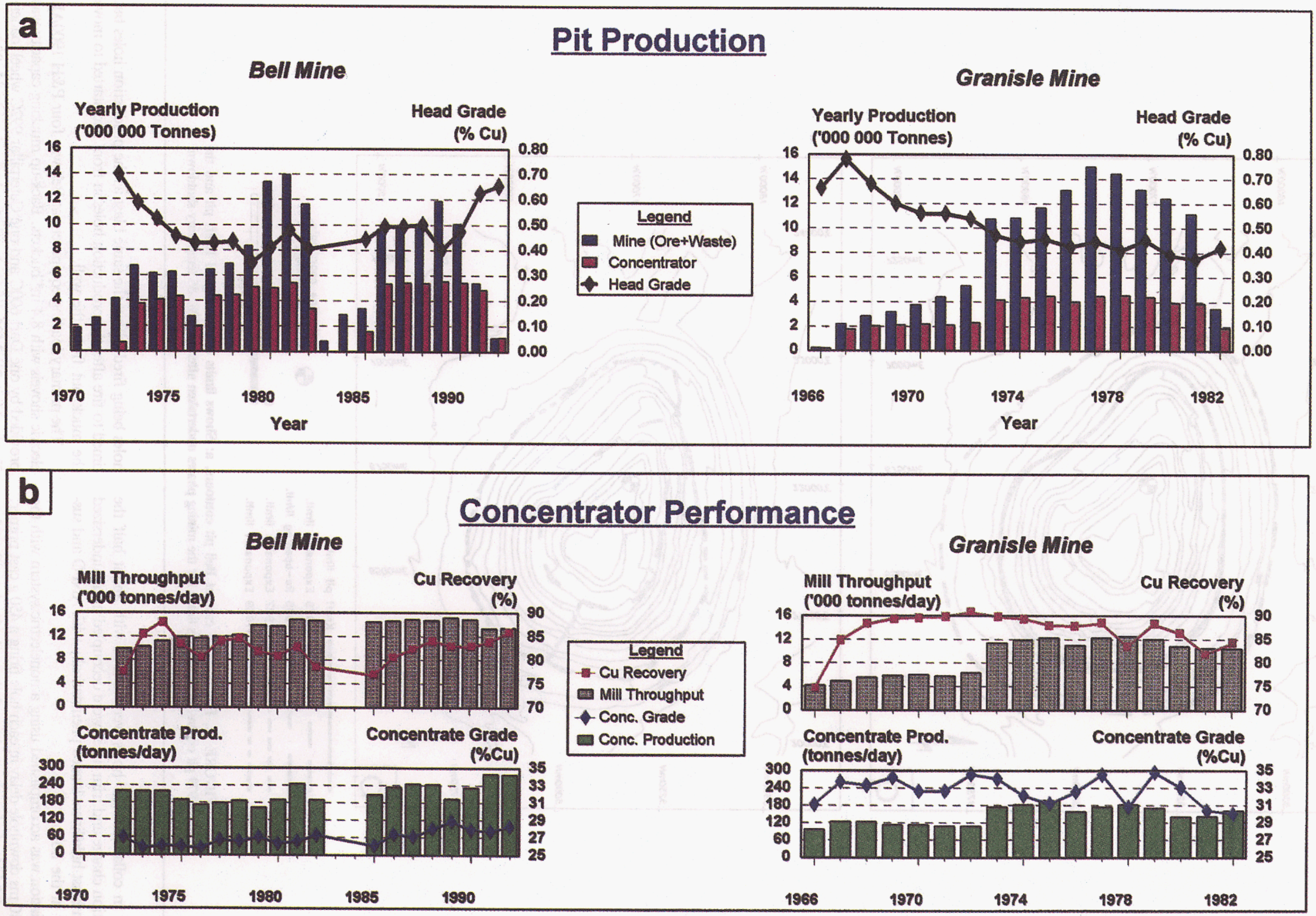


FIGURE 11. Production history at Bell and Granisle. a: Total production — note Bell shutdowns in 1976 for a prolonged strike and in 1982-85 for financial reasons; Bell steady increase in mill throughput and Granisle mill expansion in 1973. b: Concentrator performance — note higher copper recovery and copper grade in concentrate for Granisle mill due to higher bornite:chalcopyrite ratio.

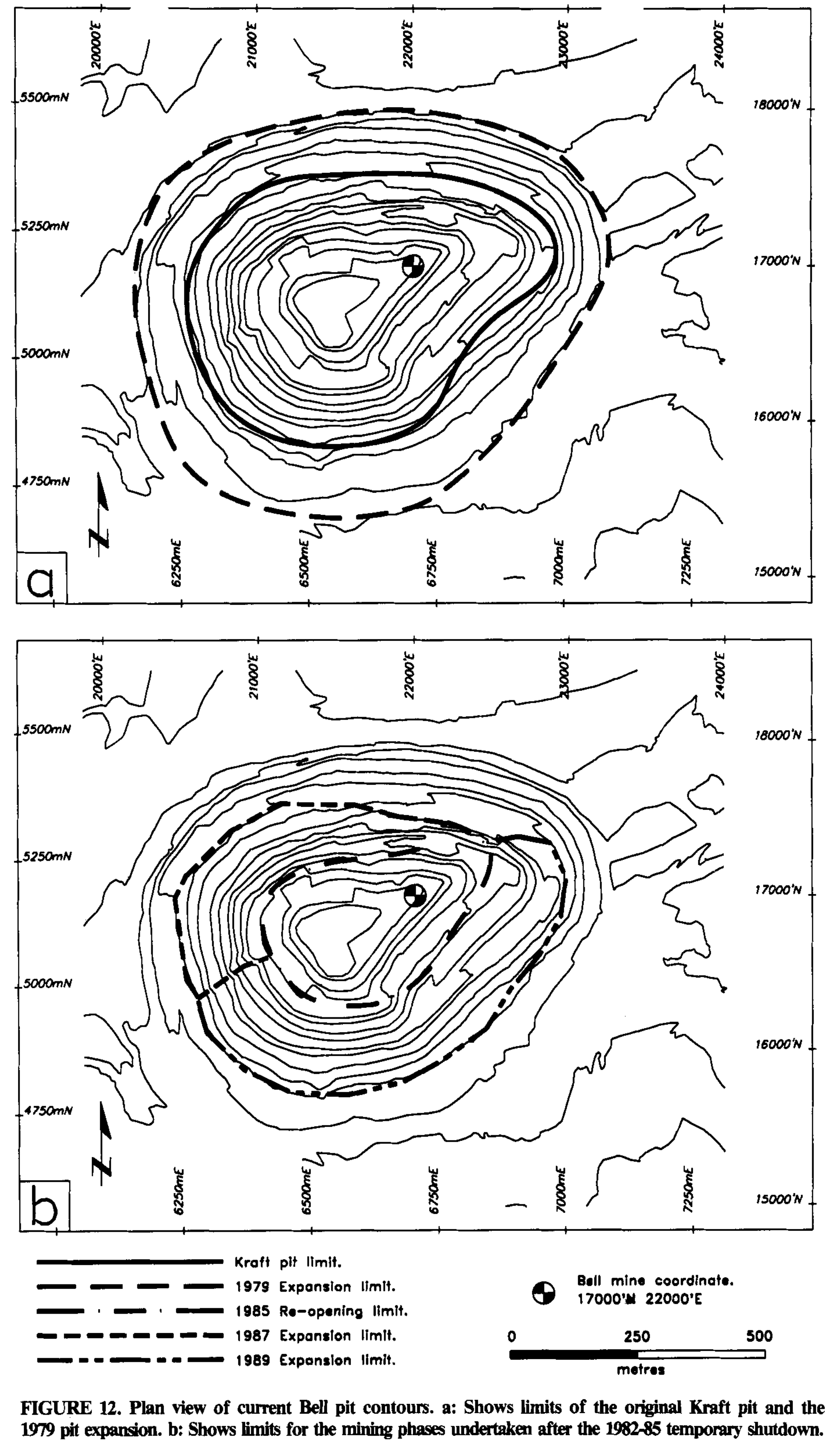


FIGURE 12. Plan view of current Bell pit contours. a: Shows limits of the original Kraft pit and the 1979 pit expansion. b: Shows limits for the mining phases undertaken after the 1982-85 temporary shutdown.

m to 6 m collars, buffer holes were loaded with about half the production charge and trim holes were decoupled using undersized 115 mm blast hole liners loaded pneumatically with ANFO and suspended in the hole.

Initiation was accomplished using a nonelectric system with the same 410 ms downhole delay in each hole firing a 450 g cast primer. Differential hole timing was achieved through the use of surface delays: commonly 35 ms between holes and 100 ms between rows. Firing sequences were "V" or "double-V" with the buffer and trim holes being fired within the same blast as the production holes but timed to fire after the production holes in front had started to move the muck out from the wall.

The primary loading equipment at Bell was four P&H 1900AL electric shovels with 8.4 m^3 buckets. Back-up mucking capacity was provided by one Dart 600C and one Caterpillar 992C wheel loader. Twelve Unit Rig M85 diesel-electric trucks equipped with 90 tonne boxes and oversized power trains were used to haul both ore and waste. The haulage fleet was expanded for a period in 1990-91 by

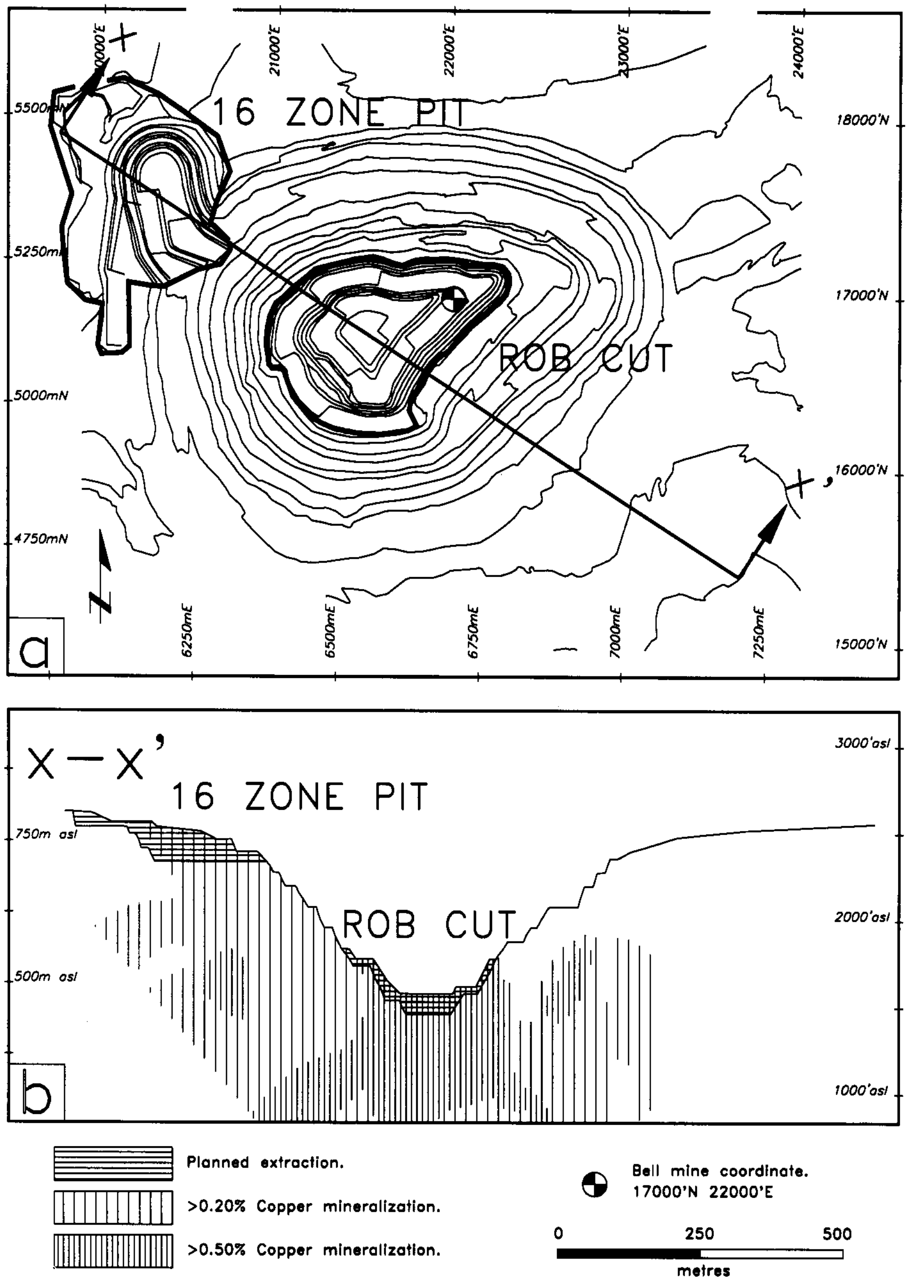


FIGURE 13. Plan and true section views of planned Rob Cut and 16-zone mining. This work was planned to occur in 1992 and 1993, however, pit wall conditions in the Rob Cut area forced cessation of work. Note the planned extraction of high-grade ore trapped in berms in the Rob Cut area created extended high-wall sections but not exceeding the pit-slope angles already existing higher on the same walls.

the addition of three 135 tonne capacity Caterpillar 785 mechanical drive trucks which were used exclusively on waste haul during a period of high stripping.

Ore was hauled either directly to the primary crusher or was deposited on the mill-feed stockpile immediately adjacent to the crusher. Waste was hauled to one of the large waste dumps on the north or east of the pit and was also used in the construction of the very large tailings dams south of the pit.

Overburden stripping was carried out by one of two methods. Contract scrapers were used to strip the glacial till when the total depth was greater than the 12.2 m bench height. When the overburden was less than a bench in depth and an adequate rock footing could be found, the overburden was stripped using the primary mining fleet of shovels and trucks.

In addition to the primary mining equipment mentioned above, there were numerous pieces of ancillary equipment which were used in a support capacity at the mine: two Caterpillar 14G graders for road maintenance; four D9-size dozers for dump maintenance, dam construction and safety berm cleanup; one rubber-tired dozer for shovel cleanup; one converted 55 tonne haul truck and one converted scraper hauling water for dust suppression; and two small excavators for berm cleanup and dam and seepage pond construction.

Rob Cut

In the fall of 1991, government approval was sought and conditionally granted to attempt the mining of an additional 3.6 million tonnes of high-grade ore at Bell below the pit design, in a project known as the Rob Cut (Fig. 13). The high-grade ore was to be supplemented with 2.7 million tonnes of low-grade ore from the top of the 16 zone to provide an additional 14 months of mill feed.

The plan was to mine out two berms starting with the 1740 berm and later the 1700 berm and, using tl   tra operating room gained on the pit floor, to extract an additional bench and a half below the current pit bottom (Fig. 13b). In any pit, the average pit-wall slope angles tend to flatten toward the pit bottom as haul ramp spirals become tighter with decreasing pit circumference. The Rob Cut plan did not increase the overall designed pit slope beyond suggested limits but did result in local sections of the pit wall with 73 m vertically between catchment berms or twice the usual height at Bell.

The Rob Cut work began in the winter of 1991 and required the use of innovative approaches and technology. Contract drilling was done using underground style down-hole hammer drills drilling angled 165 mm blast holes up to 42 m in length. Bulk delivery of emulsion explosives was also done with a unit initially designed by Imperial Chemical Inc. (ICI) for use in underground mines. New wall scaling techniques were also developed. The best method involved the use of a spiked steel weight called a "pineapple" suspended from a counterweighted pipe-laying side boom mounted on a D6 bulldozer working from a higher safety berm.

Unfortunately, the winter of 1991-92 was exceptionally mild in northern British Columbia with a number of successive freezing and thawing cycles. These conditions made it impossible to ensure the safety of operators working under the high walls as small pieces of loose rock were continually spalling and falling to the work areas below. The Rob Cut effort was abandoned in February, 1992 despite the completion of two successful longhole blasts.

Geotechnical Considerations

The primary geotechnical work at Bell involved determining the design pit-wall angles, recommending remedial action when problems were encountered and developing monitoring strategies that could be used to track pit wall performance. The geometry of drilling patterns, the limitations of drilling equipment and the overbreak character of Bell rock resulted in a fairly standard interberm wall angle of 69° (4.6 m backbreak per 12.2 m bench). Depending on rock mass characteristics, the designed overall pit wall slopes were steepened or flattened by adjusting the nominal berm widths and by the location of haulage ramps.

A considerable amount of structural wall mapping was done and the location, number, attitude, moisture conditions, infilling and continuity of structural features entered into a computer database to facilitate geotechnical interpretations. In addition, a limited amount of uniaxial compressive strength (UCS) testing was done to characterize the strength of the different rock types encountered at the mine. Results of the UCS work indicated strengths ranging from 55 Mpa to 175 Mpa. Using the data gathered, the deposit was divided into a number of structural domains which corresponded to zones of similar rock type and strength and containing discrete joints sets and/or faults. Slope design criteria were established for each domain, based on overall rock mass failure mode analysis and on structurally controlled failure envelope analysis. The criteria were specified with an objective of creating a factor of safety of 1.3 to 1.5 for overall slope stability and factors of safety of 1.1 to 1.2 for bench-scale stability.

One of the prime features that governed pit design was the structural domain within the "shattered zone" on the lower southeast side of the pit. This low rockmass strength domain was a zone of BBFP in the waste core, with numerous, tightly spaced horizontal tension fractures and cut by vertical and 60° dipping joints at regular intervals, giving a "sugar cube" appearance. The pit design counteracted the structural weakness of this domain by switching the haulage ramp back and forth across the southeast wall resulting in overall pit wall slopes of 34° to 36° in this area. The comparatively shallow pit slopes in this zone gave the Bell pit its characteristic kidney shape in plan with a re-entrant wall to the southeast.

In contrast to the weak shattered zone, the highest structural stability at Bell was found in the north wall rhyodacite zone and in the silicified core ore zone. These domains were characterized by high rock strengths and a relative absence of adverse structural features. Inter-ramp . angles in these zones were 50° to 52° and overall wall angles were 48° to 50°.

The remaining structural domains at Bell had an intermediate degree of stability. These areas include the east, west, southwest and upper southeast walls. Rock strengths for these areas were moderate to high and slope stability was primarily a function of the prevalent structural features. All these zones were characterized by adverse structural features striking parallel or subparallel to the pit walls in plan and dipping in toward the pit centre at 45° to 60°. Overall wall angles for these zones were typically 44° to 46°.

The general practice to stabilize overburden slopes above the pit rock-walls was to reslope to 20° to 25° and to establish safety berms at the rock-overburden interface. As well, ditches were dug back from the overburden crests to redirect surface water away from the till slopes.

In comparison with other open pit porphyry operations in North America, the wall stability at Bell mine was good. Bench-scale failures did occur in a number of areas, especially in the upper southeast wall of the pit. These small failures typically did not pose a great problem after they were cleaned up, except to occasionally cut off access to safety berms.

Specific structural features which caused local problems were the "haul road failure" in the northwest corner of the pit directly beneath the main North ramp, the south wall failure at the rock-overburden contact and a structural wedge in the northeast corner above the conjunction of the North and East ramps. Corrective measures taken to lessen the impact of these problems included resloping above the failure areas and stepping the walls in below the failures to decrease the slope angles locally. Additional remedial steps were taken on a case-by-case basis to proactively reduce the possibility of wall failures. These included regular wall scaling, safety berm cleanup and the installation of some horizontal drain holes in specific areas, to dewater potential wedges. The structural wedge in the northeast corner required some redesign of the main haulage ramp leaving a rock buttress on the toe of the wedge.

Despite being located on a peninsula in Babine Lake and with the pit bottom approximately 230 m below the elevation of the lake surface, the groundwater inflow rates at Bell were relatively moderate. Inflow rates averaged 1000 litres per minute over the year and peaked at about 2000 litres per minute during spring runoff. In comparison, the Brenda mine near Peachland, located above 1500 m asl and quite distant from any large bodies of water, experienced average inflow rates four times greater than those at Bell. The low inflow rates at Bell could be attributed to low overall rock mass permeability and the lack of continuity of major unhealed structural features. At the Granisle mine, inflows were two to three times higher than those at Bell.

Pit Slope Monitoring

A sophisticated pit slope monitoring system was developed and implemented at Bell in 1991. Previous pit slope monitoring had been carried out on an *ad hoc* basis using industry standard procedures: survey prisms installed in areas experiencing problems were monitored on a daily basis, or as required, until activity had subsided to normal levels. Monitoring requirements increased as mining progressed: more wall was exposed, existing walls became weathered and pit slope angles approached practical limits. The existing practice was useful only during daylight hours and often required additional survey labour outside of regular hours to maintain adequate coverage.

The new system, called Norpit, employed a Wild-Lietz robotic total survey station (digital theodolite plus electronic distance measurement — EDM). The unit was installed in a stable, weatherproof station on the north wall of the pit. The robotic theodolite was controlled by a microcomputer at the remote station and shot a number of survey prisms located around the pit on walls identified as potential problem areas. The theodolite located each target in the cycle by centering over the prism using a search algorithm based on rebound signal characteristics. The prism location data were relayed by radio telemetry from the remo... station computer to a base station computer located in the pit foreman's office. Here the prism location was automatically calculated, stored in a database and compared to previous location information. Alarms were set up to alert pit foremen when any prism was showing significant movement or was not located by the instrument. The remote theodolite system was capable of surveying up to 50 targets per hour.

The Norpit system had many advantages over the old system. It worked 24 hours/day, hindered only by heavy fog. It compiled a large database of target information which was available for analysis by the geotechnical staff and provided some valuable feedback directly to the operations department without the need of direct technical input. A planned second remote station on the southeast wall, reporting to the same base station, would have provided full 360° coverage of the Bell pit.

Milling

The Bell concentrator was a traditional 1970s designed process using conventional crushing, grinding and flotation to effect the extraction of the chalcopyrite followed by dewatering of the concentrate. In the final years, it treated an average of 16 000 tonnes of ore per day operating on a 24 hour/day 365 day/year basis. Production was 185 tonnes to 225 tonnes of concentrate per day grading an average 27% Cu and 9.6 g/t Au. As with mining, the milling process at Bell evolved over the 20 year life of the operation. This summary focusses on the process as it existed at the close of operations in 1992.

Primary crushing of pit ore was through a 1.65 m × 1.07 m gyratory crusher producing a −15 cm product from the pit-run feed (typically 80% −45 cm); screening out the −13 mm material to the fine-ore bins and depositing the remainder on the coarse ore stockpile. The coarse ore was reclaimed to the secondary crushing plant and run in open circuit through a 2.13 m standard head secondary cone crusher and then into closed circuit through one of two 2.13 m short head tertiary crushers. The final −13 mm product from the crushing process was stored in two fine-ore bins.

With an original design throughput of 9100 tpd, large increases in production at the crushing stage of the Bell concentrator circuit were realized without changing any of the major crushing units. These increases were achieved by three main strategies (pers. comm., R. Tessier, 1994). It was recognized that the initial step in the comminution process was pit blasting and that crusher throughput could be enhanced relatively cheaply if blast fragmentation of ore was pushed as high as practical. Secondly, the primary crusher was identified as the tightest bottleneck in the crushing circuit. To overcome this, the crusher gap settings were opened as much as possible and the crusher was choke fed to allow maximum throughput. As well, detailed maintenance planning and close attention paid to crusher operating parameters allowed for maximum equipment availability. Lastly, modifications to the screen decks in the secondary crushing plant significantly reduced the amount of fines which were incorrectly reporting to the secondary and tertiary crushers and affecting their performance.

The −13 mm fine ore was fed into one of two identical grinding circuits; each with an open circuit 1335 Kw, 3.96 m × 5.49 m rod mill followed by a 2675 Kw, 4.11 m × 8.53 m ball mill in closed circuit. Final output from the grinding process was designed to be 80% passing 74 μ.

Grinding practice changed toward the end of the mine life as the concentrator operating philosophy evolved. This philosophy focussed on maximizing the total metal units which could be produced by the mill for a given period of time and cost. The modified operating practice required a thorough understanding of the grind-grade-recovery relationships for the Bell ore. A significant portion of the ore processed was of low grade (0.25% to 0.45% Cu) and when this ore was being milled, tonnage throughput was the prime operating focus. Reduced recoveries due to coarser grinds were accepted to allow greater throughputs. When processing higher grade material (>0.5% Cu), attention shifted to maximizing recoveries and reducing tailings ...es by slowing down the throughput and increasing the fineness of the grind. Operating changes which allowed this philosophy to be put into practice included optimization of the grinding media and improvements to the hydrocyclones to optimize the recirculating loads and to provide better control of the final grinding product.

After grinding, the ore was fed to one of two rougher flotation lines followed on the underflow side by the scavenger flotation circuit and on the overflow side by two precleaner lines. The precleaner underflow was reground and recombined with the overflow and fed into the three-stage cleaner circuit which had one additional regrinding stage before the final cleaner scavenger bank.

Originally, the Bell concentrator used a number of complicated recirculation and regrinding stages in the flotation circuit that created many chances to recover each particle of chalcopyrite. As with the grinding and crushing circuits, the flotation operating philosophy changed toward the end of the operation. The flotation circuit was altered to provide a more open system with greater first-pass residence time, but lower recirculating loads. As well, every effort was made to route fully liberated chalcopyrite particles to the dewatering stage as early as possible and avoid regrinding of clean ore. These changes allowed the operators greater control over the flotation circuits with greater sensitivity and quicker response time to operator-controlled variables. The commissioning of an on-stream analyzer in 1990 gave the flotation operators the circuit feedback information necessary to match the control they were expected to exert.

The circuit changes improved the overall metallurgy of the Bell concentrator significantly (Fig 11b). Copper recovery during the later years of production ranged from 83% to 85% compared to 80% to 82% average recoveries in the earlier years. Gold recoveries also increased through the life of the operation from 55% to 65% in the early years to 65% to 70% in the last few years. Some of the gold recovery improvement, however, is attributed to changes in gold distribution and its sulphide association with depth.

The final stage of processing at Bell was dewatering. Dewatering was accomplished in three stages. First, the concentrate pulp density was raised by passing it through a 18.3 m diameter thickener. Secondly, the concentrate pulp was caked using a 2.44 m diameter disc filter. Finally, the moisture content of the concentrate was reduced to a designed 7% in a gas-fired rotary-kiln drier.

Mill tailings were pumped to one of two large tailings impoundments south of the pit on the Newman Peninsula for final deposition. The engineered tailings structures consisted of run-of-mine rockfill dams with fine rock filter lining, cycloned sands and, finally, tailings slimes and decant water.

Granisle Mine — Mining, Milling and Engineering Practices

Operating History

Granisle Copper Limited commenced development of the Granisle deposit in early 1965 with initial work focussing on access, plant construction and the clearing for a townsite on the west shore of Babine Lake. Of prime importance during the first year was the installation of a 3.2 km "bubbler" system to keep an ice-free channel open across Babine Lake to Sterrett Island through ice up to 1 m thick during winter conditions (Carter and Clarke, 1966). This bubbler system utilized an air-line 3900 m long suspended at a maximum depth of 50 m below surface. Air fed through small holes spaced 10 m apart in the pipe, coupled with regular barge crossings, kept a 30 m wide channel open under normal winter conditions. A similar bubbler system 13 km further north on Babine Lake was later used to provide access to the Bell mine.

By the end of 1966, 320 000 tonnes of overburden and 450 000 tonnes of waste rock had been removed in preparation of the Granisle open pit. On November 16, 1966, the Granisle concentra-

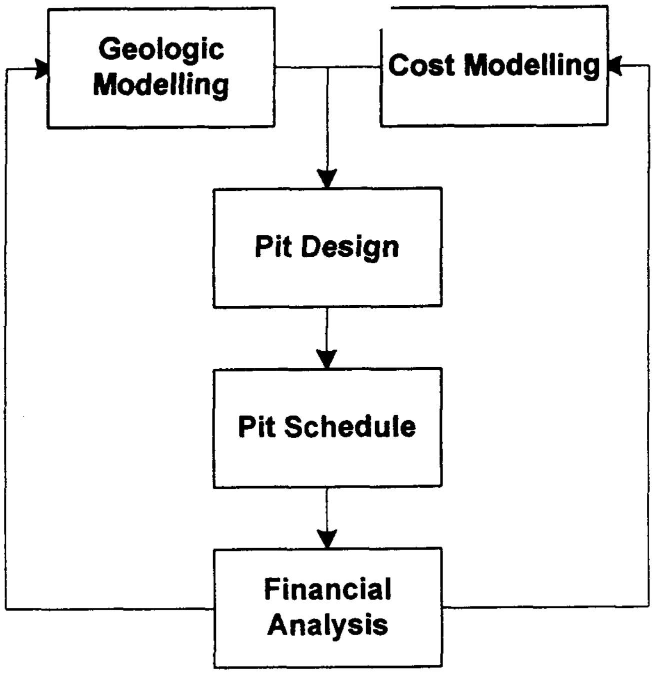


FIGURE 14. Work flowchart showing the major work stages common to a financial and technical study of pit expansion or new pit development, highlighting the iterative nature of the process.

tor commenced processing at a rate of 4500 tpd (Clarke, 1967; Fahrni et al., 1976).

Power for the Granisle plant and townsite was supplied initially by 11 diesel-electric generating units with a total capacity of 5500 kW. This power was replaced with hydroelectric grid power in 1971 when a 138 kW transmission line was brought in to service the Bell mine and surrounding region.

In 1969, Granisle Copper Limited initiated a major drilling program to search for additional reserves. This program was successful and led to a plant expansion which saw throughput increase to 12 000 tpd by 1973.

Total production from startup in 1966 to closure in 1982 was 52.7 million tonnes averaging 0.47% Cu with an average waste to ore ratio of 1.37:1. The original feasibility open pit reserve was 20.6 million tonnes grading 0.52% Cu using a cutoff grade of 0.3% Cu and a waste to ore strip ratio of 0.15:1.00.

Amalgamation of the Granisle and Bell operations took place in late 1979 following the Noranda Group acquisition of the Granisle mine. Operations continued until 1982 when low metal prices forced closure of both mines; the Granisle mine was decommissioned as its remaining resources were considered subeconomic at mid-1980 metal prices and operating costs.

Although low grade, Granisle was a successful operation during its early years due to a low strip ratio (deposit outcrop formed a small knoll), a high concentrate grade (bornite:chalcopyrite ratio significantly higher than most British Columbia porphyry copper deposits), strong copper prices during the critical early years of operation and overall low capital cost due to the effective use of used equipment. Granisle Copper Limited recovered its initial investment in the first two years of operation.

Operating Practice

As at Bell, operating practices at Granisle evolved over the life of its operation. These involved conventional mining and milling practices that differed little from those described for Bell and, hence, will not be described in detail.

Diamond drilling at Granisle totalled 41 700 m completed in four stages with 1200 m in 1929, 7900 m during the period 1955 to 1962, 29 000 m during the period 1969 to 1973 and a final 9600 m in 1990. Of the 280 holes completed, 213 were vertical. Average depth was 110 m, the deepest hole was 522 m (450 m asl) and drill hole spacing averaged 60 m.

The Granisle pit was initially mined in 9.1 m (30 ft) benches but was increased to 10.7 m (35 ft) benches below the 2470 bench. Production ranged from just over 5000 tpd of mostly ore in the early years to 40 000 tpd with 12 000 tpd of this being ore in 1977. Pit operations ran 24 hours/day and 365 days per year. Granisle production performance is illustrated in Figure 11. The difference in concentrate grade between Bell and Granisle (due to the higher bornite component at Granisle) is clearly evident in Figure 11b.

In 1977, Granisle Copper Limited initiated an investigation into the possible recovery of a molybdenum byproduct. This led to the construction of a molybdenum circuit in the Granisle mill. This circuit became operational in 1980 but was shut down shortly thereafter when a sharp decrease in price made molybdenum recovery uneconomic.

Reclamation and Environmental Considerations

During the operating life of the Granisle and Bell mines, care was taken to protect the environment and to initiate reclamation of disturbed areas as soon as possible. Testing and research were carried out to establish the best type of natural vegetation, fertilizers and ground preparation to optimize reclamation results. Since closure of Granisle in 1982 and Bell in 1992, decommissioning plans at both sites have been implemented.

Economic Considerations: Bell 2000 Project

The feasibility of extending the life of the Bell operations beyond 1992 was studied in earnest beginning in late 1988. The study was named the Bell 2000 project. The project work included diamond drilling, geologic mapping, revision of existing geologic models, ore reserve estimation, metallurgical investigations, metallurgical modelling, evaluation of alternative mining methods, investigation of alternative waste disposal options, operating cost modelling, revenue modelling, geotechnical investigations and modelling, idealized pit designs, detailed pit designs, mining schedules, detailed operating and capital cost estimations and financial analyses (Stothart, 1991).

The project included a review of all known potential mining opportunities within 30 km of the Bell facilities, including the Granisle and Morrison deposits. The Bell deposit became the prime focus of the study as it was recognized that a tonne of ore from Bell would displace material of similar grade from either Granisle or Morrison on the basis of transportation costs alone.

The study was a reiterative process (Fig. 14) which was carried on at varying levels of intensity before final reports were submitted early in 1992. Unfortunately for financial reasons, the decision was made not to proceed with the Bell 2000 expansion. The type of evaluation carried out for Bell was also carried out to a lesser extent for the Granisle mine. To avoid repetition, only the Bell evaluation is detailed in this paper.

Much of the work during the evaluation study relied heavily on computers (Stothart, 1991). These were used to manipulate the large amount of data and to facilitate multiple iterations of complicated procedures. Imperial measurements and drawings were used throughout the Bell 2000 evaluation to minimize conversion errors and to preserve continuity with operating practice and existing data.

Reserve Estimation

Estimation Techniques

The original production feasibility study for the Bell deposit (Hall and Kraft, 1969) was based on an ore reserve model constructed using spherical interpolation of diamond drill hole assays representing

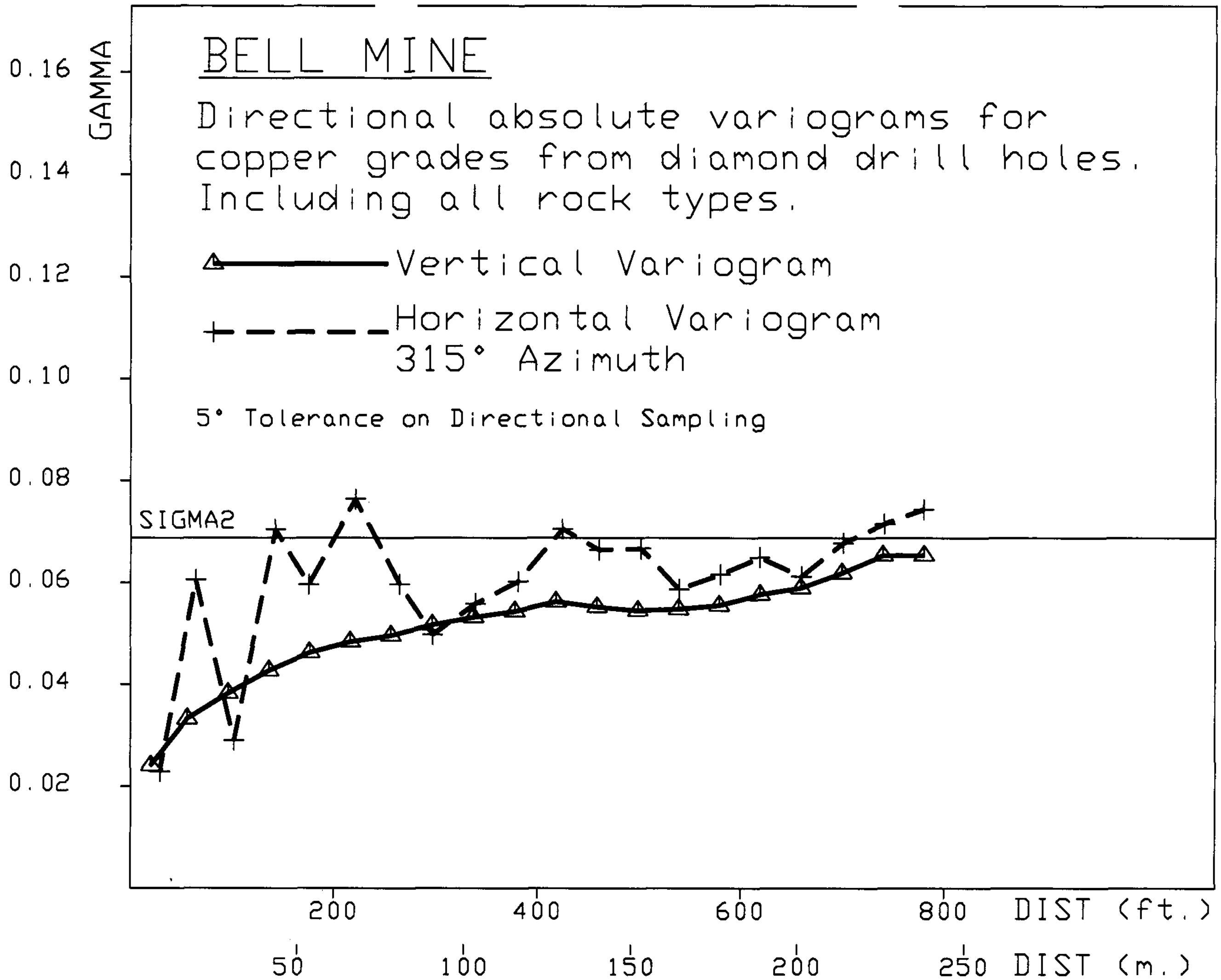


FIGURE 15. Experimental copper variogram for the Bell deposit. Both variograms show a nugget effect of about 0.018% Cu. The vertical variogram is equivalent to the major axis of the spherical search volume used for grade interpolation and exhibits a range of ~250 m. The horizontal variogram at a 315° azimuth with a range of ~75 m is roughly equivalent to the minor axis of the search volume. The actual minor axis for the search volume varies depending which interpolation zone is being modelled and is radial to the orebody annulus.

30 000 m of drilling in 215 vertical AX and AQ diamond drill holes spaced 30 m to 60 m apart. As more detailed geological and grade information became available during the course of mining, the model was revised beginning in 1974 using the proprietary General Pit System (GPS) suite of computer programs developed by Noranda.

The principal considerations to design an ore reserve model for Bell are the pipe-like symmetry of the deposit, the "assay wall" type of contacts and the occurrence of post-ore features. The assay wall boundaries present a modelling problem in that the inner and outer assay walls are not symmetrical. Whereas grades decrease gradually outward from the porphyry system, the inner assay wall boundary between the stockwork ore and the biotite-magnetite altered core is much more abrupt and dips inward at around 75° in the upper parts of the deposit. Averaging of grades across the inner boundary would have resulted in upgrading of waste blocks and downgrading of ore blocks. These considerations were largely resolved by the use of inverse distance squared weighted interpolation of diamond drill and blast hole assays within a search volume defined by an oriented parallelepiped. The dimensions and orientation of the search parallelepiped were designed to reflect sample density, the continuity of grades in the vertical direction and continuity of grades in directions tangential to the porphyry "doughnut" or annulus. Post-ore features, in particular a post-ore dike or plug in the northeast corner of the deposit, were partially isolated using planar constraints across which interpolation could not occur. This was only partially successful, as the plug had been essentially straddled by diamond drilling and proved to be much more irregular than modelled. The GPS ore reserve model, revised from time to time with new diamond drilling data, was the basis for the later expansions of the mine beyond the original feasibility pit.

During the three-and-a-half years of the Bell 2000 project, the Bell orebody was modelled a number of times. Techniques used included inverse distance weighted interpolation, relative kriging and sectional polygon estimation methods. Early in the study, and in confirmation of earlier ore modelling efforts, it was determined that Bell was a relatively 'well behaved' deposit in regard to modelling for copper distribution. Estimates obtained using given sets of geologic assumptions and given sets of assay data were highly repeatable, both at the global and local level, regardless of the estimation technique used. Repeated modelling runs were necessary to manage the significant amount of additional data that was generated during the course of the project and to accommodate evolution in geologic thinking.

The reason ore estimation results obtained at Bell are relatively insensitive to the technique used is best explained by the grade distribution characteristics determined by the geostatistical analysis done early in the project. The two important points to note on the sample variograms for copper grade (Fig. 15) are the small nugget effect

(0.018% Cu) and the long range (: m). Expressed differently, this means a good estimate of a copper grade can be obtained by using the nearest known sample point(s) and the reliability of the estimate is high even if the known sample point is relatively distant from the point of estimation. Additionally, the geostatistical evaluation confirmed geological observations that local grade continuity within the ore was highest in a vertical or near vertical direction (range $\approx$250 m) and in directions tangential to the porphyry annulus (range 150 m to 200 m) and was low in directions which were radial to the annulus (range 75 m to 100 m). It was also concluded that the orebody exhibited a strong proportional effect. This effect could be filtered out, but it ruled out simple kriging as a useful estimation technique at Bell because of that method's fundamental assumption of independence of variability.

As the Bell resource estimation results were shown to be relatively insensitive to the estimation technique employed, it was possible to select a method that would provide the desired degree of confidence at minimum cost in terms of time and resources. The majority of reserve estimations completed during the Bell 2000 review were performed using an iterative adaption of the inverse distance method. This method was chosen over other options as it was well suited to producing a regular block model (as was required for input to pit design optimizing routines) and was capable of rapidly handling the very large data sets involved. In addition, the method itself is straightforward and the influence of the design parameters on the execution and results were well understood by all team members and could easily be communicated to others.

The orebody modelling work that was done at Bell involved five basic steps: data gathering, geologic interpretation, identification of the estimation design parameters, running the model on the computer, and finally quantifying and analyzing the model. As mentioned earlier, this process was performed repeatedly as more data became available, as the geologic interpretation was refined and as the interpolation technique evolved.

Data Gathering

A very large and diverse volume of raw data was used during the Bell 2000 evaluation and required constant review to ensure integrity. Both diamond drill hole and blast hole assay intervals were used as sample points. Lithologic and alteration data were derived from core logging with additional information provided from geologic pitwall mapping. Topographic data were obtained from aerial topographic maps and from mine surveys. Geotechnical data were not used significantly in the ore estimation process, but did see later service during the pit design process. Geotechnical data were gathered by structural mapping of pit walls and during core logging by measuring the rock quality designation (RQD) and noting characteristics of structural discontinuities.

Diamond drilling history at Bell includes the exploration phase mentioned earlier, followed by 9600 m of mixed vertical and angle, largely BQ, core holes completed during the mining phase from 1972 to 1988 and 37 300 m of NQ drilling in 128 angle drill holes completed as part of the Bell 2000 project. Total diamond drilling on the Bell property at closure in 1992 was 78 000 m in 445 holes. Of this total, 73 300 m in 382 holes delineated the Bell deposit with 60% of the meterage drilled in angle holes. Drill hole spacing varied from 30 m to 120 m. A target pierce point spacing of 60 m on the "ultimate" pit surface was largely achieved.

The diamond drilling information was entered into "Normin", a proprietary computer database program developed by Noranda. The actual database included administrative data, collar locations, down-hole acid dip tests, Sperry Sun measurements, interval assays for copper (typically 3 m to 4 m), composite assays for copper and gold (typically generated over bench intervals of 12.2 m), geotechnical information, geologic descriptive logs and a six-character geological code. Coding of pre-1989 diamond drill logs was done and included in the new database. However, this effort was made difficult because of the changing geological concepts and terminology used by some 23 geologists over a 27-year period.

Sectional and pl aaps were developed from the database. A curve-fitting algorithm was used to generate drill hole traces based on collar location and downhole survey information. Part way through the Bell 2000 project, down hole Sperry-Sun azimuth measurements were discarded when it was noted that many calculated drill hole traces showed abnormal deviations toward the core biotite altered biotite feldspar porphyry (BBFP) unit. These abnormal deviations were attributed to the relatively high magnetite content of the BBFP unit.

Using the Normin program, an assay sample data file (SDF) was generated for copper grades for all samples greater than 1.5 m in length. Each assay was represented as a point sample located at the midpoint in space of the sample interval.

Blast hole data were digitized from existing blast hole plans and included collar coordinates and copper assay. Copper assaying at Bell was typically carried out for almost every blast hole giving an average sample separation of 7.6 m in plan. The samples were taken as cuts across the entire cuttings pile and were intended to be representative of the entire blast hole length (typically 13.4 m). The largest mining benches had upward of 8000 blast holes and the overall data set was some 97 000 assays. A separate data set for gold assays of blast holes was also input by digitizing. In many locations, particularly in areas that were mined early in the life of the mine, no gold assaying had been carried out. Even when done, gold sampling of blast holes was carried out on a much broader interval than that for copper, giving an average sample spacing of about 30 m in plan.

An SDF for blast hole assays was created with the point location for each sample being set at the mid-bench elevation. The blast hole SDF was screened so that only those assays contained in a 60 m thick shell adjacent to the walls of the pit remained in the file. The approach was justified on the basis that data from the centre of the mined out pit were too distant to be used for estimation of block grades in unmined areas and that, in any event, samples in the shell would mask the effect of data in the centre due to the inverse distance estimation method being used. The diamond drilling SDF and the blast hole SDF were merged into a single consolidated SDF for use in interpolation runs.

Geological information gathered through wall mapping was not entered into the computer database but was plotted on geologic bench plans and used for verification purposes. Structural mapping information, however, was collected in a database program developed by the Noranda Technology Centre. Information could be generated from this database as graphical output either in the form of stereoplots or as structural maps.

Topographic data were digitized into the AutoCAD drafting system where they could be used as regular mine plans and topographic maps or as a sample database of point elevations. Computer representations of the original topography and of the overburden/bedrock interface were also maintained.

Maps used most extensively during the Bell modelling phase included plans generated at bench intervals (12.2 m) corresponding to existing mine bench elevations and two sets of vertical sections generated at 30 m intervals (100 ft): one set looking north and the other west. These composite plans and sections were produced at 1:1200 and 1:2400 scales and contained current pit and surface topography, potential pit expansion limits, coded geological information and assay information plotted as colour-coded grade histograms, profiles and/or text along the drill hole trace. Software-generated blast hole copper assay contours or isopleths (colour-coded by grade interval) were also plotted on the sections and bench plans and were invaluable in interpreting geological contacts. Similar plans and sections were developed for the Granisle and Morrison deposits to maintain standards and to facilitate comparisons.

Geologic Interpretation

Geologic interpretation at Bell and Granisle was an iterative and evolving process that focussed on the economic aspects of grade

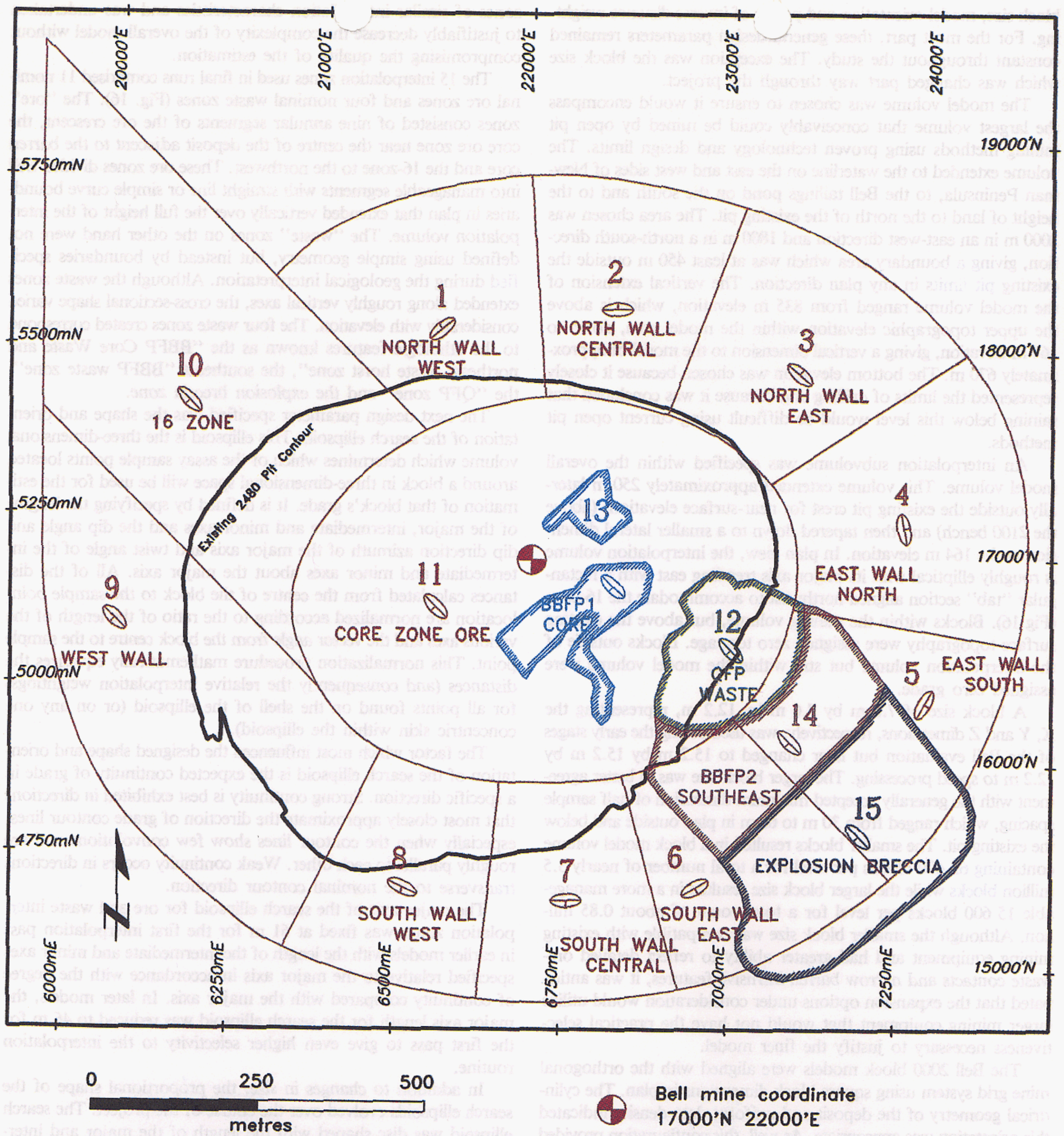


FIGURE 16. Interpretation zones, Bell copper model. Interpolation algorithm used soft boundaries between the nominal "ore" zones (1 - 11) giving gradual "assay" walls to the outside of the deposit and hard boundaries around the nominal "waste" zones (12 - 15) giving sharp ore/waste contacts to the inside of the deposit. "Ore" zone contacts are regular geometric shapes. "Waste" zone contacts are from geologic interpretation with the "waste" zone being carved out of the coincidental "ore" zone and are shown for the 450 m (1500 ft) asl level. Search volume ellipsoids are shown for each zone illustrating the intermediate and minor axes directions. The major axis for all ellipsoids was vertical or near vertical.

estimation. The interpretation had to predict, with a high degree of certainty, copper and gold grade distributions within the overall deposit as well as the nature and form of boundary regions between areas of high- and low-grade mineralization. In addition, areas lacking sufficient information and areas of exceptional exploration potential had to be identified. These areas were drilled immediately if it could be demonstrated that the outcome of the drilling had the potential to impact on the overall economics of the deposit.

The procedures adopted for geologic interpretation generally involved individual work with later review by others or small group efforts. Interpretive work was typically carried out using physical plans and sections rather than computer images as they were easy to edit, showed evolution of thinking and allowed for more aggressive interaction between participants in the interpretive process.

Identification of Interpolation Design Parameters

The next step in this orebody modelling involved translating the geological interpretations into a set of "rules" or design parameters. These parameters were applied to the interpolation computer algorithm to generate a block model that conformed to the grade distribution concepts envisioned by the geologists.

The initial design parameters specified were: model volume, block size, model orientation and p · of inverse distance weighting. For the most part, these genera. design parameters remained constant throughout the study. The exception was the block size which was changed part way through the project.

The model volume was chosen to ensure it would encompass the largest volume that conceivably could be mined by open pit mining methods using proven technology and design limits. The volume extended to the waterline on the east and west sides of Newman Peninsula, to the Bell tailings pond on the south and to the height of land to the north of the existing pit. The area chosen was 2000 m in an east-west direction and 1800 m in a north-south direction, giving a boundary area which was at least 450 m outside the existing pit limits in any plan direction. The vertical extension of the model volume ranged from 835 m elevation, which is above the upper topographic elevation within the model area, down to 164 m elevation, giving a vertical dimension to the model of approximately 670 m. The bottom elevation was chosen because it closely represented the limits of drilling and because it was concluded that mining below this level would be difficult using current open pit methods.

An interpolation subvolume was specified within the overall model volume. This volume extended approximately 250 m laterally outside the existing pit crest for near-surface elevations (above the 2100 bench) and then tapered down to a smaller lateral dimension at the 164 m elevation. In plan view, the interpolation volume is roughly elliptical with its major axis trending east with a rectangular "tab" section aligned northwest to accommodate the 16-zone (Fig.16). Blocks within the mining volume, but above the existing surface topography were assigned zero tonnage. Blocks outside of the interpolation volume but still within the model volume were assigned zero grade.

A block size of 7.6 m by 7.6 m by 12.2 m, representing the X, Y and Z dimensions, respectively, was used during the early stages of the Bell evaluation but later changed to 15.2 m by 15.2 m by 12.2 m to speed processing. The larger block size was in better agreement with the generally accepted minimum dimension of half sample spacing, which ranged from 30 m to 60 m in plan outside and below the existing pit. The smaller blocks resulted in a block model volume containing 62 400 blocks per level for a total number of nearly 3.5 million blocks while the larger block size resulted in a more manageable 15 600 blocks per level for a total count of about 0.85 million. Although the smaller block size was compatible with existing mining equipment and had greater ability to reflect detailed ore-waste contacts and narrow barren intrusive features, it was anticipated that the expansion options under consideration would utilize larger mining equipment that would not have the practical selectiveness necessary to justify the finer model.

The Bell 2000 block models were aligned with the orthogonal mine grid system using square block dimensions in plan. The cylindrical geometry of the deposit and uniform data density indicated this orientation was appropriate. As well, this configuration provided the added benefit of allowing the manipulation of data already defined in mine coordinates without coordinate transformations.

An inverse power of 2 was used to estimate copper grades during the inverse distance sample weighting procedure. Use of inverse distance squared is a common practice for copper porphyries and reflects a good balance between the smoothness of data continuity at Bell which would dictate a lower inverse power and the portion of the estimation error reflected by the low nugget effect which would dictate a higher inverse power.

The interpolation volume was subdivided into a number of interpolation zones in order to group blocks together that could reasonably be estimated using the same interpolation rules. The number, size, shape and orientation of the interpolation zones, as well as the specific interpolation design parameters associated with the individual zones, evolved over the course of the project with changes in geologic interpretation. Earlier estimation runs used up to 23 separate zones while later estimations used 15 zones. The reduction in the number of zones was accomplished by coalescing smaller zones of similar int lation characteristics and was undertaken to justifiably decrease the complexity of the overall model without compromising the quality of the estimation.

The 15 interpolation zones used in final runs comprised 11 nominal ore zones and four nominal waste zones (Fig. 16). The "ore" zones consisted of nine annular segments of the ore crescent, the core ore zone near the centre of the deposit adjacent to the barren core and the 16-zone to the northwest. These ore zones divided Bell into manageable segments with straight line or simple curve boundaries in plan that extended vertically over the full height of the interpolation volume. The "waste" zones on the other hand were not defined using simple geometry, but instead by boundaries specified during the geological interpretation. Although the waste zones extended along roughly vertical axes, the cross-sectional shape varied considerably with elevation. The four waste zones created correspond to the lithologic features known as the "BBFP Core Waste and northeast waste horst zone", the southeast "BBFP waste zone", the "QFP zone" and the explosion breccia zone.

The next design parameter specified was the shape and orientation of the search ellipsoid. This ellipsoid is the three-dimensional volume which determines which of the assay sample points located around a block in three-dimensional space will be used for the estimation of that block's grade. It is defined by specifying the lengths of the major, intermediate and minor axes and the dip angle and dip direction azimuth of the major axis and twist angle of the intermediate and minor axes about the major axis. All of the distances calculated from the centre of the block to the sample point location are normalized according to the ratio of the length of the various axes and the vector angle from the block centre to the sample point. This normalization procedure mathematically equalizes the distances (and consequently the relative interpolation weightings) for all points found on the shell of the ellipsoid (or on any one concentric skin within the ellipsoid).

The factor which most influenced the designed shape and orientation of the search ellipsoid is the expected continuity of grade in a specific direction. Strong continuity is best exhibited in directions that most closely approximate the direction of grade contour lines, especially when the contour lines show few convolutions and are roughly parallel to each other. Weak continuity occurs in directions transverse to the nominal contour direction.

The major axis of the search ellipsoid for ore and waste interpolation zones was fixed at 61 m for the first interpolation pass in earlier models with the length of the intermediate and minor axes specified relative to the major axis in accordance with the degree of continuity compared with the major axis. In later models, the major axis length for the search ellipsoid was reduced to 46 m for the first pass to give even higher selectivity to the interpolation routine.

In addition to changes in size, the proportional shape of the search ellipsoids evolved over the course of the project. The search ellipsoid was disc shaped with the length of the major and intermediate axes being of equal length and the minor axis length at 50% of the major axis length for all of the ore interpolation zones in the later runs. The orientation of the ellipsoid in the nine annular ore zones had the major axis at a vertical or near-vertical attitude, the intermediate axis was aligned roughly tangential to the crescent shape of the ore in plan and the minor axis was aligned roughly radial to the crescent shape. These orientations show a high correlation with the geostatistical observations which were noted earlier.

The search ellipsoids for the 16-zone and the core ore zone also oriented the major axis in the vertical direction. However, the intermediate axis for these ellipsoids was aligned radial to the main ore crescent in a direction parallel to the general strike direction of the apophysis to the northwest.

The search ellipsoid for all of the waste zones was kept spherical because the data density was comparatively sparser than in the ore areas, making continuity predictions more difficult, and because the very low grade in this area made the interpolated results of this

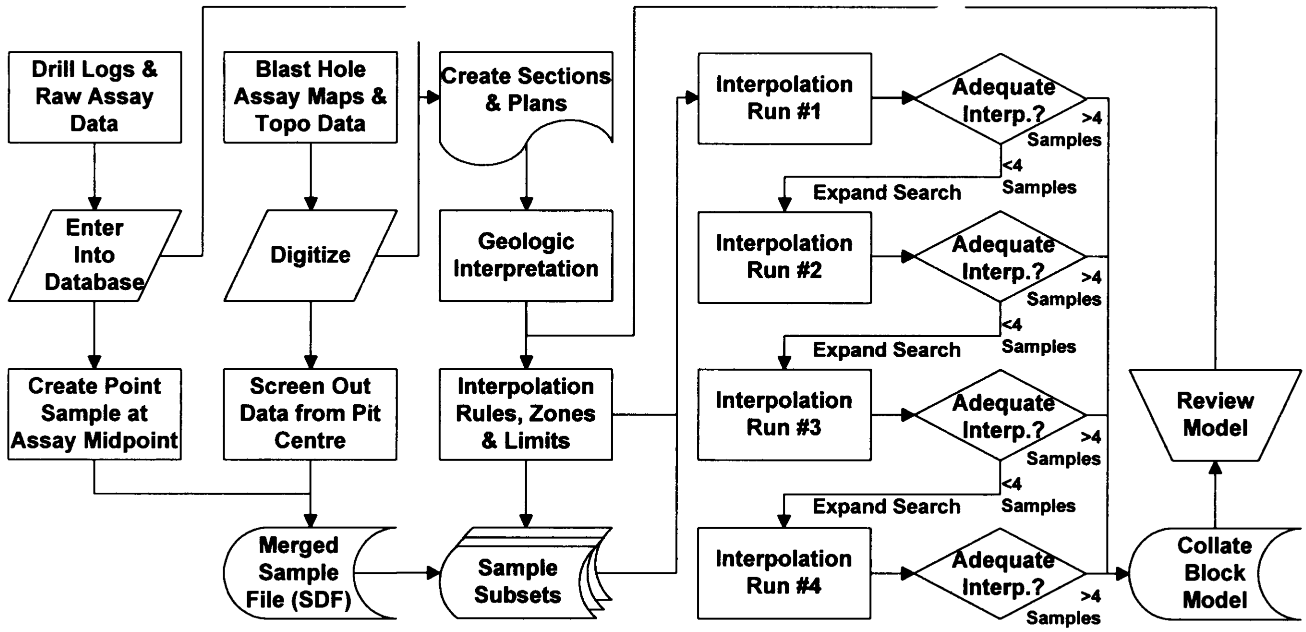


FIGURE 17. Bell 2000 project ore modelling flowchart. Steps involved with acquiring, collating and analyzing the input data are on the left of the diagram. Steps showing the procedure used during actual running of the model on the computer are on the right of the model. Again, note the iterative nature of the process (from Stothart, 1991).

material virtually inconsequential from an economic standpoint — waste is waste whether it is 50% of the cutoff grade or 25% of the cutoff grade.

The last step in establishing the interpolation design parameters for each interpolation zone involved specifying which assay data would be "visible" to the interpolation routine within each zone. This was done by describing what were known in mine terminology as "hard" (or "nominally intrusive") and "soft" (or "assay wall") contacts.

Hard contacts were stipulated as discontinuities across which the copper assay changed very rapidly from higher grade to very low (nearly barren) grade material in a distance less than the block size. In contrast, soft contacts exhibited a relatively slow gradational change from ore to waste grade material. The hard contacts at Bell coincided with the interpolation zone boundaries of the four waste zones and all other areas were assumed to be assay walls. During interpolation, the process for determining the interpolation sample set for a block in a given zone would search across soft boundaries to find sample points, but would not search across hard boundaries. In this manner, it was possible to "hard wire" some of the ore-waste contacts and to let the interpolation procedure itself determine the remaining ore-waste transitions mathematically.

Running the Model

The next step in the creation of a block model involves generating block interpolation values. The technique at Bell involved running multiple interpolation passes across the model (Fig. 17). This technique estimated the grade of a block and, at the same time, generated a factor that provided a measure of estimation confidence.

The software used for interpolation during the Bell 2000 project was the microcomputer-based ISD3® program developed by GEOSTAT Système International Inc. (GSII) of Montreal. Because of computer memory limitations, the software imposed restrictions on the number of blocks which could be estimated in a single run and on the size of the data set which could be used. These restrictions required that the interpolation zones be subdivided into units small enough to allow the program to execute properly. Even after the block size was doubled in the X and Y dimensions, the creation of a single block model typically took about 24 hours of actual computation time spread over two or three working days.

The initial task in the procedure was to partition the combined diamond drill hole and blast hole sample data file (SDF) into smaller subsets which the software could handle more easily. The first division of the SDF was along hard boundaries. This separated the data into "ore" and "waste" data sets which could be used respectively with the ore and waste interpolation zones blinding the search ellipsoid to data on the other side of a hard boundary. It should be reiterated that the ore SDFs still contained a significant number of assay points below the economic cutoff grade. The final partitioning step further divided the ore sets into a number of overlapping subsets corresponding to the volume in space which could potentially be selected by the search ellipsoids within a group of adjacent interpolation zones.

After the data sets had been screened, the actual interpolation runs were begun. The final Bell 2000 models were generated using four iterative passes or interpolation runs.

For Run 1, all blocks within the total interpolation volume were estimated using inverse distance weighted interpolation with search ellipsoids and data sets dictated by the interpolation zone in which a particular block fell. Output was an ASCII file that used one record per block containing the X, Y and Z indices of the block, the number of samples found within the search ellipsoid and used in the estimation to a maximum of the 30 closest assay points, the estimated copper grade and an alpha-numeric code identifying the interpolation zone of the block.

The resultant zone block model was then sorted. Blocks that had been interpolated on the basis of four or more assay points known as "adequately interpolated" blocks were sorted out of the full model and a "run number" field appended to their block record identifying them as "Run 1" blocks. The remaining blocks — including those using less than four samples known as "underinterpolated" blocks and blocks which had failed to locate any sample points within the search ellipsoid known as "uninterpolated" blocks — were cycled back into the interpolation routine for the second run after discarding their Run 1 grade estimates (Fig. 17).

During the second run the size of the search ellipsoid was doubled with the relative proportions between axes held constant giving a major axis of 91 m length. After the second interpolation run, the blocks were again sorted into an adequately interpolated group which were assigned a "Run 2" designation and an inadequately interpolated group. The cycle was repeated on a steadily decreasing number of blocks for th ‘hird and fourth runs with the search ellipsoid being expanded ι ιple and quadruple its Run 1 size, respectively.

Following Run 4, all of the files of adequately interpolated blocks were merged. The resulting block file contained not only the estimated grades for all blocks but also the run number corresponding to the run used to generate the block estimate. It should be noted that even for Run 4 the maximum distance any sample point could be pulled into an interpolation (in the direction of the major axis) was ~180 m which is well within the range exhibited in the sample variogram (Fig. 15).

The final and most important step in the block modelling process involved a review of the block model by geologists and mine engineers to ensure it accurately reflected current geologic interpretation. This review was facilitated by the use of graphic software such as GSII's BLOCKCAD® program which allows users to view many orthogonal views of a model in relatively quick succession. Such reviews frequently led to adjustments being made and rerunning of the block model (Fig. 17).

Figures 18 and 19 show typical block model section and plan views through the Bell deposit. The upper plot in each figure shows the copper grade distribution while the lower plot shows the "run number" distribution. The run numbers were used during the Bell 2000 project to quantify grade confidence with the material estimated in runs 1 through 3 being categorized as combined proven and probable ore reserves (Nilsson, 1992). The total geologic resource within interpolation volumes for Bell and Granisle is shown in Figure 20.

Gold Estimation

A block model estimate of gold distribution was also done for the Bell orebody. The process was essentially identical to that used for copper. The same interpolation zones and search ellipsoids were used with the exception that the core ore zone was subdivided into three separate interpolation zones using different search criteria.

The sample density for gold was much sparser than for copper. In addition, gold assay confidence was low due to the high portion of the samples with gold grades at or near the detection limits of the assay technique. Further, gold check assays identified serious variance problems that were not resolved at the time of closure in 1992.

Dummy gold assays were calculated for early diamond drill holes that had not been analyzed for gold by applying a ratio of 0.46 g/t per % Cu to weighted copper assay composites. This ratio was developed on the basis of experience obtained during the Bell 2000 program. This step was taken to increase apparent gold sample density in the pit core area where very little new diamond drilling was done.

Pit Design

Floating Cone Optimization

Following the production of a satisfactory block model, various expansion pits were designed through a two-step process: an "optimized" pit design as defined using various "floating cone" computer programs and a detailed sequence of working pit designs to an ultimate pit-limit using computer-aided drafting tools. The process required many iterations.

The existing pit surface topography and the block model were the primary geometrical inputs to the floating cone programs. For the final Bell 2000 study, only reserves classified as proven or probable (Runs 1 to 3) were considered by the floating cone program. Run 4 blocks were assigned a grade of 0% Cu (Nilsson, 1992). The primary technical inputs to the programs were pit-slope design parameters, specific gravity and projected concentrator performance.

Pit design slopes were developed by geotechnical consultants as an extension of work done on design slopes for the existing Bell pit. The consultants based their work on the detailed structural database that had been ‘ ‘oped, together with certain aspects of the geologic interpretatic.. ınat was ongoing. The final factor used in the development of the pit slope recommendations was the final depth of the pit which was projected to be in the range of 400 m to 500 m below the existing pit crest.

Based on work done by Bell metallurgical staff, the specific gravity of the rock was calculated to be inversely proportional to the copper grade between ceiling and floor limiting values. All unmineralized blocks and blocks with an estimated copper grade less than 0.20% were assigned a specific gravity of 2.67. Blocks with copper grade between 0.20% and 0.75% were assigned a specific gravity based on the equation:

$$\text{s.g.} = 2.67 + [(\%\text{Cu} - 0.20) \times 0.333]$$

All blocks grading above 0.75% Cu were assigned a specific gravity of 2.85.

Future mill operating performance, including copper and gold recovery, was forecast based on historical results and projections for results from planned process improvements.

The last group of inputs to the floating cone routines were economic factors. Mining, milling and site operating costs were estimated based on historical performance at Bell and projections for different scale operations were based on data from other properties. Shipping, smelting and refining terms were estimated based on standard contract conditions in effect at the time. Commodity prices and exchange rates used for revenue estimations were based on forecasts made by the Noranda Sales unit of Noranda Minerals. The floating cone evaluation is an operating cost versus revenue based algorithm. Therefore, capital cost estimates were not among the inputs at this stage of the evaluation.

When all of the inputs had been compiled, the floating cone program was executed. Various commercial, consultant-supplied and in-house floating cone programs were used during the evaluation work. All programs gave basically the same results for a given set of inputs. However, there was a wide variation in the execution speed of the programs, the amount of preparation work to prepare the input files and the degree of on-line feedback which the various programs provided. Execution time for the same block model on the same high-powered desktop computer equipment using different programs could vary from as much as seven to eight days to as little as 30 minutes.

It became apparent during the early stages of the evaluation work that the final pit size was extremely sensitive to a wide variety of minor manipulations of the input parameters. For example, varying the copper price upward by 15 cents but still remaining within the possible range of future prices could triple the ultimate size of the optimum pit as projected by the floating cone algorithm. The primary reason for this extreme sensitivity was the steepness of the grade-tonnage relationship in the range of the cutoff grades being considered for the operation (Fig. 20). Small changes in cutoff grade produced drastic changes in ore tonnes, ore grade and strip ratio.

An additional observation made during the floating cone runs was that the apex of the search cone needed to be well into the higher grade mineralization below the existing pit before sufficient operating profit could be accumulated in order to displace the waste above. This observation indicated that a large amount of stripping would be required before sufficient ore was exposed to begin generating operating profits. This situation also demonstrated one of the weaknesses of a simple floating cone evaluation which is solely operating-cost based and does not account for the time differential between when costs are incurred for waste mining and revenues are realized from ore mining.

The final "optimum" pit design was based on best estimates for the input parameters using achievable projections and contained approximately 100 million tonnes grading 0.41% Cu above a 0.20% cutoff grade with a 1.35:1 strip ratio.

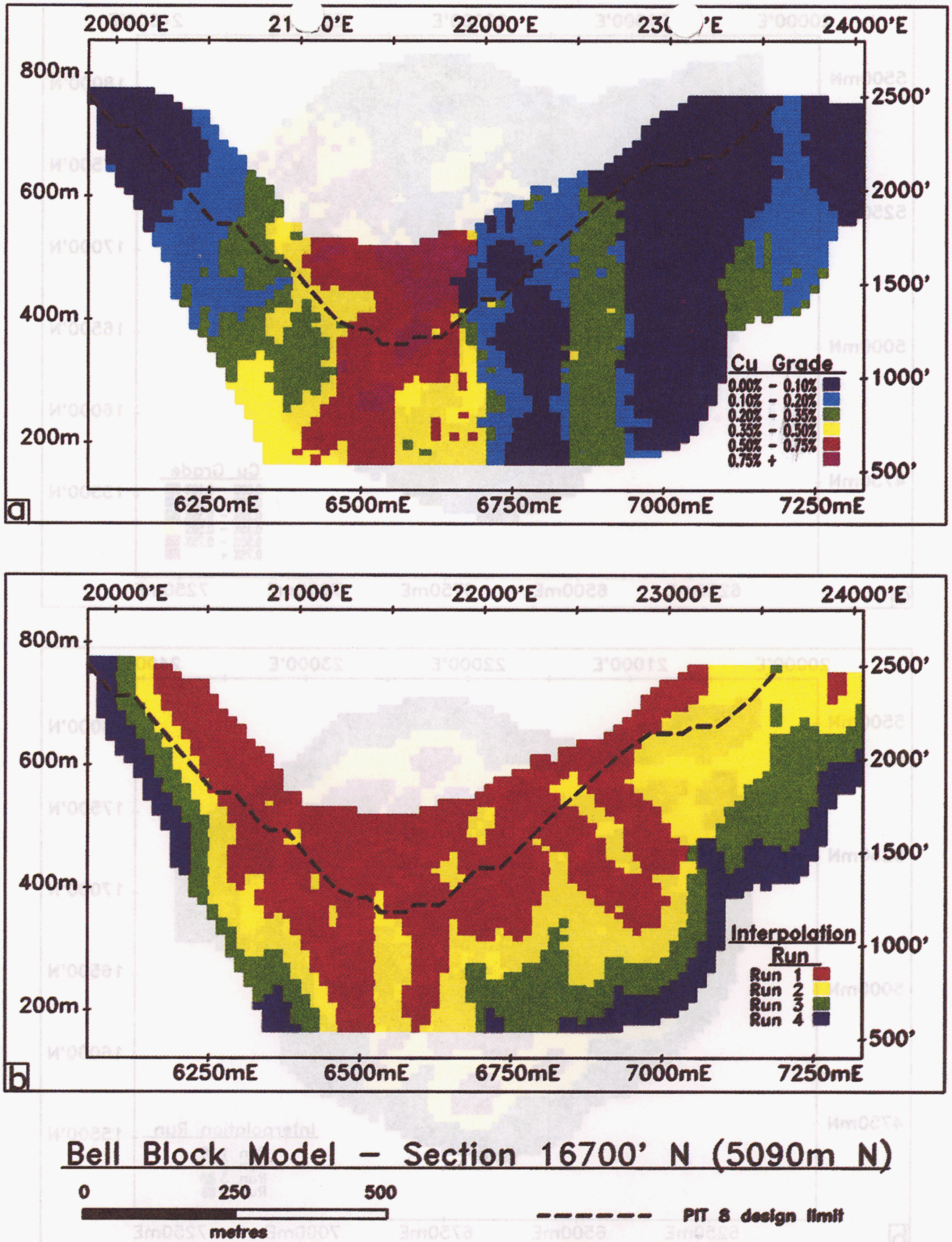


FIGURE 18. Section 16700 North (5090 m N), Bell copper block model. a: Blocks coloured by estimated copper grade. b: Blocks coloured coded by interpolation run number. Note southeast extension to the orebody below the 580 m elevation.

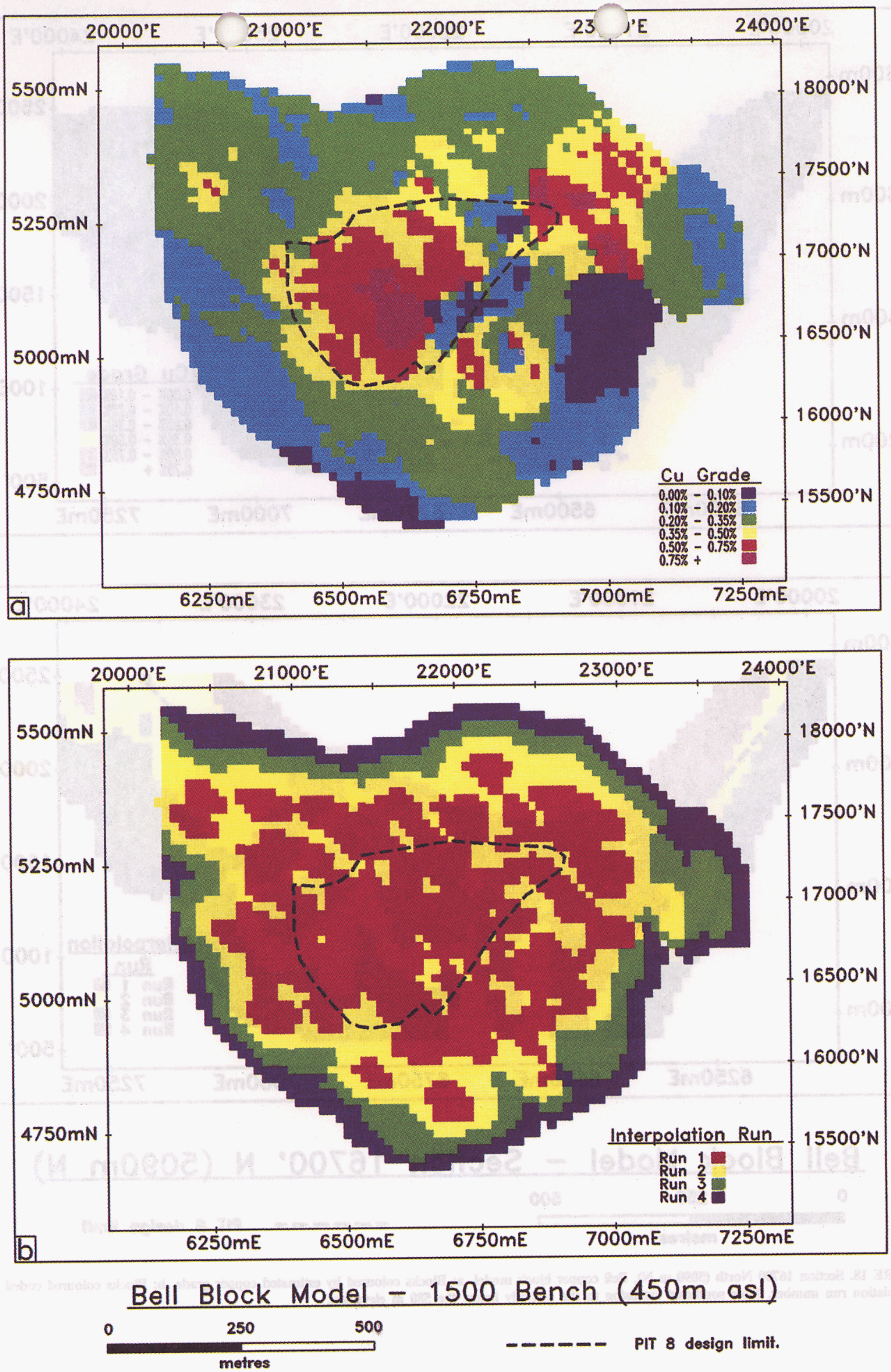
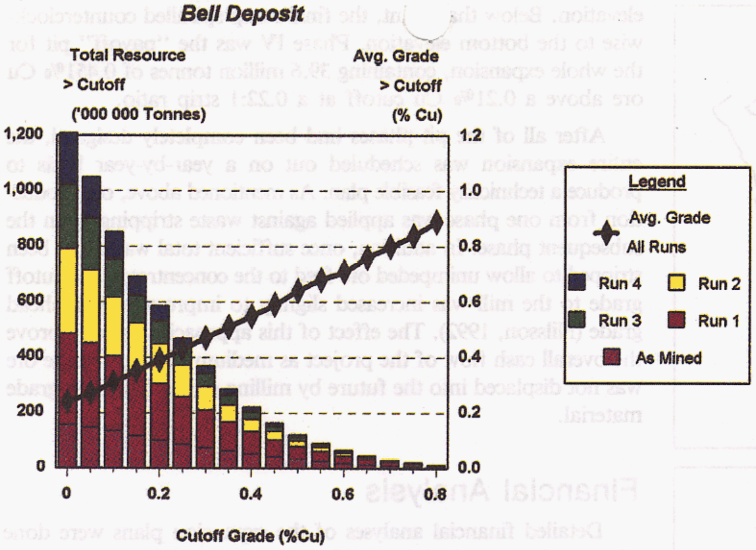


FIGURE 19. Plan view 1500 Bench (450 m asl), Bell copper block model. Colour coding is same as in Figure 18.



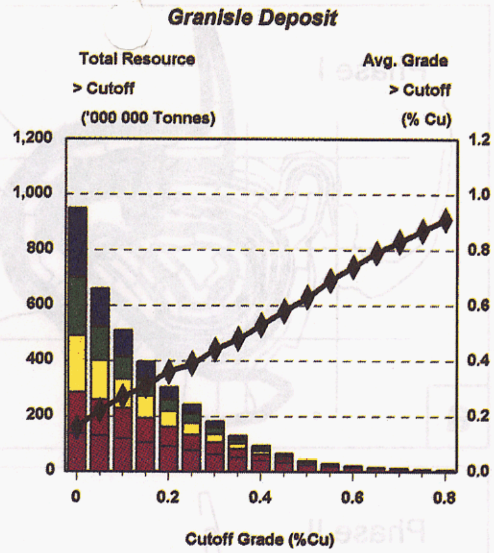


FIGURE 20. Geological resource vs copper cutoff grade at Bell and Granisle showing a summation of all blocks contained within the interpolation volume plus the mined out volumes and sorted by run number. Note larger overall resource size at Bell (almost 600 million tonnes at a 0.20% Cu cutoff) versus Granisle (just over 300 million tonnes at a 0.20% Cu cutoff). Also note the greater proportion of resource contributed by runs 1 to 3 (proven and probable) at Bell versus Granisle, reflecting the density of diamond drilling carried out on each deposit. The extreme sensitivity of the Bell deposit economics is partially accounted for by the steepness of the contained ore tonnage curve in the range of economic cutoff (≈0.20% Cu) such that small changes in cutoff drastically affect the relative volumes of material that reports as ore versus reporting as waste.

Working Pit Designs and Pit Scheduling

Workable pit designs were created by smoothing the computer-generated floating-cone optimum pit, adding haulage ramps and scheduling pit production. The tool used to do this was an in-house program running inside the AutoCAD® environment. The program ensured that all geotechnical and engineering design criteria were met from a drafting point of view, while allowing the mine engineer to concentrate on the reasoning behind the resulting pit design.

A number of pit design options were created during the Bell 2000 project. The initial step was to design the ultimate pit limit and ultimate haulage ramp locations. The ore and waste volumes and grades were then quantified on a bench-by-bench basis at different cutoffs for the ultimate pit boundaries.

The next step was to design and schedule a series of phased pits which would result in the ultimate pit being extracted. The objectives of the phased designs were:

- to ensure haulage access at all times to the working benches within the phased pit while minimizing the need for temporary ramps;
- to ensure adequate working widths were maintained allowing maximum productivity and minimizing mining costs;
- to allow each phase to be mined as independently as possible from the next phase, reducing operational interference between the phases; and
- to ensure sufficient volumes within each phase to provide mill feed for at least two years.

A number of strategies were investigated, including whether it was better to maintain mill production throughout the expansion process or to shut the concentrator down for a period of time to allow significant volumes of waste to be prestripped. In the end, it was determined that maintenance of concentrate production and resultant revenues was preferable to a shut down. The general philosophy of phase development was that the ore from one phase must provide sufficient mill feed to ensure that the waste from the next phase was removed to the point that this next phase could sustain ore production.

The final design and schedule produced during the Bell 2000 project was known as the Pit 8 design (Fig. 21). Pit 8 used the Rob Cut and upper 16-zone ore as phase 0 or the initial ore supply to the mill during the pit expansion.

Phase I targeted ore relatively high on the east wall. Phase I was to be serviced by two ramps: an ore ramp entering the pit near the primary crusher and descended the south wall in a counterclockwise spiral and a waste ramp which exited the pit in the middle of the north wall and spiralled clockwise down the north and east walls. These two ramps would meet at the 2180 bench in the southeast corner of the pit and become permanent parts of the ultimate ramp design. Below this elevation, a temporary ramp would access the ore on the east wall. In total, phase I contained 14.5 million tonnes grading 0.365% Cu above a 0.21% cutoff with a 2.50:1 strip ratio.

Phase II targeted ore at moderate depth on the south wall of the pit as well as providing the first significant deepening of the pit bottom below the existing configuration. Large volumes of waste across the entire south wall would be stripped and another section of the permanent pit ramp would be established from the intersection of the primary ore and waste ramps on the 2180 bench spiralling clockwise down on the south wall to the 1940 elevation. Below 1940 the remainder of the phase II ramp would be temporary and switched back and forth down the south wall to an intermediate pit bottom at the 1420 elevation. Phase II contained 16.9 million tonnes grading 0.459% Cu above a 0.21% cutoff with a 2.25:1 strip ratio.

Phase III would mine the remainder of the 16-zone and strip the west and north walls down to the 1940 elevation. The phase III ramp would crest at the same location as the primary ore ramp near the primary crusher but would descend clockwise down the west wall to a switchback at the 2180 elevation on the north wall, roughly vertically below where the main waste ramp would enter the pit. The portion of the phase III ramp below 2180 down to the same location as the 1940 switchback on the phase II ramp would be temporary and be mined out during the last phase of the pit expansion. Phase III contained 13.8 million tonnes grading 0.278% Cu above a 0.21% cutoff with a 2.65:1 strip ratio.

The final phase of the mining plan would complete the extraction of the ultimate pit. Phase IV would continue mining below the phase III pit and the phase II ramp as well as deepening the pit bottom to the ultimate 1180 elevation (more than 400 m below the pit rim on the north wall). The phase IV ramp would continue clockwise around on the west wall from the old phase II switchback at 1940 to a new switchback on the north wall at the 1720

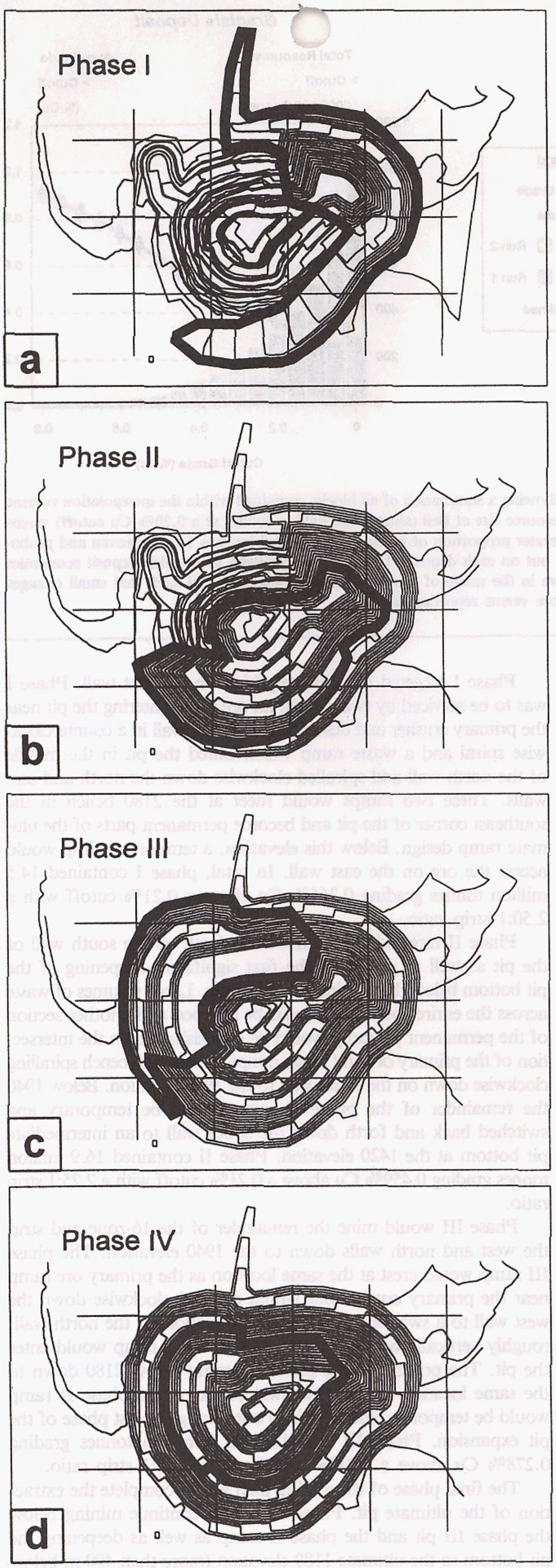


FIGURE 21. Bell Pit 8 design phases showing scheduled sequence of extraction used for financial analysis during the Bell 2000 expansion study.

elevation. Below that point, the final ramp spiralled counterclockwise to the bottom elevation. Phase IV was the "payoff" pit for the whole expansion, containing 39.6 million tonnes of 0.451% Cu ore above a 0.21% Cu cutoff at a 0.22:1 strip ratio.

After all of the pit phases had been completely designed, the entire expansion was scheduled out on a year-by-year basis to produce a technically feasible plan. As mentioned above, ore production from one phase was applied against waste stripping from the subsequent phase. In addition, once sufficient total waste had been stripped to allow unimpeded ore feed to the concentrator, the cutoff grade to the mill was increased slightly to improve average head grade (Nilsson, 1992). The effect of this approach was to improve the overall cash flow of the project as medium and high-grade ore was not displaced into the future by milling of marginal low-grade material.

Financial Analysis

Detailed financial analyses of the expansion plans were done after production schedules had been produced. Operating expenses for mining, milling and general site costs were estimated from the scheduled production volumes. These numbers were forecast from zero-based engineering calculations wherever practical (e.g., haulage cycles and drill factors). Capital costs were estimated for new and replacement mining and milling equipment, and tailings construction, and were distributed in time to meet schedules.

A number of options were explored during the Bell 2000 project (Allan, 1991). These options included using supplementary mill feed from Granisle or Morrison, contract prestripping, in-pit crushing and conveying of ore, run-of-mine conveying of waste, alternative tailings deposition options and acid dump leaching with solvent extraction and electrowinning (SX/EW) to produce copper metal from low-grade material. Very large expansion options (expanded mill production rates and very large pits) were studied but suffered due to the time lag between initial capital outlay and positive revenue from mining higher grade ore in the pit bottom. Conversely, very small expansion options (minimal pushbacks) suffered from insufficient revenue over the life of the project to justify any capital expenditures, low productivity due to minimal mining widths and an inability to develop significant volumes of ore in the pit bottom.

To improve cash flow, a floating cutoff grade ranging from 0.21% to 0.25% Cu was applied to the Pit 8 expansion. This increased the grade of the resource but reduced the tonnage and increased the strip ratio. The "final" open pit resource at Bell was 70.4 million tonnes of ore grading 0.44% Cu and 0.20 g/t Au at a 1.9:1 strip ratio. This resource did not satisfy Noranda Minerals' minimum financial requirements for development in March, 1992.

The Bell evaluation was extremely sensitive to a number of operating parameters. The size of the geologic resource was substantial (Fig. 20) but development was handicapped by the amount of waste stripping required to expose sufficient mill feed in a timely fashion. The project generated positive cash flow on an operating cost basis but failed to meet financial hurdles on a discounted cash flow basis. Had the full extent of the Bell deposit been realized early, perhaps a different extraction plan would have resulted in a very different final pit.

Acknowledgments

Many individuals, including prospectors, geologists, engineers and miners have contributed to the Bell-Granisle story. The authors wish to acknowledge all these individuals for their very significant contributions. In addition, the authors are grateful for the advice and encouragement received from J. Kalmet, D.J.T. Carson, J.D. Williams and M.J. Gray as a result of their reviews of early versions of this paper. The authors wish to thank Noranda Mining and Exploration Inc. for permission to present this paper.

REFERENCES

ALLAN, M.J., 1991. Bell Copper project 2000: prefeasibility study of expansion options. Unpublished company report, Noranda Minerals Inc., Toronto, Ontario.

BURNHAM, C.W., 1985. Energy release in subvolcanic environments: implications for breccia formation. Economic Geology, 80, p. 1515-1522.

CARSON, D.J.T. and JAMBOR, J.L., 1974. Mineralogy, zonal relationships, and economic significance of hydrothermal alteration at porphyry copper deposits, Babine Lake area, British Columbia. The Canadian Institute of Mining and Metallurgy, CIM Bulletin, 67, No. 742, p. 110-113.

CARSON, D.J.T., JAMBOR, J.L., OGRYZLO, P.L. and RICHARDS, T.A., 1976. Bell Copper: Geology, geochemistry, and genesis of a supergene-enriched, biotitized porphyry copper deposit with a superimposed phyllic zone. *In* Porphyry Deposits of the Canadian Cordillera. *Edited by* A. Sutherland Brown. Canadian Institute of Mining and Metallurgy, Special Volume 15, p. 245-263.

CARTER, N.C., 1966a. Northern Babine Lake area. *In* British Columbia Minister of Mines and Petroleum Resources, Annual Report 1965, p. 90-93.

CARTER, N.C., 1966b. Newman. *In* British Columbia Minister of Mines and Petroleum Resources, Annual Report 1965, p. 99-102.

CARTER, N.C., 1972. Granisle Mine. *In* Geology, Exploration and Mining in British Columbia 1971. British Columbia Department of Mines and Petroleum Resources. p. 178-183.

CARTER, N.C., 1974. Geology and geochronology of porphyry copper-molybdenum deposits, west-central British Columbia. Unpublished Ph.D. thesis. The University of British Columbia, Vancouver, British Columbia.

CARTER, N.C., 1982. Porphyry copper and molybdenum deposits, west-central British Columbia. Ministry of Energy, Mines and Petroleum Resources of British Columbia, Vancouver, British Columbia, Bulletin 64, 150 p.

CARTER, N.C. and CLARKE, W.G., 1966. Granisle Copper Limited. *In* British Columbia Minister of Mines and Petroleum Resources, Annual Report 1965, p. 93-99.

CLARKE, W.G., 1967. Granisle Mine. *In* British Columbia Minister of Mines and Petroleum Resources 1966, Annual Report, p. 97-99.

CUDDY, A.S. and KESLER, S.E., 1982. Gold in the Granisle and Bell Copper porphyry copper deposits, British Columbia. *In* Precious Metals in the Northern Cordillera. The Association of Exploration Geochemists, p. 139-155.

DENNIS, J.G., 1987. Structural Geology. Wm. C. Brown.

DIROM, G.E., 1962. Newman Group. Unpublished company report, Noranda Exploration Company, Limited, Vancouver, British Columbia.

DIROM, G.E., 1964. Report on the Newman property, Babine Lake (Smithers), British Columbia. Unpublished company report, Noranda Exploration Company, Limited, Vancouver, British Columbia.

DOLMAGE, V., 1943. Newman copper deposit, Babine Lake, British Columbia. Unpublished company report, Noranda Exploration Company, Limited, Vancouver, British Columbia.

EMMENS, N.W., 1914. Babine Lake. *In* British Columbia Minister of Mines, Annual Report 1913, p. K112-K114.

ETHIER, M.C. and LARAMIE, A.L., 1990. Performance improvement at the Bell mine. Canadian Institute of Mining and Metallurgy, CIM Bulletin, 83, No. 939, p. 79-83.

FAHRNI, K.C., 1967. Geology of the Granisle ore zone. Western Miner, 40, No. 5, p. 34-38.

FAHRNI, K.C., KIM, H., KLEIN, G.H. and CARTER, N.C., 1976. Granisle. *In* Porphyry Deposits of the Canadian Cordillera. *Edited by* A. Sutherland Brown, Canadian Institute of Mining and Metallurgy, Special Volume 15, p. 239-244.

FEISS, P.G., 1978. Magmatic sources of copper in porphyry copper deposits. Economic Geology, 73, p. 397-404.

GABRIELSE, H. and Y[illegible]ATH, C.J., 1989. Tectonic evolution of the Canadian Cordi[illegible] *In* DNAG #4. The Cordilleran Orogen in Canada. Geoscience Canada, Volume 16, Number 2.

GILL, J.B., 1981. Orogenic andesites and plate tectonics. Springer Verlag, 390 p.

GUILBERT, J.M. and PARK, C.F., 1986. The geology of ore deposits. W.H. Freeman and Company, New York.

HALL, J.A. and KRAFT, J.E., 1969. A preliminary feasibility study of the Newman copper property. Unpublished company report, Noranda Exploration Company, Limited, Toronto, Ontario.

HARINGTON, C.R., TIPPER, H.W. and MOTT, R.J., 1974. Mammoth from Babine Lake, British Columbia. Canadian Journal of Earth Sciences, 11, p. 285-303.

KIRKHAM, R.V., 1971. Intermineral intrusions and their bearing on the origin of porphyry copper and molybdenum deposits. Economic Geology, 66, p. 1244-1249.

MONGER, J.W.H., SOUTHER, J.G. and GABRIELSE, H., 1972. Evolution of the Canadian Cordillera, A plate-tectonic model. America Journal of Science, 272, p. 577-602.

NEWELL, J.M., CARTER, N.C. and SUTHERLAND BROWN, A., 1995. Porphyry deposits of the northwestern Cordillera: A retrospective. *In* Porphyry Deposits of the Northwestern Cordillera of North America. *Edited by* T.G. Schroeter. Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46.

NILSSON, J., 1992. Bell Mine. Review of development options. Unpublished company report, Noranda Minerals Inc., Toronto, Ontario.

O'KEEFE, B., 1993. The mines of Babine Lake. Bell mine, Noranda Minerals Inc.

OGRYZLO, P.L., DIROM, G.E. and STOTHART, P.G., 1995. Morrison - Hearne Hill copper-gold deposits, Babine region, west-central British Columbia. *In* Porphyry Deposits of the Northwestern Cordillera of North America. *Edited by* T.G. Schroeter. Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46.

OGRYZLO, P.L., in prep. Hearne Hill, British Columbia, Canada: Collapse brecciation in a continental volcano-plutonic arc. M.Sc. thesis. University of Regina, Regina, Saskatchewan.

PARLIAMENT, J.H., 1964. Granisle Copper Limited feasibility report. Unpublished company report, The Granby Mining Company Limited, Vancouver, British Columbia.

PITEAU, D.R. and MARTIN, D.C., 1977. Granisle Copper Limited Report on slope stability analysis and design of the open pit slopes. Unpublished company report, Granisle Copper Limited, Vancouver, British Columbia.

RICHARDS, T.A., 1988. Geologic setting of the Stikine terrane. Abstract. *In* Geology and Metallogeny of Northwestern British Columbia. Smithers Exploration Group, Geological Association of Canada Cordilleran Section.

STOTHART, P.G., 1991. Computerized geologic modelling and mine design: Bell mine Project 2000. Paper Presented to 4th annual Noranda mining and geology computer users group seminar, Saskatoon, Saskatchewan.

TIPPER, H.W. and RICHARDS, T.A., 1976a. Jurassic stratigraphy and history of north-central British Columbia. Geological Survey of Canada, Bulletin 270.

TIPPER, H.W. and RICHARDS, T.A., 1976b. Geology of the Smithers map area, British Columbia. Geological Survey of Canada, Open File 351.

WHEELER, J.O. and McFEELY, P., 1991. Tectonic assemblage map of the Canadian Cordillera and adjacent parts of the United States of America. Geological Survey of Canada, Map 1712A.

WILSON, J.W.J., KESLER, S.E., CLOKE, P.L. and KELLY, W.C., 1980. Fluid inclusion geochemistry of the Granisle and Bell porphyry copper deposits, British Columbia. Economic Geology, 75, p. 45-61.

ZALUSKI, G., 1992. Hydrothermal alteration and fluid sources associated with the Babine Lake porphyry copper deposits, west-central British Columbia. Unpublished M.Sc. thesis, University of Alberta, Edmonton, Alberta.