that the mineralizing solutions were either rich in these elements or that vast quantities of fluid were responsible for deposition of these elements in the lodes. There has been some mobilization of Co, Cu, Ni and Zn but no overall introduction of these elements. Nor, according to the work of Bartram (1969), has there been any overall introduction of SiO₂, Fe or H₂O.

Some variation in the nature of the alteration can be detected in progressing from the southern to the northern part of the field. In the most southerly example of mineralization studied (from D.D.H. S.E.15), Na metasomatism dominates over K metasomatism. This contrasts with the Golden Mile area which is characterized by intense K metasomatism and leaching of Na and Ca. In the northern part of the field, K metasomatism again becomes less important, especially in the Hannans North Lode where introduction of K is negligible and Ca and Na metasomatism prevail (Chapter 4.2.1).

5.2.4 Regional Carbonation

Introduction of CO₂ has occurred on a much larger scale than that represented by the K-metasomatism of the lodes and is almost regional in its extent. Bartram (1969) and Bartram and McCall (1971) have demonstrated that the CO₂ metasomatism post-dates the peak of regional metamorphism thus eliminating the possibility that the CO₂ is deuteric.

Since there is very little difference in the Au, Ag, Te, Se, As and Sb contents of carbonated (>1% CO₂) and non-carbonated (<1% CO₂) samples of G.M.D. (Table 47), regional carbonisation has had little effect on the Au content of the G.M.D. However, in some lodes and veins, intense CO₂ metasomatism is superimposed on the regional carbonatization and this is almost certainly related to the Au mineralization.

5.2.5 Stratigraphic Distribution of the Ore Deposits

Hannans Lake Serpentine and Devon Consols Basalt

As far as the author is aware, no ore deposits have been found within
these units either in drill core or where they outcrop in the northern and southern parts of the field (Fig. 2, in pocket). In the Golden Mile, these units are buried deeper than the deepest workings and could have, at most, only an indirect association with the lodes.

**Kapai Slate**

A number of small mines were worked in the Kapai Slate (the interflow sediments between the Devon Consols Basalt and Williamstown Dolerite). The most productive of these was the Devon Consols Mine (Fig. 2) where, according to Feldtmann (1916), rich patches of gold occurred at the junction of cross-cutting quartz veins with a belt of jasper. On the 237' drive of this shaft, the jasper graded into a graphitic schist indicating that the jasper merely represented oxidized and silicified pyritic black shales. 27205 tonnes of ore from this mine yielded 363.46 kg of gold (13.4 g/tonne).

Other workings in the Kapai Slate included the Kapai, Sir John, Sons of Gwalia and possibly the Red White and Blue in the northern part of the field (Fig. 2) and the Hazeldon mine, just to the west of the Golden Pike Fault in the central part of the field (Fig. 3, in pocket). According to Feldtmann and Farquharson (1913), the gold in these mines was present both in quartz veins and in the "ironstone" (oxidized sediments).

**Williamstown Dolerite**

The author was unable to confirm the association of any mineralization with the Williamstown Dolerite. The Lady Elizabeth shaft appears to have been put down in Williamstown Dolerite (Fig. 2) but porphyrites and jasper (oxidized sediments) were also present in the workings according to Feldtmann (1916); in any case, no production has been recorded from this shaft. The geology of the Hidden Secret mine is uncertain but it could be situated at least partly on the faulted contact between the Paringa Basalt and the
Williamstown Dolerite. The Williamstown Dolerite is not present in any of the Golden Mile workings although it has been penetrated by deep drilling in the core of the Kalgoorlie Anticline (G.A. Travis, pers. comm.).

**Paringa Basalt**

Some lodes of the Eastern Lode System extend into the Paringa Basalt for hundreds of metres (see Figs. 2 and 3). According to Finucane (1964), these lode structures are "influenced by a series of flatly south-pitching planes which are possibly bedding planes between individual flows" of the Paringa Basalt.

Paringa Basalt at the northern end of the field is also mineralized. Telluride-bearing ore was obtained from sheared Paringa Basalt in the Creswick shaft (Feldtmann, 1916) and the rich telluride ores of the Hidden Secret mine appear to have been predominantly in the Paringa Basalt. Gold mineralization in the North Collier mine was associated with interflow sediments near the base of the Paringa Basalt. According to Feldtmann and Farquharson (1913), the richest ore in this mine was won from places where pale, slatey material which they assumed to represent a line of movement, intersected graphitic "indicators" (i.e. black shales); 158 tonnes of this ore yielded 19.15 kg of Au (i.e. 121 gm/tonne), an amount negligible in terms of the total production of the Kalgoorlie goldfield but significant in terms of its location and richness.

Tomich (1959, 1974, 1976) has emphasised the close association of "green leader" ore with tuffite and graphitic sediments along the flatly plunging interface between the Golden Mile Dolerite and the underlying Paringa Basalt and has concluded that the "green leader" represents a volcanogenic ore bed. No evidence has been found to date that the "green leader" actually is a sediment (in many places it simply looks like sheared Paringa Basalt) but the strong localization of ore deposits at this position suggests that there may be some stratigraphic control.
G.M.D.

Most of the Au won from the Kalgoorlie area has come from the steeply dipping lodes within G.M.D. adjacent to the Boulder Dyke in the Golden Mile area. These lodes are not however strictly confined to the G.M.D.; many of the Western Lodes pass laterally or with depth into the Boulder Dyke (Figs. 3, 10, 58C) whilst in the Eastern Lode System, the Lake View Lode is related to the Duck Pond Slates (infolded B.F.B.) and others (including the Kalgurli) are also related to sediments according to Tomich (1974).

The Mt. Charlotte and Golden Pike orebodies are enclosed entirely in the G.M.D. and are best developed in unit 8 probably because it is the most competent of the units.

The Hannans North and related lodes are also confined to the G.M.D. as are all the Au-bearing samples from the South End Drilling.

B.F.B.

One of the richest orebodies in the field, known as the "Duck Pond" shoot was associated with a lens of slate which Campbell (1953) and Travis and Woodall (1975) have interpreted as a tight synclinal infold of the B.F.B. (Chapter 2.4.1).

A number of the Main Lodes in the Western Lode System penetrate the B.F.B. of the Boulder Dyke laterally and with depth. Stillwell (1931) reports that segregations of chalcopyrite, tellurides and enargite were present in graphitic slate from the 2360 ft. (719 m) level of the Great Boulder mine but in general, the extensions of the lodes in the Boulder Dyke are poorly mineralized. One of the few examples of a lode enclosed entirely in the B.F.B. of the Boulder Dyke is the Golden Dream orebody at the northern end of the field (Fig. 2, in pocket).

At the southern end of the Golden Mile, mineralized quartz veins were present at the contact of the G.M.D. and B.F.B. in the Idaho mine (Finucane,
1948) and Simpson (1912, p.80) described a specimen from the Forrest King Mine as "a schistose rock carrying over 1½ ounces of gold per ton and sufficient finely granular graphite to give an unusually dark colour to the rock both in mass and in thin sections".

Further south, the only occurrence of Au mineralization in the B.F.B. known to the author, is from drilling carried out by International Nickel Australia Limited (Chapter 2.4.3).

Porphyries and Porphyrites

In some of the mines, for example, the South Kalgurlie and Lake View mines in the Golden Mile and the Mt. Charlotte and Mystery mines in the northern part of the field, there is a spatial relationship between porphyries or porphyrites and mineralization.

Stillwell (1929) concluded that mineralization was the result of discharge of solutions from the cooling porphyry mass. However, as Gustafson and Miller (1937) point out, there are some porphyries without associated mineralization and many orebodies have no spatial relationship to porphyry dykes.

As mentioned in Chapter 2.2.9, some of the rocks described as porphyries appear to be tuffites; others are probably the intrusive equivalents of the felsic volcanics of the B.F.B. but some may be later than, or synchronous with, folding and metamorphism. However, since even the transgressive porphyries (e.g. those in the South Kalgurlie mine and the Mt. Charlotte mine) are cut by mineralized veins, it would appear that mineralization post-dated porphyry intrusion.

5.2.6 Mass Balance Calculations

To date, Au production from the Kalgoorlie goldfield is $1.5 \times 10^6$ kg. However, it is likely that a considerable amount of Au has been lost by erosion and that there is still much left in the ground. A figure of
$2 \times 10^6$ is probably a better estimate of the true amount of Au introduced into the Kalgoorlie system.

If it assumed that the average trace element ratios found for the Eastern and Western Lode Systems (Table 49) approximate those for all of the Kalgoorlie mineralization, then approximately $5 \times 10^6$ kg Te, $2 \times 10^6$ kg Ag, $1.5 \times 10^8$ kg As, $2 \times 10^7$ kg Sb, $2 \times 10^6$ kg Se and $4 \times 10^{10}$ kg S have also been introduced.

If the country rocks were the source of these elements, then the original concentrations of these elements in the rocks must have been such that the elements could be derived from a geologically reasonable volume of rock.

On the hypothesis that the present day levels of Au in the least altered rocks from the South End Drilling represent the primary Au contents of the rocks, rough mass balance calculations have been carried out to determine whether or not the altered equivalents of these rocks in the mining area could have been the source of the Au in the lodes. The results of the calculations are presented in Table 65.

When the Au contents of the least altered samples of all mafic and ultramafic units of the Kalgoorlie Succession (from Tables 50 and 51) are averaged out according to their stratigraphic thickness, a value of 1.3 ppb Au is obtained. Even if Au was completely leached from these rocks, a block of volcanics the length of the field (8 km) by the maximum width (2 km) by a depth of 32 km would be required to provide all of the Au in the system. Leaching of Au would not be expected to extend below 10 km and thus, at most the volcanics could supply only 30% of the Au.

Because of the high average Au content of the interflow sediments, a considerably smaller volume of these sediments would be required to produce all the Au in the system. However, the interflow sediments are so thin that
<table>
<thead>
<tr>
<th>Rock type</th>
<th>Maximum stratigraphic thickness (m)</th>
<th>Au (ppm)</th>
<th>Volume of rock for total Au (km$^3$)</th>
<th>Maximum feasible dimensions (km$^3$)</th>
<th>Maximum feasible volume (km$^3$)</th>
<th>Maximum % of total Au</th>
</tr>
</thead>
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<tr>
<td>Hannans Lake Serpentinite</td>
<td>500</td>
<td>0.82</td>
<td></td>
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<tr>
<td>Williamstown Dolerite</td>
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<td>Devon Consols Basalt</td>
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<td>G.M.D.</td>
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<tr>
<td>Stratigraphically adjusted average</td>
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<td>8 x 2 x 10</td>
<td>160</td>
<td>30</td>
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<tr>
<td>Interflow Sediments</td>
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<td>90</td>
<td>8</td>
<td>8 x 0.01 x 10</td>
<td>0.8</td>
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<td>B.F.B. in Boulder Dyke:</td>
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<td></td>
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<tr>
<td>Black shales</td>
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</tr>
<tr>
<td>Other sediments</td>
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<td>Tuffs and porphyries</td>
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<td></td>
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<tr>
<td>Average</td>
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<td>8 x 0.09 x 10</td>
<td>7.2</td>
<td>10</td>
<td></td>
</tr>
<tr>
<td>B.F.B. overlying the ore-deposits</td>
<td>3000?</td>
<td>10?</td>
<td>70</td>
<td>8 x 2 x 3</td>
<td>48</td>
<td>70</td>
</tr>
</tbody>
</table>

Au calculations based on an average country rock density of 2.88 g/cm$^3$ (Larcombe, 1911).
Total Au in the Kalgoorlie ores estimated at $2 \times 10^6$ kg.
Length of field = 8 km.
Width of field = 2 km.
Width of Boulder Dyke = 0.09 km.
Maximum stratigraphic thicknesses from Travis and Woodall (1975).
only about 10% of the Au in the lodes could be derived from them at their present average Au content. The amount would be even less if the one high value in Table 51 is rejected from the average.

The B.F.B. are difficult to treat quantitatively because of the complex structural setting of these sediments, the variable Au contents, the large number of lithotypes and the facies changes between the central and south-eastern parts of the field. According to the plans of Stillwell (1929), the Boulder Dyke varies in width from about 25 m to 180 m in the Golden Mile area. An average of twenty traverses marked on Stillwell's plans gives a width of 90 m of which only about 30 m are marked as slate. If it is assumed that half of the remainder consists of other sedimentary types (carbonate-bearing sediments, laminated shales, mud-chip conglomerates, evaporite-bearing sediments etc.) and the other half of tuffs and porphyries, rough mass balance calculations can be carried out. These indicate that the Boulder Dyke could have provided only about 10% of the total Au on the assumption that the present Au content of the B.F.B. measured reflects the initial Au content of the B.F.B. in the mining area. Although there is no perceptible decrease in width of the Boulder Dyke with depth on the mine plans of Stillwell, it could well die out before a depth of 10 km, in which case, an even smaller percentage of the total Au could be provided by the infolded sediments of the B.F.B. at their present Au levels.

If a mechanism was available for leaching Au from the overlying 3 km of B.F.B. and depositing it in the lodes, then theoretically the B.F.B. could provide up to 70% of the total Au if the Au concentrations in the upper parts of the B.F.B. are similar to those in the lower part, a rather tenuous assumption in view of the observed facies changes.

It can be concluded that on the basis of the present average Au content of the Kalgoorlie Succession at the southern end of the field, none of the
country rocks could have provided all of the Au in the lodes and veins. This implies either that at least some of the Au was derived from an external source or that the present average Au content of the rocks analysed is not a reflection of the primary Au content of these rocks. These alternatives will be discussed in a later section.

It should be noted however, that if the present average composition of the B.F.B. at the southern end of the field represents the primary composition of the B.F.B., then the sediments infolded into the Boulder Dyke alone, could have provided approximately $7.6 \times 10^6$ kg Ag, $5.2 \times 10^6$ kg Te, $2.4 \times 10^9$ kg As, $5.6 \times 10^7$ kg Sb, $3.8 \times 10^7$ kg Se and $2.6 \times 10^{13}$ kg S; these values are greater than the estimated amounts in the Kalgoorlie ores. Although it is not possible to carry out mass balance calculations for CO$_2$ and K, it is probable that the B.F.B. could also have provided all the CO$_2$ and K in the ores as many of the sediments are rich in carbonate and sericite.

It is obvious that the sediments with evaporite-pseudomorphs have already lost virtually all of their S, even at the southern end of the field. If the evaporite bearing sediments are assumed to be 40 m thick, 1000 m wide and 10 km in length as suggested by their distribution (Chapter 2, p.81), and these rocks originally contained an average of 20% gypsum (3.1% S) as indicated by the volume of pseudomorphs present, then about $4 \times 10^9$ kg of S, or an amount equivalent to 10% of the total S in the Kalgoorlie ores has been lost from these sediments at some stage in their history.

5.2.7 Spatial Variation in Mineralogy and Geochemistry Across the Kalgoorlie Goldfield

Morphologically, the quartz-stockwork ores of the Mt. Charlotte orebody and the siliceous lodes of the Golden Mile bear little similarity to each other. However, in each, pyrite is the major sulphide present, although it
is coarser grained in the Mt. Charlotte ores than in the Golden Mile ores. Both native gold and tellurides are present in both types of mineralization but whilst gold tellurides are abundant in the Golden Mile, altaite was the only telluride positively identified in the Mt. Charlotte ore where native gold is obviously the dominant Au-bearing mineral. Although the Mt. Charlotte samples as a whole have a lower Te/Au ratio than the lodes, samples from the lower levels of the Mt. Charlotte orebody have Te/Au ratios similar to those of samples from the upper levels of the Western Lode mines. Ag/Au ratios of the two types of mineralization also overlap. Sulphosalts are much rarer in the Mt. Charlotte ores than in the lode ores and this is reflected in the higher Sb/As and Sb/Au ratios of the lode samples. Other geochemical differences between the Mt. Charlotte and the Golden Mile ores are the lower V/Au, V/S and generally lower Te/Se ratios in the former.

Mineralogically and morphologically, the Mt. Charlotte and Golden Pike ores are very similar; they share some geochemical characteristics (low Te/Au, Sb/Au, Te/Se and V/Au ratios and high S/Se ratios) but the Golden Pike ores differ from the Mt. Charlotte ores in having much higher Sb/As ratios and slightly higher V/S ratios (Chapter 4.2.2).

The Kalgoorlie Enterprise ores are mineralogically and geochemically intermediate between the Mt. Charlotte and the Golden Mile ores; they resemble the Mt. Charlotte ores in their high S/Se ratio and the lode samples in their high Sb/As and Te/Se ratios.

Although there is a progressive change in wall rock alteration from the Golden Mile through the Mt. Charlotte orebody to the Hannans North Lode in the northern part of the field, this is not reflected in the trace-element geochemistry. The high Sb/As, Sb/Au and V/S ratios in the Hannans North Lode are closer in value to those of the Golden Mile lodes than they are to the Mt. Charlotte ores (Chapter 4.2.2). Massive telluride ore was
mined from the Hidden Secret ore deposit in the northern part of the field indicating that this type of mineralization was not restricted to the Golden Mile; the tellurides in this deposit were however, predominantly Ag-rich tellurides (hessite and petzite) whereas Au-rich tellurides (calaverite and krennerite) predominate in the Golden Mile ores (Chapter 3).

A narrow Au-telluride-bearing interval was intersected in drilling south of the Golden Mile. Except for its unusually low As/Au, Sb/Au and V/Au ratios and high Sb/As ratio, this interval has a close geochemical affinity with the Golden Mile Lodes. Another sample from the south end drilling bears a closer resemblance to the Mt. Charlotte ore but differs in its exceptionally low S/Se ratio (Chapter 4.2.2).

Within the Golden Mile, no significant difference was detected mineralogically or geochemically between the steeply dipping lodes of the Eastern and Western Lode Systems.

The flatly dipping lode samples have higher V/S and higher absolute V contents than the steeply dipping lode samples analysed (Chapter 4.2.2) but abundant V-bearing minerals have been reported from some steeply dipping lodes (3.2.1) indicating that V is not restricted to the flatly dipping lodes. With the exception of V, the steeply dipping and flatly dipping lodes are geochemically similar and tellurides are abundant in both lode types.

There are no clear trends in the spatial distribution of the common telluride phases at Kalgoorlie (Fig. 96). However, all observed occurrences of montbrayite or antimonian montbrayite have come from the outer fringes of the field (the "green leader" lodes, Doolittle Lode, Kalgoorlie Enterprise Drilling and South End Drilling) and the only mattagamite and frohbergite found came from D.D.H. S.E.15 at the southern end of the field (Chapter 3.3.1).
5.2.8 Vertical Variations in the Mineralogy and Geochemistry of the Ore Deposits

In the Western Lodes, there is an overall increase in the Sb/Au ratio of the tetrahedrite group of minerals (Chapter 3.3.3) and an increase in the Ag/Au ratio in the native gold (3.3.2) with depth. There is no evidence of any variation in the telluride assemblages with depth: all common tellurides are found throughout the vertical range of mineralization. There is an increase in the Te/Au, Ag/Au, Sb/As and Sb/Au ratios of the whole rocks with depth (4.2.2) corresponding with the mineralogical changes.

In the Mt. Charlotte orebody, the composition of the gold shows a strong dependence on its paragenesis and the variations between different generations of gold were too great to enable any variation with depth to show up. No other mineralogical variation with depth was noted in this deposit, sulphosalts being virtually absent. Geochemically, the Mt. Charlotte ores show an overall decrease in Te/Au, Ag/Au, Sb/Au and Te/Se ratios with depth; i.e. the exact opposite of the variation found in the Western Lodes.

5.2.9 Occurrence of Natural Gas in the Mines

Montgomery (1905) reported that an inflamable gas was encountered when drilling at the 1048 ft. (319 m) level of the Northern Associated mine. Methane was the principal constituent of the gas. Montgomery concluded that this gas was derived from carbonaceous matter in the country rock. Simpson (1912) reported further occurrences of inflammable gas from workings in graphitic slates in the Boulder Dyke.

In this study, hydrocarbons were found in a pyritic black shale from the B.F.B. at the southern end of the field (Chapter 2.2.8).

5.2.10 Geochronology

The Pb and by inference the other ore constituents are about 2700 m.y.
(+ 50 m.y.?) old, the same age as the volcanogenic-sedimentary pile.

Rb/Sr dating of the porphyries by Turek (1966) gave an age of $2670 \pm 90$ m.y. but unfortunately, it is not certain whether the porphyries dated by Turek were intrusive or merely tuffaceous sediments, and hence whether the age reflects that of intrusion or that of deposition of the B.F.B. Similar porphyries at Kambalda which are known to be intrusive have an age of $2630 \pm 45$ m.y. whilst more plagioclase-rich porphyries give an age of $2400$ m.y. according to the Pb isotope data of Roddick (1974). Oversby (1975) has dated the Mungari granite, the closest granite to Kalgoorlie, at $2640 \pm 35$ m.y. using Pb isotopes. The granites and porphyries thus appear to be just slightly younger than the Pb in the ores. A secondary isochron suggests that Pb was mobilized and mixed with radiogenic Pb at about $2600$ m.y.; scatter about this isochron suggests that mineralization either took place over a period of time or that there were variations in the initial Pb-Pb ratios in the source rocks. Turek (1966) obtained an age of $2460$ m.y. for the Kalgoorlie mineralization but no conclusive evidence of a mineralization event at this time was found in this study.

5.3 Discussion

The Kalgoorlie deposits were extremely rich and the concentration of such a large amount of Au obviously required very special geological conditions. In view of the extensive wall rock alteration about the lodes and veins it can be inferred that large quantities of fluid were responsible for the transport of the ore constituents into their present sites. Pb isotopes indicate that the Pb, and by inference, the Au, in the ores is the same age as the country rocks (approximately 2700 b.y.) providing strong evidence that the country rocks were the source of the Au and other ore constituents. Various source rock models are examined in detail below.
(a) **The Mafics and Ultramafics as Source Rocks**

The mafic and ultramafic members of the Kalgoorlie Succession presently contain an average of 1.3 ppb Au, less than the average crustal abundance of this metal (1.8 ppb according to Shaw *et al.*, 1976) and, at these levels, are most unfavourable source rocks. Even if it was possible to completely leach Au from these rocks, an unreasonably large volume of rock would be required. Furthermore, as pointed out by Keays and Scott (1976), the availability of Au is a function of the mineralogical siting of the metal and not the total concentration; at Au levels as low as those in the Kalgoorlie mafics and ultramafics, much of the Au is likely to be locked up in silicates or oxide phases and would not be readily leached from the samples.

One possibility which must be considered is that the rocks analysed originally contained much more Au but have lost this Au at some stage in their history.

In the case of the G.M.D., there is no appreciable difference in Au content between amphibolitic and carbonated chloritic samples indicating that if any Au loss occurred, it was prior to the regional CO₂ metasomatism and lode formation and thus could not have been directly concentrated in the lode structures. Furthermore, since modern low-K tholeiites are also low in Au (Gottfried *et al.*, 1972, obtained an average of 0.6 ppb Au on 42 samples) and Archaean tholeiites from the Kakagi Lake Area, Ontario analysed by Kwong and Crocket (1978) have a mean of 1.97 and a median of 1.31 ppb Au, it is unlikely that the G.M.D. was ever rich in Au.

Unfortunately, there are no modern analogues of the komatiitic suite and the primary Au content of these Mg-rich rocks is thus conjectural. Two samples of Hannans Lake Serpentinite with low CO₂ content contained less than 1 ppb Au. The Devon Consols Basalt and basal Paringa Basalt samples contained similar low Au levels although the intrusive Williamstown Dolerite
contained slightly higher values. Anhaeusser et al. (1975) also found low Au concentrations (1-1.5 ppb) in komatiites and related rocks from Barberton, South Africa. It cannot, however, be certain that there was no loss of Au during serpentinization, metamorphism or other geological event especially in view of the fact that Ross and Keays (in prep.) have found that high Mg basalts at Kambalda, W.A., which are thought to be the equivalents of the Devon Consols Basalt (pers. comm. from G.A. Travis to R.R. Keays), contain an average of 5.5 ppb Au whilst thin meta-picrites, thick meta-picrites and sulphide-free cumulates thought to be equivalent to the Hannans Lake Serpentinite average 8.3, 3.95 and 8.77 ppb Au respectively. R.R. Keays (pers. comm.) considers that some Au has already been lost from these rocks and that 8 ppb is the minimum primary Au content of these ultramafics. If the Au levels measured by Ross and Keays reflect primary Au levels and these rocks are the equivalents of the Hannans Lake Serpentinite and Devon Consols Basalt, then at least 7 ppb Au has been lost from the Hannans Lake Serpentinite and 4.5 ppb from the Devon Consols Basalt in the Kalgoorlie region. Any Au mobilized during regional metamorphism and prior to formation of the lodes would have been dispersed and could not be considered as the source of the Au in the lodes. However, if Au was leached from the ultramafics and high Mg basalts at the time of deposition of the interflow sediments and concentrated in these sediments then the ultramafics could be considered as the ultimate source of the Au; the mechanisms by which leaching might occur will be discussed under the heading of the interflow sediments. If an average of 7 ppb Au was leached from the ultramafics, then 85 km$^3$ of these rocks would have been required to produce $2 \times 10^6$ kg Au; this is a large volume but is far more feasible than the 530 km$^3$ which would be required at the present Au content of these rocks. These figures can at best be regarded as tentative however, since the Au content of these rocks was probably
a function of their primary Cu-Ni-sulphide contents (R.R. Keays, pers. comm.) and these could be lower at Kalgoorlie than Kambalda.

Other Archaean Au deposits for which the enclosing greenstone pile has been postulated as the source of Au are much smaller than the Kalgoorlie deposits: the Yellowknife deposit has produced $1.2 \times 10^5$ kg Au (Boyle, 1961) and the Steynsdorp Goldfield only 67.6 kg Au (Viljoen et al., 1969) compared with a production of $1.11 \times 10^6$ kg and a postulated total original Au content of about $2 \times 10^6$ kg. Even in these much smaller deposits, Boyle and Viljoen et al., respectively have suggested that interflow sediments may have acted as concentrators of Au and contributed a significant amount of the Au in the veins.

(b) Interflow Sediments as Source Rocks

The Au content of the interflow sediments at Kalgoorlie (16-420 ppb, mean 90 ppb, median 32 ppb) is similar to that of interflow sediments from other Archaean terrains: 10-70 ppb at Yellowknife (Boyle, 1961); 75 ppb in the Komati tuffaceous shale, Steynsdorp Goldfield (Viljoen et al., 1969); 300-400 ppb at Cobalt, Ontario (Boyle, 1976); 69 ppb in a sample from the Kirkland Lake-Lardner Lake area, Canada (Tihor et al., 1977); and 0.02-1809 ppb, mean 146 ppb, median, 30.7 at Kambalda, W.A. (Bavinton and Keays, 1978). High concentrations of Ag, Te, As, Sb, S, base metals, CO$_2$ and K are also present in the interflow sediments at Kalgoorlie and similar values have been reported from the Yellowknife interflow sediments (Boyle, 1976). The high Te levels are particularly interesting in view of the characteristic presence of tellurides in the ores.

The constituents of the interflow sediments could be the product of fumarolic activity, propylitization connected with the original volcanic activity (Viljoen et al., 1969), sea water interaction with the freshly erupted volcanics (Keays and Scott, 1976), degassing of lava sheets (Boyle,
1976) or talc-carbonate alteration of the underlying ultramafics (Keays and Davison, 1976; Keays et al., 1978; Bavinton and Keays, 1978).

It could be argued that the Au in the sediments is not primary and that the structural or chemical properties of the sediments made them a suitable locus of deposition for hydrothermal Au. Travis and Woodall (1975) point out that because of the incompetent nature of the shales, bedding plane shears would readily develop in these during folding and such shears could conceivably channel mineralizing solutions derived from elsewhere. According to Tomich (1975) however, the Oroya Shoot is not directly controlled by bedding plane shears and is even offset by them. Another theory is that Au has been deposited in the vicinity of the sediments because of the reducing environment created by the carbonaceous material. This is the explanation given by Feldtmann and Farquharson (1913) for the Au-rich graphitic "indicator" in the North Collier mine at the northern end of the field. Honman (1916) also concluded that pyrite or graphite was responsible for precipitation of Au in equivalent interflow sediments at the Golden Ridge deposit. The fact that Au in the "green leader" deposits is associated with quartz-carbonate-mica bearing rocks, and not with the black shales themselves, however, tends to negate this possibility for the "green leader" ores. It is quite likely that there has simply been redistribution of Au in these sediments as suggested by Sawkins and Rye (1974) for the Homestake deposit.

Mass balance calculations indicate that at their present average Au content, the interflow sediments could not have provided more than about 10% of the total Au in the ores. However, in view of the small number of samples analysed and the high standard deviation in the Au content, it is possible that the interflow sediments were richer in places. No evidence was found of primary volcanogenic ore beds in the interflow sediments from
the southern part of the field. The "green leader" ore is strongly sheared and its original nature is uncertain; if, as suggested by Tomich (1959, 1974, 1976), this mineralization represents primary exhalative ore, then it must have been of restricted occurrence. It should be pointed out that the auriferous exhalites described by Ridler (1970) and Hutchinson et al. (1971) in the Canadian Archaean, are associated with felsic volcanics and not with the mafic and ultramafic lavas. However, even if the possibility of the former existence of primary ore-grade sediments is excluded, the interflow sediments could still account for all of the small Au deposits localized in these sediments at the northern end of the field, and the "green leader" ores also, if these contained only 8% of Kalgoorlie's total production as stated by Travis and Woodall (1975).

(c) The B.F.B. as Source Rocks

Many of the steeply dipping lodes cut into the Boulder Dyke, laterally or with depth and it is conceivable that the B.F.B. sediments infolded and faulted into this structure were an important source of Au. Although mass balance calculations based on the present Au content of samples from the southern end of the field suggest that the Boulder Dyke could only provide about 10% of the total Au in the Kalgoorlie ores, these calculations do not take into account the possibility of considerable volume reduction during folding and shearing on the Golden Mile Fault.

Black, pyritic shales currently contain more Au than any other lithological type analysed from the B.F.B. Where these shales are least deformed, they contain abundant fine-grained bedded pyrite which is apparently syn-genetic in origin and abundant carbonate and non-carbonate C (Chapter 2.2.8). As these shales were found both above and beneath sediments containing gypsum pseudomorphs, they may mark the position of former lagoons and salt pans, in which case there would have been plenty of opportunity for concentration of metals.
Felsic volcanism contemporaneous with the deposition of the evaporite-bearing sediments and black shales (as indicated by the interbedded tuffs, and the occurrence along strike, of coarse-grained volcanic breccias) may have been an important source of metals, S and other constituents in the sediments. Ridler (1970) and Hutchinson et al. (1971) have concluded that some Archaean Au deposits in Canada are auriferous carbonate and sulphide facies iron formations associated with felsic volcanic centres. Although this interpretation has been challenged by Tihor and Crocket (1977) who considered that the quartz-carbonate-rocks in the Lardner-Lake "Break" are merely altered ultramafics, the observation by Fripp (1976) of inclusions of gold in bedded sulphides in banded iron formation from Rhodesia and S isotope data obtained by Sawkins and Rye (1974) on the Homestake deposit strongly support the concept of syngenetic, possibly exhalative concentration of gold. The high Mg content of the Lardner Lake quartz-carbonate-rocks could be simply a reflection of a high original dolomite content if these were, in fact, sediments. At Kalgoorlie, the final explosive stages of volcanism of the G.M.D. may also have enriched the sedimentary environment in precious and volatile constituents. Concentration of metals provided by active volcanism and hot spring leaching of the underlying pile in shallow water basins or lagoons could conceivably produce sediments which were locally highly enriched in ore metals.

The presence of hydrocarbons and carbonaceous material in the black shales and possible algal laminates in the evaporite-bearing sediments suggests that primitive life was abundant at the time of sedimentary deposition and may have played an extremely important role in fixing Au and other metals. The role of primitive plants in extracting Au from the environment has been clearly demonstrated by Hallbauer and Van Warmelo (1974) and Hallbauer et al. (1977) for the carbonaceous basal reefs of the Witwatersrand
Basin, South Africa. The auriferous carbonaceous material in the Witwatersrand reefs occurs as thin seams of coal-like material; this material is so soft that it is not recovered in drill core (Pretorius, pers. comm.). If such material was ever present in the region of the South End Drilling, it could thus go undiscovered because of drill core loss. From this point of view, it is interesting to note that Montgomery (1905, p.90) stated that in the Queen Margaret mine at Bulong, "a bunch of about 10 lbs weight of black coal-like material was found, carrying gold freely through its mass, and with rich free gold and telluride of gold in close proximity to it. Analysis showed this to be carbon with 9.3 per cent of ash; specific gravity 1.76 after boiling." From the Kurnalpi sheet, it would appear that the Bulong sediments belong to an older volcanic cycle than the B.F.B. and hence cannot be correlated. However, the occurrence of natural gas in the Kalgoorlie mines (Montgomery, 1905; Simpson, 1912) suggests the presence of abundant carbonaceous material and led Montgomery to the conclusion that "the existence of carbonaceous matter in the country may have some connection with the genesis of gold in the lodes at Kalgoorlie."

Deformation probably played a significant role in mobilizing gold and sulphides into zones of weakness from which they could be readily leached.

The effect of progressive deformation on the pyritic black shales is illustrated in Fig. 118. Slight deformation results in a tendency for the bedded pyrite (Fig. 118A) to segregate and migrate into fractures and cleavage zones (Fig. 118B). In more intensely deformed samples, the sulphides are concentrated in recrystallized shear zones and fractures and pyrrhotite frequently replaces pyrite as the dominant sulphide (Fig. 118C). In extreme cases, as in the Golden Mile Fault zone in the Kalgoorlie Syncline, the black shales interbedded with more competent rocks have been reduced to pods of crumpled slate containing segregated graphite and sulphides (either
Fig. 118. Effect of progressive deformation on pyritic black shales.
A. Bedded pyrite in least deformed sample available (2691/4).
B. Slightly deformed black shale. Note segregation of pyrite in micro-fractures and shears (2648/4).
C. Concentration of pyrrhotite (white) within a recrystallized fracture zone in strongly deformed shale (2657/4).
pyrite or pyrrhotite with minor chalcopyrite and sphalerite). Similar deformational mobilization of sulphides in sediments has been noted by a number of authors including Marmo (1960), Boyle (1968, 1976) and Sawkins and Rye (1974). Large scale migration of sulphides and presumably Au into dilatant zones in the Homestake area, South Dakota has been demonstrated by the use of S isotope analysis (Sawkins and Rye, 1974). The fact that sulphide separates from highly sheared black shales from the Boulder Dyke contain high Au values (Table 66), clearly indicates that Au migrated with the sulphides although some Au may have been fixed by the C on graphitic shear planes since the graphite separates also have high Au contents relative to the whole rock (Table 66).

Sulphides and gold segregated into shear zones and dilational sites would be much more susceptible to dissolution by metamorphic fluids than those locked up in relatively undeformed shales. Evidence of leaching of sulphides in the Boulder Dyke was observed in a specimen from the Golden Mile Fault zone in the Mt. Charlotte Mine. The pyrite in this specimen, including some with well developed crystalline outlines, is very spongy in nature (Fig. 119). Since the fractures which traverse the pyrite contain recrystallized sulphides along them, it can be assumed that these were the channelways along which the solutions responsible for leaching the pyrite passed. Other reports of leaching of pyrite, except in the zone of oxidation, are not common; one example being described from the Pb-Zn ores of Central Peru by Lacy and Hosmer (1956); Glasson and Keays (1978) also report that pyrite has been dissolved from cleavage zones in geosynclinal sediments from Clunes, Vic. Ewers (1977) found that significant percentages of As, Sb and S were extracted from greywackes by hot distilled water at temperatures to be expected in natural geothermal systems (200-500°C) and suggested that Au, Ag and Te would also be leached under these conditions.
Table 66
Gold values in sheared sediments and mineral concentrates

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Description</th>
<th>Au (ppb)</th>
</tr>
</thead>
<tbody>
<tr>
<td>MC11/1</td>
<td>Sheared graphitic shale with pyrite segregations from the Golden Mile Fault zone, 11 level, Mt. Charlotte mine.</td>
<td>66</td>
</tr>
<tr>
<td>&quot;</td>
<td>Graphite from above sample.</td>
<td>320</td>
</tr>
<tr>
<td>&quot;</td>
<td>Pyrite from above sample.</td>
<td>990</td>
</tr>
<tr>
<td>MU5843</td>
<td>Graphite from black slate, Kalgoorlie Syncline, 22 level, Great Boulder Mine.</td>
<td>90</td>
</tr>
<tr>
<td>&quot;</td>
<td>Pyrite from above sample.</td>
<td>1,300</td>
</tr>
<tr>
<td>6468/13</td>
<td>Sheared pyrrhotite bearing shale from Kalgoorlie Syncline, S.E.13.</td>
<td>2.5</td>
</tr>
<tr>
<td>6462/13</td>
<td>Graphite from sheared shale, Kalgoorlie Syncline, S.E.13.</td>
<td>48</td>
</tr>
<tr>
<td>MU5857</td>
<td>Strongly sheared graphitic shale from interflow sediments, 9 level, Oroya-Brownhill mine.</td>
<td>190</td>
</tr>
<tr>
<td>&quot;</td>
<td>Graphite from above.</td>
<td>730</td>
</tr>
</tbody>
</table>
Fig. 119. Pyrite showing evidence of hydrothermal leaching. Note the crystalline outlines and spongy interior. A fracture which cuts across the spongy pyrite contains recrystallized sulphides within it.
Sulphate S has obviously been leached from the evaporite-bearing sediments as these contain abundant carbonate or silicate pseudomorphs after gypsum, and in some cases, anhydrite. However, the timing of release of this S is not certain; if it was released during mineralization, it could have provided up to 10% of the total S in the lodes and veins. Since the S in the lodes is generally light (with the exception of some sulphate veinlets), considerable fractionation of the presumably heavier sedimentary S would be necessary if this was a significant ore component.

CO₂ metasomatism is widespread in the Golden Mile (Bartram and McCall, 1971). Some of this CO₂ may have been liberated from carbonate-bearing sediments in the B.F.B. during deformation. According to Muffler and White (1969), the CO₂ in the thermal springs of the Salton Sea area is being formed by active metamorphism of the underlying sediments. These authors concluded that dolomite, ankerite, kaolinite and ferric ions react to produce chlorite, calcite and CO₂ at temperatures as low as 180°C and possibly as low as 125°C. At higher temperatures (290°C to 310°C), muscovite, calcite, quartz and ferric ions react to form epidote, K-feldspar and CO₂. Sheared sediments from the Golden Mile Fault zone on the No. 14 Level of the Mt. Charlotte mine show extensive development of chlorite and minor epidote suggesting that reactions similar to those described by Muffler and White have taken place in these sediments and it could be expected that more extensive reaction has occurred at depth with the resultant liberation of large quantities of CO₂. Silicification of carbonate-rich sediments, particularly those with pseudomorphs after gypsum has been observed in the Boulder Dyke in the mines and in deep drilling at the southern end of the field and this process would also lead to loss of CO₂.

Since the evaporite-bearing sediments are known to have lost S and CO₂, it is possible that they have lost other ore-forming constituents including
Au. However, unless sediments with unaltered sulphates are found, this will remain an unproved hypothesis.

The mass of black shales which originally overlay the G.M.D. in the mining area would probably have been rather impermeable to the flow of solutions. Gustafson and Miller (1937) even suggested that this impermeable cover may have restricted fluid flow from below and caused an increase in grade of ore near the surface. Carbonate-rich sediments and tuffs would have provided more ready access to fluids and may have contributed metals to the lodes if large scale fluid circulation took place.

Orebody associated with the basal shales of the B.F.B., particularly the "Duck Pond" shoot and the ores from the Idaho and Forrest King Mines may have developed as a direct result of syngenetic enrichment of the shales but it was not possible to test this theory.

The Role of Porphyries, Porphyrites and Granites

The association of some ore deposits with porphyries and porphyrites together with the presence of tourmaline in the ores led many authors (Larcombe, 1911; Feldtmann and Farquharson, 1913; Stillwell, 1929, and others) to the conclusion that the porphyries and porphyrites were the source of the Au.

At least some of the rocks originally regarded as porphyries are now known to be tuffs or sediments as they show bedding but even in those porphyries which do appear to be intrusive, the mineralization post-dates the porphyry. This indicates that the mineralizing solutions were not discharged directly from the cooling porphyries but does not rule out the possibility that the parent magma was responsible for mineralization as pointed out by O'Beirne (1968).

Although tourmaline is characteristically associated with ores of known granitic derivation, the presence of tourmaline at Kalgoorlie merely indicates
that the mineralizing fluids were rich in B; the interflow sediments and the black shales would be an alternative source of B.

Since the Pb isotope data obtained in this study suggests that the ore minerals (with Pb about 2700 m.y. old) have been mobilized and mixed with radiogenic Pb at about 2600 m.y., it is possible that the porphyries or granites played some part in the mobilizing the ore metals since these also appear to be about this age but at this stage, the isotopic data are too uncertain to draw any definite conclusions regarding timing of mineralization and intrusion.

Transport and Deposition of the Ore Constituents

Fluid inclusions are the only direct evidence of the composition of the ore-transporting fluids in ancient deposits. Unfortunately, fluid inclusions suitable for study are exceptionally rare in the fine-grained lodes of the Golden Mile and are only well developed in the gold-quartz ores. In the Mt. Charlotte ores examined, most of the fluid inclusions consisted simply of a liquid and a vapour phase although a few contained an additional CO₂ phase; there was no sign of any daughter minerals, indicating that the salinity of the ore-solutions was low.

Indirect evidence of the nature of the ore-forming fluids comes from a study of the wall rock alteration and the mineralogy and geochemistry of the ores.

CO₂ metasomatism is widespread and extends well beyond the limits of the K metasomatism associated with the lodes in the Golden Mile. Bartram (1969) and Bartram and McCall (1971) pointed out that there may have been a gradual change in the composition of the solutions with an early CO₂ rich solution becoming progressively richer in K with time, or alternatively, more than one phase of alteration. Another possibility is that CO₂ was present partially in the gaseous state and was more mobile than the other ore-forming constituents.
Difference in wall rock alteration and geochemistry of the ores between the Golden Mile and the northern and southern parts of the field suggest lateral changes in the composition of the mineralizing solutions or changes in the physical or chemical conditions of deposition. One possibility is that K-rich fluids were channelled up into the central Golden Mile area and then spread outwards, losing K by reaction with the country rock chlorite and leaching Ca and Na which were thus available for precipitation at the northern and southern parts of the field. This model would explain the higher S/Se ratios at the northern end of the field, S being more readily transported in solution than Se. It would not however, readily explain the low S/Se ratios at the southern end of the field. Nor would it explain the negative δ³⁴S values in the Golden Mile compared with a positive δ³⁴S value on a Mt. Charlotte sample; under constant chemical conditions, S will be fractionated with increase in the lighter isotope during solution transport.

Another possibility is that ore fluids did not migrate out laterally from the central part of the field but came up a series of shear zones and fractures along the entire length of the field. According to this model, there could be local variations in the composition of the solutions as a result of variations in the nature of the rocks through which the solutions passed. The differences between the S/Se ratios and δ³⁴S values in the Golden Mile lodes and those in the Mt. Charlotte, Golden Pike and Kalgoorlie Enterprise ores could be explained by assuming that the S and Se in the lodes were derived by leaching of volcanics or volcanogenic sulphides from the black shales or interflow sediments whilst the gold-quartz ores have a greater component of S from other sediments, for example, the evaporite-bearing sediments. However, changes in temperature, fO₂ and pH in the ore solutions can also result in changes in δ³⁴S values (Ohmoto, 1972) and Se/S ratios (Yamamoto, 1976). Such changes would explain the spatial variation
in the S/Se and $\delta^{34}$S values observed in the Golden Mile lodes although the variations could also be a reflection of differences in composition of successive phases of mineralization.

On the basis of the solubilities of siderite, calcite, magnesium hydroxide and magnetite, Bartram (1969) concluded that the ore fluids were neutral to weakly alkaline. Rytuba and Dickson (1975) found that there was no evidence of Au-chloride complexing even in 1 M NaCl solutions at temperatures up to $400^\circ$C in near neutral and reducing conditions and concluded that HS$^-$ was needed for complexing. In the presence of HS$^-$, the amount of complexing increases with the salinity. However, the lack of daughter minerals in the Mt. Charlotte fluid inclusions indicates that in this orebody at least, the salinity was low. Experimental work by Seward (1973) indicates that considerable quantities of Au may be carried as thio complexes in near neutron solutions at temperatures below $300^\circ$C. In view of the abundance of S in the Kalgoorlie ores, it is likely that thio complexes have played a significant role in Au transport. However, there are other possibilities. Boyle (1975) demonstrated that Na and K carbonate and bicarbonate solutions containing AsH$_3$ and SbH$_3$ are capable of dissolving Au and concluded that the soluble complexes were Au arsenides and stibnides such as [AuAs]$^{2-}$ and [AuSb]$^{2-}$ or more complicated polymerized and hydrolysed species. Boyle (1975) also found that the solubility of Au increased in the presence of H$_2$S and concluded that the complex could be a soluble sulphide of the type [AuS]$^{-}$ or thioarsenites, thioarsenates, thioantimonites, or thioantimononates of the type [Au(AsS$_3$)$_2$]$^{2-}$, [Au(SbS$_3$)$_2$]$^{2-}$. Seward (1973) suggested that in telluride deposits such as those of Kalgoorlie, telluro-Au complexes of the type Au(Te$_2$)$_2^-$, Au$_2$(Te$_2$)$_0$ and Au(Te$_2$)$_2^3$- may also have been important. Unfortunately, no experimental work has yet been carried out on these complexes.

As may have been carried as telluro-complexes (Seward, 1973) or as chloro complexes (Seward, 1976).
Diffusion may have played a role in the movement of some constituents. Boyle (1978) maintained that forced diffusion due to dilatancy of structures could result in the movement of large quantities of Ag and silica and Boyle (1963) suggested that diffusion was responsible for the migration of CO₂, H₂O, S, As and Sb into the gold-quartz veins of the Yellowknife district of Canada.

According to Seward (1973), deposition of Au carried as thio-complexes in solution would result from a decrease in temperature, a drop in pressure, a shift in pH to either acid or more alkaline conditions, increasing oxidation of the system (as would occur when hydrothermal solutions meet oxygenated meteoritic waters) or a decrease in the activity of reduced S (due to conversion of H₂S to H₂SO₄ or precipitation of sulphides). Boiling of ore solutions can also result in precipitation of ore-metals (Ewers and Keays, 1977).

No reliable mineralogical geothermometers have been found at Kalgoorlie. However, the tellurides do place some restraints on temperature range. The incongruent melting of sylvanite at approximately 354°C and krennerite at 382°C (Cabri, 1965), give the maximum temperature of formation of the tellurides whilst the inversion point of hessite at 145 ± 3°C (Kracek et al., 1966) places a minimum formation temperature on the steeply dipping lodes and the Hidden Secret ores. If the association of stuetzite (Ag₅₋ₓTe₃) with stoichiometric sylvanite (AuAgTe₄) in a specimen from the deep drilling of the Western Lode System is an equilibrium one, then temperatures could be lower than 220°C throughout the entire vertical range mined. The apparent restriction of montbrayite and antimonian montbrayite to the outer fringes of the field (the Doolette Lode, New North Boulder mine; the Oroya Shoot, Associated mine; the Kalgoorlie Enterprise Drilling; and the South End Drilling) may indicate that these areas formed at even lower temperatures than the central part of the field but little is yet known about the stability
of montbrayite. A single grain of hessite with apparently uniform anisotropism was found in a specimen of "green leader" ore from the Associated mine suggesting a formation temperature below $145 \pm 3^\circ C$ but further occurrences would be required to verify this.

There is no evidence of any variation in the telluride mineralogy with depth. The only evidence of vertical mineralogical zoning is in the tetrahedrite and gold compositions of the Western Lodes.

Tetrahedrite members change in composition from Sb-Ag-Zn rich members at depth to As-Fe rich members in the upper levels of the mines. However, the interpretation of zonation in tetrahedrite is not straightforward. Wu and Peterson (1977) found that the Sb and Ag contents of tetrahedrites increased away from the presumed centre of hydrothermal mineralization in the Casapalca area, Peru and indicated that a similar zonal pattern is present in other ore deposits. This pattern is exactly the opposite of that found at Kalgoorlie and it is obviously important to consider possible reasons for this difference. On the basis of fluid inclusion data, Wu and Peterson concluded that the temperature gradient throughout the Casapalca mine was small and thus that temperature was not significant in causing zonation. However, chemical controls may have been important. Wu and Peterson found a strong positive correlation between Ag and Sb in the tetrahedrite from Casapalca suggesting that an Sb-rich tetrahedrite can accommodate more Ag in the crystal structure than Sb-poor tetrahedrite. In addition, they concluded that the Ag content of tetrahedrite at a given Sb/As ratio is a function of the ratio of Ag to Cu in the hydrothermal solutions. It would appear that the overall zoning from As-rich, Ag-poor to Sb- and Ag-rich tetrahedrites at Casapalca is due to an overall increase in the ratios of Sb to As and Ag to Cu in the hydrothermal solutions as crystallization proceeded.

Analyses of the Kalgoorlie samples indicate that there is a decrease in
the overall Sb/As ratios of the rocks as well as the tetrahedrite in progressing from the deep to the upper levels. This suggests that the hydrothermal solutions became depleted in Sb relative to As as they moved upwards, probably as a consequence of the greater stability of the Sb-rich end member of the tetrahedrite series (even in the Casapalca mine where the vertical zoning is the reverse of that at Kalgoorlie, individual crusts and crystals show normal zoning with the Sb-rich member being the earliest crystallized according to Wu and Peterson, 1977).

Local variations in the composition of the tetrahedrite series may also be a reflection of the stability of coexisting minerals. Skinner et al. (1972) have pointed out that stable As minerals such as arsenopyrite, enargite and loellingite are common in nature whilst their Sb analogues are rare. Tellurantimony and bournonite were the only Sb-bearing minerals observed in contact with tetrahedrite at Kalgoorlie and these were of rare occurrence. As, however, is present in most of the pyrite at Kalgoorlie as well as in rare occurrences of arsenopyrite and hence the ratio of Sb/As in the tetrahedrites is greater than in the total rocks. The correlation between Ag and Sb found by Wu and Peterson (1977) in the Casapalca tetrahedrite is not as strong in the Kalgoorlie tetrahedrites: although the deep Sb-rich tetrahedrites have more Ag than the As-rich members from the upper levels, there are some Sb-rich tetrahedrites from intermediate levels that are low in Ag. This is possibly the result of preferential incorporation of Ag in the associated petzite.

The decrease in the Ag content of the native gold as the surface is approached in the Western Mines is probably related to a depletion in Ag of the hydrothermal solutions since there is a corresponding decrease in the Ag/Au ratio of the whole rocks but as in the case of the tetrahedrite series, the stability of coexisting minerals may also be important.
There is no evidence that temperature controlled the zonation of either the tetrahedrite or the gold; the temperature was obviously low enough for both minerals to crystallize at the deepest levels examined (present 1384 m).

The localization of most of the ore deposits in shear zones and fractures may indicate not only that these zones were excellent channelways for the migration of fluids, but also that movement on these structures resulted in a pressure drop and consequent precipitation of ore minerals. The commonly observed occurrence of shattered pyrite with fractures infilled with gold and tellurides is evidence that the mineralization was contemporaneous with deformation, and possibly the result of it.

If deformation led to boiling of the ore-fluids, then this would almost certainly have resulted in precipitation of ore-metals. Ewers and Keays (1977) concluded that loss of CO₂, H₂S and NH₃ during boiling of the Broadlands geothermal solutions, New Zealand, resulted in a reduction in the stability of the thio complexes and caused deposition of pyrite together with Au, As and Sb. The presence of CO₂ and variation in the fill of some of the fluid inclusions from the Mt. Charlotte mine suggests that boiling may have occurred at the northern end of the field but it was not possible to determine whether or not boiling had occurred in the Golden Mile.

Differences in the tectonic style of the Mt. Charlotte ore (tension gas veins) and the lodes (shear zones) may have resulted in large differences in the rate of pressure release in these two areas; if rapid pressure release resulted in boiling in the Mt. Charlotte ores but not in the lodes this could explain the differences in mineralogy of the two types of deposit.

Wall rock reaction has played a significant role in fixing ore constituents at Kalgoorlie. S in the ore fluids has reacted with Fe in the country rocks to form pyrite; according to Seward (1973), the reduced S activity would result in precipitation of Au. Although titano-magnetite is
generally replaced by pyrite in the ore zone, there are examples where the
titano-magnetite has been replaced or coated by native gold and tellurides
and in these cases, the ferrous Fe in the magnetite may have been responsible
for precipitation as suggested by Larcombe (1911).

Other factors which may have been important in ore deposition are mixing
of oxygenated and reduced waters, the presence in the ore fluids of methane
released from the carbonaceous shales during deformation and the reducing
environment created by the carbonaceous shales in the region of ore deposition.

5.4 Conclusions

The structural control of most of the Kalgoorlie orebodies, together
with the intense wall rock alteration, suggests that the ore-constituents
were introduced in solution. High concentrations of Te, Ag, As, Sb, Se, S,
B, Pb, K and CO₂ in addition to Au in some of the interflow sediments and
shales from the B.F.B. indicate that the country rocks were enriched in the
elements characteristically found in the ores; this, together with the fact
that Pb isotope ages of the ores are similar to those of the country rocks
(about 2700 m.y.) strongly suggests that the volcanic-sedimentary pile which
hosts the ore deposits was the source of the ore constituents.

At their present composition, the mafic and ultramafic rocks of the
Kalgoorlie succession are too low in Au to have provided a significant per-
centage of the Au in the lodes. If there was significant loss of Au and
other ore metals from the ultramafic and mafic igneous rocks at a very early
stage of their history, and these metals were concentrated syngenetically in
the interflow sediments or B.F.B. sediments, then the mafics and ultramafics
could be indirectly the source of the Au in the lodes.

Interflow sediments associated with the mafic members of the succession
are rich in Au but are volumetrically too small to account for all of the
Au in the Kalgoorlie ores. They may however have been the source of the Au
in the flatly dipping "green leader" ores and also in the small workings
associated with these sediments in the northern part of the field.

Most of the steeply dipping lodes at Kalgoorlie are clustered about
the Boulder Dyke, a faulted, synclinal infold of B.F.B. sediments. If the
Au content of these sediments had originally been high, they would be ideal
source rocks in view of their structural setting. On the basis of the pre-
sent average Au content of the sediments and tuffs analysed from the
essentially unmineralized southern end of the field, the B.F.B. sediments
infolded into the Boulder Dyke could have provided only a small percentage
of the Au in the lodes. There is however, some possibility that Au has
already been lost from these sediments; S has certainly been lost from the
evaporite-bearing sediments. In view of the variable lithologies, it is
also probable that some sediments were locally highly enriched in Au.

As a tentative model, it is postulated that the interaction of active
felsic volcanism with the shallow-water to subareal environment of the B.F.B.
in which organic-rich black shales and evaporite-bearing sediments were
accumulating, provided ideal conditions for substantial localized concen-
tration of metals. One possibility is that Au was adsorbed by primitive plants
and was preserved in thin beds of soft carbonaceous material. Such material
would deform very readily, and possibly together with evaporite-bearing sedi-
ments, could account for the tightness of the Kalgoorlie Syncline.

Folding and faulting probably played a vital role in segregating sul-
phides and gold into sites from which they could be readily leached.

Heat generation by underlying granitic or porphyry magmas may have
played a part in the mobilization of ore constituents but there is no evi-
dence for this; the nearest outcropping granite is 15 miles away and the
porphyries themselves predate mineralization.

In view of the inferred near neutral to slightly alkaline nature of the
mineralizing fluids and the abundance of sulphides, thio complexes were probably more important than chloro complexes in the transport of Au in the mineralizing fluids. However, telluro Au complexes, thioarsenites, thioarsenates, thioantimonites and thioantimononates may also have been significant.

Fracturing of the rocks and successive movement along lode shears resulting in a drop in pressure and possibly boiling of the ore solutions may have been instrumental in causing precipitation of sulphides, gold and other ore minerals. Other conditions which may have led to precipitation of ore metals are changes in Eh, pH and activity of S species during wall rock alteration, mixing of thermal fluids with meteoritic waters and the production of methane during shearing of the carbonaceous sediments.

There is no evidence that the minerals crystallized under a vertical temperature gradient. Vertical zoning in members of the tetrahedrite series and native gold can be explained simply by depletion of the mineralizing fluids in Sb and Ag as they moved upward.

Variations in mineralogy and geochemistry between deposits is probably related to fractionation of elements during solution transport, local variation in the source rocks and differences in the tectonic style (shear zone or fracture) which in turn would control pressure release and rate of deposition. The "green leader", Kalgoorlie Enterprise and South End mineralization may have been deposited at even lower temperatures than the ore in the Golden Mile in view of the fact that these ores contain montbrayite.

Overall, however, there is a gradation in mineralogy and geochemistry between the various types of deposits suggesting that they are all genetically related with the metals being derived ultimately from the volcanic-sedimentary pile which now hosts the ores.

The complicated inter-lode relationships and common occurrence of cross-cutting veins of tellurides indicate that mobilization of ore constituents
extended over a considerable period of time. Movement on oblique faults and continuing movement on the Golden Mile Fault probably played an important role in remobilizing the gold and tellurides.
Appendix 1
LOCATION AND DESCRIPTION OF SAMPLES ANALYSED

A. Country Rocks

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>530/9</td>
<td>D.D.H. S.E.9 - 530' (162 m)</td>
<td>Carbonated spinifex-textured Hannans Lake Serpentine.</td>
</tr>
<tr>
<td>542/9</td>
<td>&quot; - 542' (165 m)</td>
<td>Cumulate-textured Hannans Lake Serpentine.</td>
</tr>
<tr>
<td>1683/9</td>
<td>&quot; - 1683' (513 m)</td>
<td>Cumulate-textured Hannans Lake Serpentine with rare relict pyroxenes.</td>
</tr>
<tr>
<td>1687/9</td>
<td>&quot; - 1687' (514 m)</td>
<td>Weakly talc-carbonated version of above sample.</td>
</tr>
<tr>
<td>2143/9</td>
<td>&quot; - 2143' (653 m)</td>
<td>Highly altered, talc-carbonated Hannans Lake Serpentine.</td>
</tr>
<tr>
<td>2153/9</td>
<td>&quot; - 2153' (656 m)</td>
<td>Weakly talc-carbonated cumulate-textured Hannans Lake Serpentine. Pentlandite and pyrrhotite containing exsolved niccolite and gersdorffite replace serpentine pseudomorphs after olivine.</td>
</tr>
<tr>
<td>1054/10</td>
<td>D.D.H. S.E.10 - 1054' (321 m) (Fig. 12)</td>
<td>Serpentinized Devon Consols Basalt.</td>
</tr>
<tr>
<td>240/6</td>
<td>D.D.H. S.E.6 - 240' (73 m)</td>
<td>Ultramafic Williamstown Dolerite altered to an actinolite-chlorite-talc assemblage.</td>
</tr>
<tr>
<td>839/6</td>
<td>&quot; - 839' (256 m)</td>
<td>Mafic Williamstown Dolerite; amphiboles are slightly altered to chlorite.</td>
</tr>
<tr>
<td>825/6</td>
<td>&quot; - 825' (251 m)</td>
<td>Mafic Williamstown Dolerite; amphiboles are more altered than in above sample.</td>
</tr>
<tr>
<td>1813/10</td>
<td>D.D.H. S.E.10 - 1813' (553 m) (Fig. 12)</td>
<td>Spotted Paringa Basalt - altered to a serpentine-chlorite-carbonate assemblage.</td>
</tr>
<tr>
<td>4193/13A</td>
<td>D.D.H. S.E.13 - 4193' (1278 m) (Fig. 14)</td>
<td>Paringa Basalt - pillow interior.</td>
</tr>
<tr>
<td>4193/13B</td>
<td>&quot;</td>
<td>Paringa Basalt - interpillow material - consisting of chlorite, carbonate, quartz, pyrite and ilmenite.</td>
</tr>
<tr>
<td>4724/13</td>
<td>D.D.H. S.E.13 - 4724' (1440 m)</td>
<td>Unit 2 G.M.D. - amphiboles partially replaced by chlorite, actinolite and biotite; saussuritized feldspar.</td>
</tr>
<tr>
<td>Sample No.</td>
<td>Location</td>
<td>Description</td>
</tr>
<tr>
<td>------------</td>
<td>----------</td>
<td>-------------</td>
</tr>
<tr>
<td>4764/13</td>
<td>D.D.H. S.E.13 - 4764' (1452 m)</td>
<td>Unit 3 G.M.D. - chloritic, carbonated, skeletal leucoxene.</td>
</tr>
<tr>
<td>5008/13</td>
<td>&quot; &quot; - 5008' (1548 m)</td>
<td>Unit 4 G.M.D. - amphiboles partially replaced by chlorite; saussuritized feldspar.</td>
</tr>
<tr>
<td>5093/13</td>
<td>&quot; &quot; - 5093' (1552 m)</td>
<td>Unit 5 or 6 G.M.D. - chloritic and carbonated.</td>
</tr>
<tr>
<td>5248/13</td>
<td>&quot; &quot; - 5248' (1599 m)</td>
<td>Unit 7 G.M.D. - amphiboles partially replaced by chlorite; saussuritized feldspar; skel. titanomagnetite, ilmenite and euhehedral magnetite.</td>
</tr>
<tr>
<td>5314/a13</td>
<td>D.D.H. S.E.13 - 5314' (1619 m)</td>
<td>Unit 7 G.M.D. - fine-grained; amphiboles partially replaced by chlorite; abundant zoisite.</td>
</tr>
<tr>
<td>5314/b13</td>
<td>&quot; &quot; &quot; &quot;</td>
<td>Coarse-grained patch in above sample.</td>
</tr>
<tr>
<td>5450/13</td>
<td>&quot; &quot; - 5450' (1661 m)</td>
<td>Unit 8 G.M.D. - chloritic, biotite-rich, carbonated and sheared.</td>
</tr>
<tr>
<td>5546/13</td>
<td>&quot; &quot; - 5546' (1691 m)</td>
<td>Unit 8 or 9 G.M.D. - chloritic, quartz-rich rock with coarse-grained skeletal LEU Coxene; epidote-bearing sheared but contains some remnant granophyric quartz.</td>
</tr>
<tr>
<td>5805/13</td>
<td>&quot; &quot; - 5805' (1769 m)</td>
<td>Unit 9 G.M.D. - chloritic with saussuritized feldspar; carbonated, sheared.</td>
</tr>
<tr>
<td>6185/13</td>
<td>&quot; &quot; - 6185' (1885 m)</td>
<td>Unit 9 G.M.D. - chloritic with saussuritized feldspar; carbonated, sheared.</td>
</tr>
<tr>
<td>6314/13</td>
<td>&quot; &quot; - 6314' (1924 m)</td>
<td>Unit 9 G.M.D. - chloritic with saussuritized feldspar.</td>
</tr>
<tr>
<td>6362/113</td>
<td>&quot; &quot; - 6362' (1939 m)</td>
<td>Unit 9 G.M.D. - chloritic with saussuritized feldspar; carbonated.</td>
</tr>
<tr>
<td>6443/13</td>
<td>&quot; &quot; - 6443' (1964 m)</td>
<td>Unit 10 G.M.D. - chloritic, carbonated, sheared.</td>
</tr>
<tr>
<td>6623/13</td>
<td>&quot; &quot; - 6623' (2016 m)</td>
<td>Unit 9 G.M.D. - chloritic with minor zoisite and epidote; carbonated.</td>
</tr>
<tr>
<td>6731/13</td>
<td>&quot; &quot; - 6731' (2051 m)</td>
<td>Unit 9 G.M.D. - amphiboles partially replaced by chlorite; saussuritized feldspar; minor epidote.</td>
</tr>
<tr>
<td>7136/13</td>
<td>&quot; &quot; - 7136' (2175 m)</td>
<td>Unit 9 G.M.D. - amphiboles partially replaced by chlorite; saussuritized feldspar; epidote.</td>
</tr>
<tr>
<td>7264/13</td>
<td>&quot; &quot; - 7264' (2214 m)</td>
<td>Unit 8 G.M.D. - chloritic; abundant albite laths and granophyric quartz; carbonated.</td>
</tr>
</tbody>
</table>
### Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>7332/13</td>
<td>D.D.H. S.E.13 - 7332' (2235 m)</td>
<td>Unit 8 G.M.D. - chloritic, biotite-bearing; carbonated and sheared.</td>
</tr>
<tr>
<td>7736/13</td>
<td>&quot; &quot; - 7736' (2358 m)</td>
<td>Unit 7 G.M.D. - amphibolitic; slight chloritic alteration.</td>
</tr>
<tr>
<td>7836/13</td>
<td>&quot; &quot; - 7836' (2389 m)</td>
<td>Chilled margin? between units 6 and 7 G.M.D. - chloritic, carbonated; contains some blue-green amphiboles.</td>
</tr>
<tr>
<td>7883/13</td>
<td>&quot; &quot; - 7883' (2403 m)</td>
<td>Unit 6 G.M.D. - amphibolitic; saussuritized feldspar.</td>
</tr>
<tr>
<td>8005/13</td>
<td>&quot; &quot; - 8005' (2440 m)</td>
<td>Unit 5 G.M.D. - chloritic, carbonated.</td>
</tr>
<tr>
<td>8048/13</td>
<td>&quot; &quot; - 8048' (2453 m)</td>
<td>Unit 4 G.M.D. - chloritic; feldspars are mainly saussuritized but some have recrystallized; carbonated.</td>
</tr>
<tr>
<td>8328/13</td>
<td>&quot; &quot; - 8328' (2538 m)</td>
<td>Unit 4 G.M.D. - amphiboles partially replaced by chlorite and biotite; contains zoisite.</td>
</tr>
<tr>
<td>8456/13</td>
<td>&quot; &quot; - 8456' (2578 m)</td>
<td>Unit 3 G.M.D. - amphiboles partially replaced by chlorite; albite partially replaced by carbonate; contains zoisite and occasional garnets.</td>
</tr>
<tr>
<td>8597/13</td>
<td>&quot; &quot; - 8597' (2620 m)</td>
<td>Unit 2 G.M.D. or mafic intrusive? - consists of biotite, clino-zoisite, chlorite, quartz, ilmenite and a little carbonate.</td>
</tr>
<tr>
<td>8671/13</td>
<td>&quot; &quot; - 8671' (2643 m)</td>
<td>Unit 1 G.M.D. - altered to an actinolite, chlorite, carbonate assemblage.</td>
</tr>
<tr>
<td>4489/13</td>
<td>D.D.H. S.E.13 - 4489' (1368 m)</td>
<td>Interflow sediment in Paringa Basalt: a black shale containing pyrrhotite in lenses parallel to foliation; also contains abundant tourmaline.</td>
</tr>
<tr>
<td>8/93</td>
<td>No. 9 Level, Judd Shaft (L.V. &amp; S. co-ords. 2720 N, 575 E)</td>
<td>Interflow sediment in Paringa Basalt: black shale with fine-grained bedded pyrite and some cross-cutting sulphides.</td>
</tr>
<tr>
<td>2787/10</td>
<td>D.D.H. S.E.10 - 2787' (849 m) (Fig. 12)</td>
<td>Interflow sediment in Paringa Basalt; consists of bands of quartz, carbonate and chlorite; bands of pyrrhotite containing minor chalcopyrite, sphalerite and pyrite parallel the bedding.</td>
</tr>
<tr>
<td>MC14/8</td>
<td>No. 14 Level, Mt. Charlotte mine (Fig. 2)</td>
<td>Banded, pyrite rich cherty sediment in Paringa Basalt.</td>
</tr>
</tbody>
</table>
Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>1250/7</td>
<td>D.D.H. S.E.7 - 1250' (381 m) (Fig. 17)</td>
<td>Interflow sediment between Paringa Basalt and G.M.D. - contains irregular lenses of pyrrhotite and melnikovite pyrite aligned roughly parallel to bedding; some pyrrhotite, sphalerite and chalcopyrite also occur in quartz-carbonate veinlets.</td>
</tr>
<tr>
<td>KS1</td>
<td>Small shaft to the east of Mt. Hunt (Fig. 2)</td>
<td>Kapai slate - black shale with fine-grained bedded pyrite; slightly weathered.</td>
</tr>
<tr>
<td>2648/4</td>
<td>D.D.H. S.E.4 - 2648' (807 m) (Fig. 14)</td>
<td>Black shale with bedded pyrite, slightly deformed.</td>
</tr>
<tr>
<td>2691/4</td>
<td>&quot; - 2691' (820 m)</td>
<td>Black shale with bedded pyrite and some sulphide bearing nodules.</td>
</tr>
<tr>
<td>2873/4</td>
<td>&quot; - 2873' (876 m)</td>
<td>Black shale with pyrite nodules.</td>
</tr>
<tr>
<td>3080/4</td>
<td>&quot; - 3079'6&quot; to 3087' (938.6 to 940.9 m)</td>
<td>Black shale with minor bedded pyrite and some carbonate pseudo-concretions.</td>
</tr>
<tr>
<td>3087/4</td>
<td>&quot; to 3095' (940.9 m to 943.4 m)</td>
<td>Black shale with carbonate-rich patches; pyritic in places.</td>
</tr>
<tr>
<td>3761/4</td>
<td>&quot; - 3761' (1146 m)</td>
<td>Laminated black shale with coarse-grained euhehedral pyrite.</td>
</tr>
<tr>
<td>6246/4</td>
<td>&quot; - 6246' (1903 m)</td>
<td>Black shale containing bands of pyrrhotite and arsenopyrite.</td>
</tr>
<tr>
<td>2657/4</td>
<td>&quot; - 2657' (810 m)</td>
<td>Pyritic, carbonate rich sediment interbedded with black shale.</td>
</tr>
<tr>
<td>4465/4</td>
<td>&quot; - 4465' (1361 m)</td>
<td>Siliceous, carbonate-bearing shale; contains some coarse-grained, euhehedral carbonate rhombs partially replaced by sulphides.</td>
</tr>
<tr>
<td>3095/4</td>
<td>&quot; - 3095' to 3101' (943.4 to 944.9 m)</td>
<td>Black shale with carbonate-rich patches.</td>
</tr>
<tr>
<td>3029/4</td>
<td>&quot; - 3029' to 3037'6&quot; (923.2 to 925.8 m)</td>
<td>Siliceous black shale with carbonate pseudo-concretions.</td>
</tr>
<tr>
<td>3038/4</td>
<td>&quot; - 3037'6&quot; to 3057' (925.8 to 931.8 m)</td>
<td>Finely laminated shale with carbonate rich and carbonaceous laminae.</td>
</tr>
<tr>
<td>3057/4</td>
<td>&quot; - 3057' to 3079'6&quot; (931.8 to 938.6 m)</td>
<td>Laminated shale.</td>
</tr>
</tbody>
</table>
Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>6241/4</td>
<td>D.D.H. S.E.4 - 6241' (1902 m)</td>
<td>Laminated shale.</td>
</tr>
<tr>
<td>14/6C</td>
<td>Boulder Dyke, No. 6 Level of Chaffers Mine, 65 to 85 m from M.P. (47210 N, 19160 E)</td>
<td>Bulk sample of black shales and mud-chip conglomerates.</td>
</tr>
<tr>
<td>MC14/7B</td>
<td>Decline in Mt. Charlotte mine - just below 1450' level at edge of Boulder Dyke</td>
<td>Fuchsite-bearing sediment.</td>
</tr>
<tr>
<td>2573/4</td>
<td>D.D.H. S.E.4 - 2573' (784 m)</td>
<td>Mud-chip conglomerate.</td>
</tr>
<tr>
<td>2849/4D</td>
<td>&quot; &quot; - 2849' (868 m)</td>
<td>Laminated sediment containing abundant carbonate pseudomorphs after gypsum.</td>
</tr>
<tr>
<td>2849/4L</td>
<td>&quot; &quot; &quot;</td>
<td>Light coloured material adjoining laminated material above.</td>
</tr>
<tr>
<td>2832/4</td>
<td>&quot; &quot; - 2832' (863 m)</td>
<td>Massive sediment with pseudomorphs after sulphates.</td>
</tr>
<tr>
<td>2955/4</td>
<td>&quot; &quot; - 2955' (900 m)</td>
<td>Lithic albite tuff.</td>
</tr>
<tr>
<td>2970/4</td>
<td>&quot; &quot; - 2970' (905 m)</td>
<td>Albite sercite tuff.</td>
</tr>
<tr>
<td>6263/4</td>
<td>&quot; &quot; - 6263' (1909 m)</td>
<td>Sericite tuff.</td>
</tr>
<tr>
<td>6316/4</td>
<td>&quot; &quot; - 6316' (1925 m)</td>
<td>Light-coloured, porphyritic rock fragment from volcanic breccia.</td>
</tr>
<tr>
<td>4327/4</td>
<td>&quot; &quot; - 4327' (1319 m)</td>
<td>Hornblende-bearing rock fragment from volcanic breccia.</td>
</tr>
<tr>
<td>1296/7</td>
<td>D.D.H. S.E.7 - 1296' (395 m)</td>
<td>Light coloured sheared rock with rare quartz phenocrysts.</td>
</tr>
<tr>
<td>5479/13</td>
<td>D.D.H. S.E.13 - 5479' (1670 m)</td>
<td>Light coloured albite-rich rock.</td>
</tr>
</tbody>
</table>

B. Mineralized Samples

Western Lodes (Fig. 3)

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>324/5022</td>
<td>D.D.H. 5022 - 324 to 325' (98.8 to 99.1 m)</td>
<td>Strongly sheared, pyritic bleached G.M.D.</td>
</tr>
<tr>
<td>327/5022</td>
<td>&quot; &quot; - 327 to 328' (99.7 to 100.0 m)</td>
<td>Siliceous core of lode - contains some segregated pyrite.</td>
</tr>
<tr>
<td>329/5022</td>
<td>&quot; &quot; - 329 to 330' (100.3 to 100.6 m)</td>
<td>Strongly sheared, pyritic bleached G.M.D.</td>
</tr>
</tbody>
</table>
Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>334/5022</td>
<td>D.D.H. 5022 - 334 to 335' (101.8 to 102.1 m)</td>
<td>Strongly sheared, weakly bleached, pyritic G.M.D.</td>
</tr>
<tr>
<td>335/5022</td>
<td>&quot; &quot; - 335 to 335'6&quot; (102.1 to 102.3 m)</td>
<td>Moderately sheared and bleached, weakly mineralized.</td>
</tr>
<tr>
<td>338/5022</td>
<td>&quot; &quot; - 337'6&quot; to 338'3&quot; (102.9 to 103.1 m)</td>
<td>&quot; &quot; &quot; &quot; &quot; &quot;</td>
</tr>
<tr>
<td>282/5027</td>
<td>D.D.H. 5027 - 282' (86.0 m)</td>
<td>Chloritic, weakly sheared unit 9 G.M.D.</td>
</tr>
<tr>
<td>310/5027</td>
<td>&quot; &quot; - 310' (94.5 m)</td>
<td>Chloritic, epidote-bearing, sheared unit 9 G.M.D.</td>
</tr>
<tr>
<td>318/5027</td>
<td>&quot; &quot; - 318' (96.9 m)</td>
<td>Chloritic, moderately sheared unit 9 G.M.D.</td>
</tr>
<tr>
<td>320/5027</td>
<td>&quot; &quot; - 319' to 320'3&quot; (97.2 to 97.6 m)</td>
<td>Sheared, slightly bleached unit 9 G.M.D.</td>
</tr>
<tr>
<td>321/5027</td>
<td>&quot; &quot; - 321' to 321'6&quot; (97.8 to 98.0 m)</td>
<td>Strongly sheared, pyritic bleached G.M.D.</td>
</tr>
<tr>
<td>323/5027</td>
<td>&quot; &quot; - 323' to 324' (98.5 to 98.8 m)</td>
<td>Slightly bleached unit 9 G.M.D. with long curved amphiboles.</td>
</tr>
<tr>
<td>331/5027</td>
<td>&quot; &quot; - 331' to 332' (100.9 to 101.2 m)</td>
<td>Sheared, moderately bleached G.M.D.</td>
</tr>
<tr>
<td>336/5027</td>
<td>&quot; &quot; - 336' to 337' (102.4 to 102.7 m)</td>
<td>Sheared, bleached G.M.D. with aggregates of pyrite along shear planes.</td>
</tr>
<tr>
<td>338/5027</td>
<td>&quot; &quot; - 338' to 339' (103.0 to 103.3 m)</td>
<td>Sheared, bleached G.M.D. with disseminated fine-grained pyrite.</td>
</tr>
<tr>
<td>342/5027</td>
<td>&quot; &quot; - 341'6&quot; to 342' (104.0 to 104.2 m)</td>
<td>Siliceous rock containing lenses of carbonate and quartz.</td>
</tr>
<tr>
<td>343/5027</td>
<td>&quot; &quot; - 343' to 344' (104.5 to 104.9 m)</td>
<td>Siliceous rock with fine-grained disseminated pyrite.</td>
</tr>
<tr>
<td>346/5027</td>
<td>&quot; &quot; - 345'6&quot; to 346' (105.3 to 105.5)</td>
<td>Siliceous core of lode.</td>
</tr>
<tr>
<td>347/5027</td>
<td>&quot; &quot; - 346'3&quot; to 347' (105.6 to 105.8 m)</td>
<td>Siliceous rock with fine-grained disseminated pyrite.</td>
</tr>
<tr>
<td>Sample No.</td>
<td>Location</td>
<td>Description</td>
</tr>
<tr>
<td>-----------</td>
<td>----------</td>
<td>-------------</td>
</tr>
<tr>
<td>348/5027</td>
<td>D.D.H. 5027 - 348' to 349' (106.1 to 106.4 m)</td>
<td>Strongly sheared, bleached pyritic G.M.D. with abundant pyrite aggregates.</td>
</tr>
<tr>
<td>352/5027</td>
<td>&quot;       &quot; - 352' to 353' (107.3 to 107.6 m)</td>
<td>Sheared, moderately bleached unit 9, G.M.D.</td>
</tr>
<tr>
<td>722/5028</td>
<td>D.D.H. 5028 - 722' to 723' (220.1 to 220.4 m)</td>
<td>Strongly sheared, bleached G.M.D. with abundant very fine-grained pyrite.</td>
</tr>
<tr>
<td>727/5028</td>
<td>&quot;     &quot; - 727' 728' (221.6 to 221.9 m)</td>
<td>Strongly sheared, bleached G.M.D. with fine-grained disseminated pyrite.</td>
</tr>
<tr>
<td>9H/4</td>
<td>No. 9 Level, Horseshoe mine</td>
<td>Sheared, bleached G.M.D. with fine-grained disseminated pyrite.</td>
</tr>
<tr>
<td>9H/7</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Sheared, bleached G.M.D. with fine-grained disseminated pyrite.</td>
</tr>
<tr>
<td>9H/10</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Weakly bleached, sheared unit 9 G.M.D.</td>
</tr>
<tr>
<td>9H/11</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Sheared, bleached G.M.D. with fine-grained disseminated pyrite.</td>
</tr>
<tr>
<td>9H/12</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Sheared, bleached G.M.D. with fine-grained, disseminated pyrite.</td>
</tr>
<tr>
<td>9H/13</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Mineralized vein in chloritic G.M.D. near No. 4 Lode.</td>
</tr>
<tr>
<td>9H/14</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Chloritic unit 9, G.M.D.</td>
</tr>
<tr>
<td>9H/15</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Weakly mineralized chloritic G.M.D. containing abundant recrystallized magnetite.</td>
</tr>
<tr>
<td>9H/16</td>
<td>&quot;       &quot; &quot;       &quot;</td>
<td>Sheared, bleached G.M.D., weakly pyritic.</td>
</tr>
<tr>
<td>20C/1</td>
<td>Internal drive above No. 20 Level, Chaffers mine 2366 S, 60 W.</td>
<td>Sheared, bleached G.M.D. with fine-grained pyrite, some of which shows signs of deformation.</td>
</tr>
<tr>
<td>20C/3</td>
<td>As above</td>
<td>Sheared, bleached G.M.D. with fine-grained pyrite.</td>
</tr>
<tr>
<td>20C/4</td>
<td>As above</td>
<td>Sheared, bleached G.M.D. with fine-grained pyrite.</td>
</tr>
<tr>
<td>MU3415A</td>
<td>2770' level Chaffers Shaft</td>
<td>Fine-grained siliceous rock.</td>
</tr>
</tbody>
</table>
Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>MU3415B</td>
<td>2770' level Chaffers Shaft</td>
<td>Less siliceous than above, contains patches of fine-grained pyrite.</td>
</tr>
<tr>
<td>MU5864</td>
<td>C Lode, Over 700' level, Perseverance mine.</td>
<td>Bleached, sheared G.M.D. containing pyrite and specks of visible tellurides.</td>
</tr>
<tr>
<td>WM17082</td>
<td>No. 10 Level, Perseverance Mine, Lake View Lode</td>
<td>Bleached, pyritic G.M.D.</td>
</tr>
<tr>
<td>WM3413</td>
<td>No. 10 Level, Perseverance Mine, Lake View Lode - bulk sample.</td>
<td>Bleached, pyritic G.M.D.</td>
</tr>
<tr>
<td>AA40434</td>
<td>No. 12 Level, Associated Mine, stope at south end of Orroya Shoot 148225 N, 19273.7 E</td>
<td>Light grey, sheared rock (probably Paringa Basalt) with only slight green colouration along cleavage planes and in irregular silicate/carbonate patches. Contains disseminated fine-grained pyrite.</td>
</tr>
<tr>
<td>AA40435</td>
<td>West wall of above stope at 48297 N, 19272.1 E.</td>
<td>Green mica moderately abundant: occurs along cleavage planes and in quartz-carbonate veins. Sulphides and microscopic tellurides tend to be concentrated about the veins.</td>
</tr>
<tr>
<td>AA40436</td>
<td>No. 10 Level, Associated Mine at 48485 N, 19706 E.</td>
<td>Sample consists in part of grey siliceous lode with coarse-grained carbonate patches and in part of very micaceous sections which vary in colour from grey to light greenish grey.</td>
</tr>
<tr>
<td>AA40437</td>
<td>No. 12 Level Associated Mine, Emerald Lode. From walls of drive at 48420 N, 19610 E.</td>
<td>Patches of bright green mica present in some pieces; other pieces consist almost entirely of siliceous lode with pale green colouration.</td>
</tr>
<tr>
<td>AA40438</td>
<td>No. 10 Level, Associated Mine at 48487 N, 19680 E.</td>
<td>Quartz-carbonate rich patches in altered Paringa Basalt? Samples have only a weak greenish colouration. Finer grained pyrite is moderately abundant in some pieces. Tellurides are associated with the quartz-carbonate patches.</td>
</tr>
</tbody>
</table>
### Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>WM3027</td>
<td>No. 8 Level, New North Boulder Mine, from fairly strong shear on hanging wall of Ley Lode</td>
<td>Sheared rock with slight green colouration. Contains abundant fine-grained tellurides.</td>
</tr>
<tr>
<td>MU5854</td>
<td>No. 9 Level, Oroya Brownhill Mine</td>
<td>Brecciated lode containing abundant fine-grained pyrite some of which has been sheared.</td>
</tr>
<tr>
<td></td>
<td>Mt. Charlotte Ore-body (Fig. 120)</td>
<td></td>
</tr>
<tr>
<td>1/7</td>
<td>D.D.H. MC243 - 44' (13.4 m)</td>
<td>Mineralized unit 9 G.M.D.</td>
</tr>
<tr>
<td>3/6</td>
<td>&quot; &quot; - 71' (21.6 m)</td>
<td>Weakly mineralized unit 9 G.M.D.</td>
</tr>
<tr>
<td>3/17</td>
<td>&quot; &quot; - 85' (25.9 m)</td>
<td>Bleached unit 8 G.M.D. with coarse-grained euhedral pyrite.</td>
</tr>
<tr>
<td>3/27</td>
<td>&quot; &quot; - 103' (31.4 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>3/37</td>
<td>&quot; &quot; - 124' (37.8 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. adjacent to quartz vein.</td>
</tr>
<tr>
<td>3/38</td>
<td>&quot; &quot; - 125' (38.1 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. adjacent to quartz vein.</td>
</tr>
<tr>
<td>3/41</td>
<td>&quot; &quot; - 130' (39.6 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>3/45</td>
<td>&quot; &quot; - 136' (41.5 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>3/47</td>
<td>&quot; &quot; - 138' (42.1 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>3/51</td>
<td>&quot; &quot; - 147' (44.8 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>16/7</td>
<td>&quot; &quot; - 444' (135 m)</td>
<td></td>
</tr>
<tr>
<td>86/MC91</td>
<td>D.D.H. MC91 - 86' (26.2 m)</td>
<td>Bleached unit 8 G.M.D. with coarse-grained euhedral pyrite slightly oxidised.</td>
</tr>
<tr>
<td>146/402</td>
<td>D.D.H. MC402 - 146 m</td>
<td>Bleached pyritic unit 8 G.M.D. traversed by quartz veins.</td>
</tr>
<tr>
<td>HVA</td>
<td>No. 12 Level, Mt. Charlotte Mine</td>
<td>Flatly dipping quartz vein containing pyrite and pyrrhotite.</td>
</tr>
<tr>
<td>HVB</td>
<td>&quot; &quot; &quot;</td>
<td>Bleached, pyritic envelope directly adjacent to above vein (0 to 2 cm)</td>
</tr>
</tbody>
</table>
Fig. 120. Position of D.D.Hs MC243, MC402 and MCS2A in the Mt. Charlotte ore-body.
### Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>HVC</td>
<td>No. 12 Level, Mt. Charlotte Mine</td>
<td>Remainder of bleached pyritic envelope associated with above vein (2 to 3.7 cm). Chlorotic unit 7 adjacent to bleached zone, contains some pyrrhotite (6.8 cm). Chlorotic unit 9 (20 cm).</td>
</tr>
<tr>
<td>HVD</td>
<td>&quot; &quot;</td>
<td></td>
</tr>
<tr>
<td>HVE</td>
<td>&quot; &quot;</td>
<td></td>
</tr>
<tr>
<td>L13A</td>
<td>No. 13 Level, Mt. Charlotte Mine</td>
<td>Vertical quartz vein containing pyrite and pyrrhotite.</td>
</tr>
<tr>
<td>L13B</td>
<td>&quot; &quot;</td>
<td>Pyritic halo directly adjacent to above vein (0 to 1 cm).</td>
</tr>
<tr>
<td>L13C</td>
<td>&quot; &quot;</td>
<td>Remained of pyritic halo (1-4 cm).</td>
</tr>
<tr>
<td>L13D</td>
<td>&quot; &quot;</td>
<td>Pyrrhotite bearing chloritic unit 8 G.M.D. adjacent to bleached zone (5 cm). Chloritic unit 8 G.M.D. (25 cm). Chloritic unit 8 G.M.D. (56 cm). Chloritic unit 8 G.M.D. (89 cm)</td>
</tr>
<tr>
<td>L13E</td>
<td>&quot; &quot;</td>
<td></td>
</tr>
<tr>
<td>L13F</td>
<td>&quot; &quot;</td>
<td></td>
</tr>
<tr>
<td>L13G</td>
<td>&quot; &quot;</td>
<td></td>
</tr>
<tr>
<td>23/2A</td>
<td>D.D.H. MCS2A - 2034' (618 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>24/2A</td>
<td>&quot; &quot; - 2037' (619 m)</td>
<td>Weakly bleached unit 8 G.M.D.</td>
</tr>
<tr>
<td>38/2A</td>
<td>&quot; &quot; - 2080' (634 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>41/2A</td>
<td>&quot; &quot; - 2090' (637 m)</td>
<td>Bleached, pyritic unit 8 G.M.D.</td>
</tr>
<tr>
<td>45/2A</td>
<td>&quot; &quot; - 2104' (641 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. with quartz veins.</td>
</tr>
<tr>
<td>46/2A</td>
<td>&quot; &quot; - 2107' (642 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. with quartz veins.</td>
</tr>
<tr>
<td>73/2A</td>
<td>&quot; &quot; - 2221' (677 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. with quartz veins.</td>
</tr>
<tr>
<td>74/2A</td>
<td>&quot; &quot; - 2224' (678 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. with quartz veins.</td>
</tr>
<tr>
<td>77/2A</td>
<td>&quot; &quot; - 2229' (680 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. with quartz veins.</td>
</tr>
<tr>
<td>100/2A</td>
<td>&quot; &quot; - 2289' (698 m)</td>
<td>Bleached, pyritic unit 8 G.M.D. with quartz veins.</td>
</tr>
<tr>
<td>107/2A</td>
<td>&quot; &quot; - 2319' (975 m)</td>
<td>Bleached pyritic unit 9 G.M.D. with quartz veins.</td>
</tr>
</tbody>
</table>
### Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>5.05/GP8</td>
<td>Golden Pike Ore-body</td>
<td>Bleached unit 8 G.M.D. cut by quartz veins, contains abundant pyrite.</td>
</tr>
<tr>
<td>6.00/GP8</td>
<td>&quot;</td>
<td>Bleached unit 8 G.M.D. cut by quartz veins, contains abundant pyrite.</td>
</tr>
<tr>
<td></td>
<td>Kalgoorlie Enterprise Drilling (Fig. 3)</td>
<td></td>
</tr>
<tr>
<td>84/KE55</td>
<td>D.H. K.E.55 - 84' (25.6 m)</td>
<td>Quartz vein with bleached, pyritic halo in chloritic fine-grained G.M.D. (unit 7 or 8).</td>
</tr>
<tr>
<td>103/KE53</td>
<td>D.H. K.E.53 - 103' (31.4 m)</td>
<td>Sheared, chloritic unit 9 traversed by quartz leaders; contains abundant pyrite.</td>
</tr>
<tr>
<td>121/KE53</td>
<td>&quot;</td>
<td>Sheared, moderately bleached unit 8 traversed by quartz leaders; pyritic.</td>
</tr>
<tr>
<td></td>
<td>(36.6 m)</td>
<td></td>
</tr>
<tr>
<td>127/KE53</td>
<td>&quot;</td>
<td>Sheared pyritic, moderately bleached unit 8.</td>
</tr>
<tr>
<td></td>
<td>(38.7 m)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Hannans North Lode</td>
<td></td>
</tr>
<tr>
<td>WM3416</td>
<td>124' Level Star Shaft</td>
<td>Weakly pyritic quartz vein dipping 5°E.</td>
</tr>
<tr>
<td>WM3417</td>
<td>&quot; &quot; &quot; &quot;</td>
<td>Bulk sample of Hannans North Lode.</td>
</tr>
<tr>
<td>WM23205</td>
<td>&quot; &quot; &quot; &quot;</td>
<td>Massive siliceous lode.</td>
</tr>
<tr>
<td></td>
<td>South End Drilling</td>
<td></td>
</tr>
<tr>
<td>2550/15</td>
<td>D.H. S.E.15 - 2550' (777 m) (Fig. 63)</td>
<td>Sheared, bleached G.M.D. with disseminated pyrite.</td>
</tr>
<tr>
<td>3984.1/15</td>
<td>&quot; &quot;</td>
<td>Sheared, chloritic unit 8 G.M.D.</td>
</tr>
<tr>
<td></td>
<td>3984'1&quot;-2&quot; (1214 m)</td>
<td></td>
</tr>
<tr>
<td>3984.2/15</td>
<td>&quot; &quot;</td>
<td>Pyritic, telluride-bearing halo adjacent to quartz vein.</td>
</tr>
<tr>
<td></td>
<td>3984'2&quot;-2½&quot;</td>
<td>Halo on other side of vein.</td>
</tr>
<tr>
<td>3984.4/15</td>
<td>&quot; &quot;</td>
<td></td>
</tr>
<tr>
<td></td>
<td>3984'3½&quot;-4&quot;</td>
<td></td>
</tr>
<tr>
<td>3984.5/15</td>
<td>&quot; &quot;</td>
<td>Weakly bleached, sheared G.M.D. adjacent to above sample.</td>
</tr>
<tr>
<td></td>
<td>3984'4&quot;-6&quot;</td>
<td></td>
</tr>
<tr>
<td>3984.7/15</td>
<td>&quot; &quot;</td>
<td>Sheared, chloritic unit 8 G.M.D.</td>
</tr>
</tbody>
</table>
### Appendix 1 contd.

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Location</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>5410/13</td>
<td>D.D.H. S.E.13 - 5410'</td>
<td>Quartz vein with gold-bearing coarse-grained pyrite.</td>
</tr>
<tr>
<td></td>
<td>(1649 m) (Fig. 14)</td>
<td></td>
</tr>
<tr>
<td>4537/4</td>
<td>D.D.H. S.E.4 - 4537'</td>
<td>Mineralized top of G.M.D; contains some quartz veins.</td>
</tr>
<tr>
<td></td>
<td>(1383 m) (Fig. 14)</td>
<td></td>
</tr>
<tr>
<td>3104/4</td>
<td>&quot;</td>
<td>Flow-top breccia of G.M.D; bleached; contains some pyrrhotite.</td>
</tr>
<tr>
<td></td>
<td>3101'</td>
<td></td>
</tr>
<tr>
<td></td>
<td>(945 m)</td>
<td></td>
</tr>
<tr>
<td>3672/4</td>
<td>&quot;</td>
<td>Pyrite-bearing quartz-vein in tuffaceous sediment.</td>
</tr>
<tr>
<td></td>
<td>3672'</td>
<td></td>
</tr>
<tr>
<td></td>
<td>(1119 m)</td>
<td></td>
</tr>
</tbody>
</table>
Appendix 2

DIAMOND DRILL CORE LOGS

Abbreviations used:

G.M.D. - Golden Mile Dolerite
B.F.B. - Black Flag Beds
P.B. - Paringa Basalt
fg - Fine-grained (0 - 0.25 mm) - size refers to oxides only
f-mg - Fine-to-medium-grained (0.25-1.0 mm)
mg - Medium-grained (1.0-3.0 mm)
m-cg - Medium-to-coarse-grained (3.0-5.0 mm)
cg - Coarse-grained (>5.0 mm)
chl. - Chloritic
amph. - Amphibolitic
ep. - Epidote-bearing
Py - Pyrite
Po - Pyrrhotite
lx - Leucoxene
skel. - Skeletal (with reference to oxides)
il - Ilmenite
mte - Magnetite
mod. - Moderately
dissem. - Disseminated
oph. - Ophitic

All core has been logged in feet. Conversions to metres are added in brackets.

A. South End Drilling

D.D.H. S.E.4
1265-1269' (385.6-386.8 m) - Chl. G.M.D. with mg to m-cg skel. lx; oph. texture not developed.
1269-1274' (386.8-388.3 m) - Typical chl. unit 9 with m-cg skel. lx and well developed oph. texture.
1274-1530' (388.3-466.3 m) - Chl. G.M.D. with m-cg skel. lx, oph. texture not present. Weakly mineralized at 1285-1286' (391.7-392.0 m) and 1287-1287'7" (392.3-392.5 m).
1530-1532' (466.3-467.0 m) - Chl. Unit 9 with oph. texture gradational to G.M.D. without oph. texture.
1532-1547' (467.0-471.5 m) - Chl. G.M.D. with mg to m-cg skel. lx, oph. texture not prominent.
1547-1548' (471.5-471.8 m) - Mineralized G.M.D.
1548-1550' (471.8-472.4 m) - Sheared G.M.D.
1550-1552' (472.4-473.1 m) - Chl. G.M.D. with long branching pseudomorphs after amphiboles and feldspars.
1552-1563' (473.1-476.4 m) - Chl. unit 9 with m-cg skel. il and 1x.
1563-1572' (476.4-479.2 m) - Ep. unit 9 with mg skel. il.
1572-1608' (479.2-490.1 m) - Chl. unit 9 with mg skel. lx; Oph. texture.
1608-1608.6' (490.1-490.3 m) - Mineralized quartz vein.
1608.6'-1709' (490.3-520.9 m) - Chl. unit 9 with mg skel. lx gradually
becoming finer grained.
1709-1760' (520.9-536.5 m) - Weakly sheared, chl. unit 9 with f-mg skel.
lx (shearing at ~40° to side of core).
1760-1765' (536.5-538.0 m) - Weakly mineralized unit 9.
1765-1791' (538.0-545.9 m) - Chl. unit 9 with f-mg skel. lx.
1791-1794' (545.9-546.8 m) - Chl. unit 9 G.M.D. in juxtaposition with black
material resembling a black shale (a polished section revealed that this was
a segregation of fine-grained oxides).
1794-1795'.6' (546.8-547.3 m) - Fine-grained pinkish-coloured rock.
1795'.6'-1812' (547.3-552.3 m) - Alternation of chilled G.M.D. and unit 9
G.M.D. with f-mg skel. lx. Most boundaries are marked by thin shears.
1812-1906' (552.3-581.0 m) - Chl. unit 9 with f-mg to fg skel. lx and
occasionally il, gradually becoming finer down hole.
1906-1918' (581.0-584.6 m) - Fg. G.M.D.
1918-1920' (584.6-585.2 m) - Brecciated G.M.D; fragments of unit 9 G.M.D
in a calcite matrix.
1920-1936' (585.2-590.1 m) - Strongly sheared unit 9 traversed by a 3" wide
quartz vein.
1936-1939' (590.1-591.0 m) - Unit 9 G.M.D. with fg skel. il.
1939-1955' (591.0-595.9 m) - A rock which on the weathered surface of the
drill core has a fragmental appearance with fragments of G.M.D. with fg to
mg skel. lx in an iron-stained matrix. On sawn surface, the matrix is light
grey - appears to be a volcanic breccia.
1955-2001' (595.9-609.9 m) - Predominantly G.M.D. with f-mg skel. lx; frag-
mentary in places.
2001-2049' (609.9-624.5 m) - G.M.D. with f-mg lx grading to fg lx. Weakly
sheared at ~40° to core.
2049-2065' (624.5-629.4 m) - Chilled G.M.D. with variolitic texture, con-
taining fragments of G.M.D. with fg to mg skel. lx. Probably a flow top
breccia.
2065-2129' (629.4-648.9 m) - Pinkish coloured "porphyry" with chlorite
defining a weak foliation; contains occasional inclusions of what appear to
be G.M.D. - may be a tuff. Contact with G.M.D. at 2065' (629.4 m) is sheared.
2129-2130' (648.9-649.2 m) - Contact of porphyry with chilled G.M.D.
2130-2192' (649.2-668.1 m) - Alternation of fg and f-mg G.M.D; boundaries are always sharp, sometimes marked by a small shear but in other cases there is no sign of shearing - probably a coarse-grained volcanic breccia.
2192-2256' (668.1-687.6 m) - Moderately sheared G.M.D. with f-mg 1x.
2256-2290' (687.6-698.0 m) - Strongly sheared G.M.D; f-mg 1x where recognizable; weakly mineralized.
2290-2488' (698.0-758.3 m) - Patches of G.M.D. with fg 1x in G.M.D. with predominantly f-mg 1x; boundaries are sharp.
2488-2498.6" (758.3-761.5 m) - Fragments of G.M.D. of varying grainsize (fg to mg) in a tuffaceous matrix.
2498.6"-2511' (761.5-765.4 m) - Fragments of G.M.D. in a matrix of black shale and reworked tuffaceous material.
2511-2546' (765.4-776.0 m) - Laminated black shales and siltstones; strongly contorted in first few inches (30 cm). Slately cleavage is developed in places but in other sections the sediments are virtually undeformed and primary features, e.g. lenticular bedding, are preserved.
2546-2548' (776.0-776.6 m) - Black shale containing fragments of G.M.D.
2548-2556' (776.6-779.1 m) - Fragments of G.M.D. with f-mg to fg 1x in a tuffaceous matrix. Some of the fragments have been partially replaced by sulphides.
2556-2564' (779.1-781.5 m) - Alternation of light coloured sediments and black shale.
2564-2569' (781.5-783.0 m) - Fragments of G.M.D. in matrix of shale and tuff (core is missing at contacts with B.F.B. on either side; possibly out of order).
2569-2571' (783.0-783.6 m) - Black shale with large irregular fragments of light grey sediment.
2571-2590' (783.6-784.0 m) - Light grey sediment containing abundant angular fragments of shale (mud-chip conglomerate). The shale fragments become smaller and more orientated (at ~80° to side of core) towards the end of the section. In places, black shale beds have been brecciated and cemented by carbonate.
2590-2594' (789.4-790.7 m) - Grey medium-grained sediment with occasional black shale fragments.
2594-2616' (790.7-797.4 m) - Well bedded shale ranging in colour from light grey to black; contains abundant carbonate rhombs.
2616-2641' (797.4-805.0 m) - Massive grey sediment containing abundant carbonate rhombs. At 2624' (799.8 m) there is a coarse-grained patch containing abundant fine-grained shale fragments. This passes up into more carbonate-bearing sediments with occasional inclusions of black shale.

2641-2646' (805.0-806.5 m) - Well laminated grey shale with highly disrupted bedding.

2646-2649' (806.5-807.4 m) - Pyritic black shale.

2649-2651' (807.4-808.0 m) - Alternation of black shale and light grey sediment.

2651-2664' (808.0-812.0 m) - Black shale with white veinlets running through it; fairly highly disturbed but some features could be due to slumping; contains pyrite and carbonate nodules which are probably diagenetic as well as fg Py. There are some interbedded carbonate-rich sediments.

2664-2670' (812.0-813.8 m) - Black shale with some fg carbonate; minor Py.

2670-2686' (813.8-818.7 m) - Grey, carbonate-rich sediment; pyritic in places.

2686-2692' (818.7-820.5 m) - Black, pyritic shale.

2692-2737' (820.5-834.2 m) - Grey and greenish grey carbonate-rich sediments; some have elongate pseudomorphs after sulphates. These are interrupted by a brecciated black shale at 2694'6"-2695'4" (821.3-821.5 m).

2737-2738' (834.2-834.5 m) - Brecciated black shale.

2738-2848' (834.5-868.1 m) - Massive grey sediment containing abundant carbonate pseudomorphs after sulphates.

2848-2850' (868.1-868.7 m) - Laminated sediment; highly disrupted - laminations swing from almost perpendicular to the core sides to parallel to them. Carbonate pseudomorphs after sulphates visible in the darker coloured laminae.

2850-2874' (868.7-876.0 m) - Massive grey sediment containing abundant carbonate pseudomorphs after sulphates. At 2874' (876.0 m) there is an erosional boundary to a pyritic black shale.

2874-2877' (876.0-876.9 m) - Pyritic black shale containing fragments of the underlying sediment and abundant Py nodules.

2877-2908' (876.9-886.4 m) - Black, pyritic shale (some core is missing).

2908-2910' (886.4-887.0 m) - Massive grey sediment interbedded with black shale containing fragments of massive grey sediment.

2910-2919' (887.0-889.7 m) - Pale grey sericite-rich rock; rather fragmentary.
2919-2938' (889.7-895.5 m) - Fairly coarse-grained tuff containing euhedral albite crystals; cut by barren quartz veins.
2938-2939'6" (895.5-896.0 m) - Brecciated tuff.
2939'6" - 2940' (896.0-896.1 m) - Sericite rich rock with "cherty" appearance.
2940-2944' (896.1-897.3 m) - Alternation of black shale and "cherty" looking rock. Well laminated in places (~25° to side of core).
2944-2948' (897.3-898.8 m) - Fairly coarse-grained tuff containing euhedral albite crystals.
2948-2950' (898.6-899.2 m) - Sericitic tuff? with vug filled with coarsely crystalline calcite.
2950-2961' (899.2-902.5 m) - Fairly coarse-grained tuff containing euhedral albite crystals.
2961-2974' (902.5-906.5 m) - Pale green sericite-rich tuff?
2974-2976' (906.5-907.1 m) - Barren white quartz vein.
2976-2981' (907.1-908.6 m) - Well laminated sequence of black shale and "cherty" material (bedding ~70° to side of core).
2981-2982'6" (908.6-909.1 m) - Grey "cherty" material.
2982'6"-2988' (909.1-910.7 m) - Laminated light and dark grey shale containing moderately abundant carbonate.
2988-3009'10" (910.7-917.4 m) - Black shale with mottly appearance (due to segregations of graphite?).
3009'10"-3017' (917.4-919.6 m) - Light grey sediment.
3017-3020' (919.6-920.5 m) - Well laminated shales (bedding ~70° to side of core).
3020-3037'9" (920.5-925.9 m) - Siliceous black shale containing abundant oval-shaped carbonate "concretions" (2-5 mm long).
3037'9"-3057' (925.9-931.8 m) - Finely laminated shales with primary features, e.g. slumps well preserved.
3057-3079'6" (931.8-938.6 m) - Shale with light and dark grey laminations; not as finely laminated as above section.
3079'6"-3081' (938.6-939.1 m) - Black shale with carbonate "concretions".
3081-3087' (939.1-940.9 m) - Black, pyritic shale.
3087-3089' (940.9-941.5 m) - Black shale with light coloured carbonate-rich patches.
3089-3093' (941.5-942.8 m) - Black pyritic shale.
3093-3102' (942.8-945.5 m) - Black shale with coarser grained patches of carbonate-rich material - Fe-stained on surface of core. Some of the black shales contain abundant Po which has segregated along lines of weakness. Some lighter coloured sediments contain angular fragments of black shale (mud chip conglomerate).

3102-3107'6" (945.5-947.2 m) - Alternation of brecciated fg G.M.D. and black shales. Some of the G.M.D. is chilled and there are blebs of replaced glassy material - obviously a flow top breccia.

3107'6"-3128' (947.2-953.4 m) - Rather altered unit 9 with f-mg lx.

3128-3142' (953.4-957.7 m) - Very altered G.M.D.?

3142-3155' (957.7-961.6 m) - Altered unit 9 with mg skel. lx.

3155-3164' (961.6-964.4 m) - Chl. unit 9 with mg skel. il.

3164-3183' (964.4-970.2 m) - Chl. unit 9 with f-mg skel. il.

3183-3183'3" (970.2-970.3 m) - Mineralized G.M.D. fragment in replacement type quartz vein.

3183'3"-3198' (970.3-974.8 m) - Altered G.M.D.?

3198-3313' (974.8-1001.0 m) - Chl. unit 9 with mg skel. lx.

3313-3333' (1001.0-1015.9 m) - Altered G.M.D.?

3333-3364' (1015.9-1025.3 m) - Chl. unit 9 with generally mg skel. lx but some patches with m-cg skel. lx and prominent oph. texture; grades to fg towards end of section.

3364-3371' (1025.3-1027.5 m) - Altered unit 9?

3371-3386' (1027.5-1032.1 m) - G.M.D.? with abundant iron-bearing carbonate; lx not visible.

3386-3390' (1032.1-1033.3 m) - Variolitic G.M.D. with fg skel. il.

3390-3393' (1033.3-1034.2 m) - Altered G.M.D.? with quartz-carbonate veining.

3393-3419' (1034.2-1042.1 m) - Fine-grained altered G.M.D.?

3419-3420' (1042.1-1042.4 m) - Quartz vein.

3420-3491' (1042.4-1064.1 m) - Dark fine-grained rock with round blebs of carbonate 1-2 mm across; weakly banded in places; probably a mafic tuff.

3491-3499' (1064.1-1066.5 m) - Siliceous black shale; contains some Py in cross-cutting fractures.

3499-3590' (1066.5-1094.2 m) - Alternation of dark rock with carbonate blebs and a siliceous, carbonate-bearing black shale.

3590-3594' (1094.2-1095.5 m) - Black, carbonate bearing shale.

3594-3604' (1095.5-1098.5 m) - Dark rock with carbonate blebs.

3604-3656' (1098.5-1114.3 m) - Black shale; first section is carbonate-rich, later part is more pyritic and contains some large pyrite nodules (up to 3 cm across).
3656-3674' (1114.3-1119.8 m) - Greyish-green, tuffaceous sediment?; later 5' contains Py and is cut by a quartz vein.
3674-3680' (1119.8-1121.7 m) - Fe-stained slate.
3680-3687' (1121.7-1123.8 m) - Grey carbonate-bearing tuffaceous sediment?
3687-3689' (1123.8-1124.4 m) - Quartz vein.
3689-3695' (1124.4-1126.2 m) - Greyish green, tuffaceous sediment? containing some Py; quartz vein at end of section.
3695-3725' (1126.2-1135.4 m) - Black slate (cleavage ~70° to sides of core); contains Py nodules up to 3 cm across.
3725-3925' (1135.4-1196.3 m) - Dark grey to black shales and slates with some lighter coloured laminations (bedding ~70° to sides of core, roughly parallel to cleavage).
3925-3935' (1196.3-1199.4 m) - Greyish-green tuffaceous sediments containing some shale fragments.
3935-4250' (1199.4-1295.4 m) - Light to medium grey laminated shales or tuffaceous sediments; the bedding is generally disturbed.
4250-4258' (1295.4-1297.8 m) - Dark green tuffaceous sediment.
4258-4264' (1297.8-1299.7 m) - Black shale.
4264-4273' (1299.7-1302.4 m) - Greyish-green tuffaceous sediments.
4273-4302' (1302.4-1311.3 m) - Poorly sorted tuffaceous sediment with fine-grained to coarse-grained beds. Some of the coarse-grained beds contain accretionary lapilli. The grain-size of the lithic fragments increases down hole and the rock grades into a lapilli tuff.
4302-4453' (1311.3-1357.3 m) - Volcanic breccia with abundant light grey, hornblende-bearing rock fragments up to 20 cm across.
4453-4530' (1357.3-1380.7 m) - Grey carbonate-bearing shale; silicified; well laminated in places (~20° to side of core).
4530-4547' (1380.7-1385.9 m) - Quartz veins in mineralized G.M.D. with f-mg skel. lx (core is badly broken; relationship to sediments uncertain).
4547-4585' (1385.9-1397.5 m) - G.M.D. with mg lx.
4585-4595' (1397.5-1400.5 m) - Breccia consisting predominantly of G.M.D. fragments and some porphyry like fragments.
4595-4611' (1400.5-1405.4 m) - Grey tuff or sediment with inclusions of shale.
4611-5114' (1405.4-1558.7 m) - Bleached G.M.D. with f-mg to mg lx becoming coarser down hole. Generally has a disrupted appearance; sheared.
5114-5135' (1558.7-1565.1 m) - Unit 9 with oph. texture reasonably well developed; mg skel. lx.
5135-5160' (1565.1-1572.8 m) - Unit 9 with f-mg lx; slightly sheared.
5160-5284' (1572.8-1610.6 m) - Unit 9 with fairly well developed oph. texture; f-mg skel. lx.
5284-5422' (1610.6-1652.6 m) - Unit 9 with f-mg lx; cut by mineralized shear zones and quartz veins (some tourmaline-bearing).
5422-5464' (1652.6-1665.4 m) - Chl. unit 9 with variable lx size but generally mg, some f-mg.
5464-5503' (1665.4-1697.8 m) - Chl. unit 9, f-mg lx becoming finer down hole; contains some Py.
5503-5612' (1697.8-1710.5 m) - Chl. unit 9 with lx increasing in size from fg to mg.
5612-5617' (1710.5-1712.1 m) - Mineralized quartz vein.
5617-5729' (1712.1-1745.2 m) - Chl. unit 9 with f-mg lx decreasing in size down hole.
5729-5746' (1745.2-1751.4 m) - Disrupted G.M.D. with f-mg to fg lx.
5746-5750' (1751.4-1752.6 m) - Waxy quartz vein containing cg Py.
5750-5753'6" (1752.6-1753.7 m) - Sheared, mineralized G.M.D. with fg to f-mg lx. Sheared contact to sericitic rock.
5753'6"-5763' (1753.7-1756.6 m) - Sericitic rock with streaky banding; light green with iron stained patches and blackish streaks.
5763-5792'6" (1756.6-1765.6 m) - Silicified black sediment with abundant iron-stained streaks.
5792'6"-5842' (1765.6-1780.6 m) - Alternation of medium-grained tuff or sediment containing shale fragments and black shale. At 5842' (1765.6 m), there is a sharp contact with a silicified volcanic breccia.
5842-5855' (1780.6-1784.6 m) - Silicified volcanic breccia - fine-grained felsic volcanic fragments in a siliceous matrix; interbedded siliceous black shale.
5855-5879' (1784.6-1791.9 m) - Volcanic breccia containing fragments of light coloured volcanics and shale in a mg to cg groundmass which is light grey in colour except where iron-stained on the surface of the core. Rock fragments are up to 10 cm across but generally less than 2 cm.
5879-5890' (1791.9-1795.3 m) - Light pinkish coloured volcanic breccias and lithic tuffs.
5890-5897' (1795.3-1797.4 m) - Grey m-cg tuffaceous sediments with occasional shale fragments.
5897-6028' (1797.4-1837.3 m) - Alternation of coarse- to medium-grained tuffaceous sediments and grey to black shales - well developed graded bedding in parts, fining down hole.
6028-6064' (1837.3-1848.3 m) - Highly deformed shales with some less deformed m-cg tuffs.
6064-6106' (1848.3-1861.1 m) - Reasonably well laminated light to medium grey sediments.
6106-6157' (1861.1-1875.7 m) - Volcanic breccia with rock fragments averaging ~10 cm across but up to 30 cm across in a mg groundmass - the rock fragments vary in colour from light grey or brown to cream; the whole rock has a bleached appearance but some fragments, even of the same lithology, are more highly altered than others.
6157-6161' (1875.7-1877.9 m) - Graded tuffaceous sediment - fining down hole.
6161-6171' (1877.9-1880.9 m) - Volcanic breccia or agglomerate; closely packed rock fragments averaging ~1 cm but up to 10 cm across; many different rock types in a grey groundmass; probably reworked.
6171-6198' (1880.9-1889.2 m) - Volcanic breccia similar to above but with a pale greenish bleached appearance; contains inclusions of a bright green mica. At 6198' (1889.2 m), there is a sharp boundary to a black shale.
6198-6199'6" (1889.2-1889.6 m) - Black shale with iron stained laminations.
6199'6"-6203' (1889.6-1890.7 m) - Medium grey tuff or sediment with iron stained laminations (carbonate-rich).
6203-6214' (1890.7-1894.0 m) - Fine-grained volcanic breccia or lapilli tuff; rock fragments generally smaller than ½ cm.
6214-6237' (1894.0-1901.0 m) - Very light coloured volcanic breccia. A bright green mica is present along minor shears.
6237-6238' (1901.0-1901.3 m) - Laminated shale - light carbonate rich bands alternating with dark grey bands.
6238-6241' (1901.3-1902.3 m) - Light grey tuff containing rare fg shale inclusions.
6241-6241'6" (1902.3-1902.4 m) - Shale band.
6241'6"-6243' (1902.4-1902.9 m) - Light grey sediment or tuff.
6243-6251' (1902.9-1905.3 m) - Alternation of shales and light grey sediments.
6251-6262' (1905.3-1908.7 m) - Light grey massive rock which starts off fg and becomes coarser-grained down hole; contains abundant albite and rare fg inclusions of shale - could be a felsic flow or a massive tuff. Last part of section is brecciated.
6262-6291' (1908.7-1917.5 m) - Banded sericite-rich rock.
6291-6333' (1917.5-1930.3 m) - Volcanic breccia consisting dominantly of very porphyritic rock fragments (feldspar phenocrysts).
6333' (1930.3 m) - End of hole.


0-4410' (0-1344.1 m) - Not logged.
4410-4489' (1344.1-1368.3 m) - Pillowed P.B; chloritic at commencement of section becoming gradually more bleached towards end of section.
4489-4496' (1368.3-1370.4 m) - Laminated shale.
4496-4497' (1370.4-1370.7 m) - Fg P.B.
4497-4498' (1370.7-1371.0 m) - Shale.
4498-4516' (1371.0-1376.5 m) - Fg P.B.
4516-4518' (1376.5-1377.1 m) - Black shale.
4518-? (1377.1-? m) - Fg P.B. - core at contact of P.B. and G.M.D. has been lost due to wedging ~4630-4751' (1411.2-1448.1 m) - Unit 2 G.M.D. with prominent amphiboles and variable 1x size; not logged in detail. The core is sheared and broken in the vicinity of the contact between units 2 and 3 and the relationship between these zones is not certain.
4751-4799' (1448.1-1462.7 m) - Altered unit 3 G.M.D.? with mg skel. 1x and ill. Fairly sharp boundary to less altered rock at 4799' (1462.7 m).
4799-4837' (1462.7-1474.3 m) - Amph. unit 3 with mg skel. 1x; platey amphiboles very prominent; gradually becomes more altered down hole.
4837-4881' (1474.3-1487.7 m) - Chl. unit 3?; rock has a dull green, homogeneous appearance; amphiboles not visible.
4881-4885' (1487.7-1489.0 m) - Transitional zone to m-cg G.M.D. with prominent amphibole plates (unit 3?).
4885-4997' (1489.0-1523.1 m) - M-cg G.M.D. with prominent amphibole plates.
4997-5035' (1523.1-1534.7 m) - Amph. unit 4? G.M.D. with mg to m-cg skel. 1x becoming more prominent towards end of section; 1x are aggregated in places but in general similar to unit 3; prominent amphibole plates.
5035-5053' (1534.7-1540.2 m) - Amph. unit 4 G.M.D. with mg to m-cg skel. 1x (aggregated in places); prominent amphibole plates.
5053-5069' (1540.2-1545.0 m) - Strongly sheared G.M.D; 1x are drawn out but still visible.
5069-5070' (1545.0-1545.3 m) - Rock with abundant feldspar; ophitic texture, (core possibly out of order; contacts with sheared rock are missing).
5070-5072' (1545.3-1546.0 m) - Sheared G.M.D.
5072-5094' (1546.0-1552.7 m) - Chl. G.M.D. with prominent mg skel. 1x (unit 5?).
5094 - ?' (1552.7 - ? m) - Chl. G.M.D. with prominent feldspar laths (unit 5 or 6) (contacts on either side are missing; core very disorganised).
5161-5386' (1573.1-1641.7 m) - Amph. to partially chl. unit 7; generally fg with fg to f-mg il; has ophitic texture; very coarse-grained patches at 5237-5239' (1596.2-1596.9 m), 5269-5273' (1606.0-1607.2 m), 5314-5317' (1616.7-1620.6 m), 5347-5349' (1629.8-1630.4 m), 5360-5363' (1633.7-1634.6 m), 5371-5372' (1637.1-1637.4).
5386-5407' (1641.7-1648.1 m) Chl., sheared G.M.D. with f-mg il; highly veined and altered towards end of section.
5407-5409'6" (1648.1-1648.8 m) - Shear zone with segregations of very dark greenish-black chlorite. Some core missing.
5409'6"-5410' (1648.8-1649.0 m) - Quartz vein containing coarse-grained Py and bordered by radiating aggregates of coarse-grained chlorite.
5410-5414' (1649.0-1650.2 m) - Sheared G.M.D.?
5414-5420'4" (1650.2-1652.1 m) - Unmineralized milky quartz vein.
5420'4"-5479' (1652.1-1670.0 m) - Sheared (~60° to side of core), chloritic G.M.D. with f-mg il, some biotite (possibly sheared unit 8). There is a chilled contact with an albite "porphyry" at 5479' (1664.5 m); this is visible despite shearing.
5479-5480'6" (1670.0-1670.5 m) - Albite "porphyry"; sheared but albite phenocrysts not disturbed.
5480'6"-5481'6" (1670.5-1670.8 m) - Sheared, chloritic unit 8?
5481'6"-5497' (1670.8-1675.5) - Albite "porphyry"; weakly sheared.
5497-5516' (1675.5-1681.3 m) - Chl., sheared unit 8?; quartz rich G.M.D. with fg il needles and skel. grains.
5516-5525' (1681.3-1684.0 m) - Core missing.
5525-5535' (1684.0-1687.0 m) - Sheared chl. G.M.D. with skel. il gradually becoming prominent.
5535-5546' (1687.0-1690.4 m) - Chl. unit 8 or 9? G.M.D. with mg to m-cg skel. il; quartz-rich.
5546-5547' (1690.4-1690.7 m) - Chl. unit 8 or 9? G.M.D. with cg skel. il; quartz-rich; epidote-bearing.
5547-5557' (1690.7-1693.8 m) - Chl., weakly sheared unit 9 G.M.D. with mg skel. il.
5557-5573' (1693.8-1699.7 m) - Chl. unit 9 G.M.D. with m-cg skel. il; weakly sheared. At 5573' (1699.7 m) there is a sheared boundary to a porphyry.

5573-5596' (1699.7-1705.7 m) - Albite porphyry, weakly sheared.

5596-5740' (1705.7-1749.6 m) - Chl. unit 9 with mg to m-cg skel. il and lx; weakly sheared; bleached and weakly pyritic at 5603-5604' (1707.8-1708.1 m); not logged in detail.

5740-5741' (1749.6-1749.9 m) - Coarse grained unit 9? with long curved chloritic pseudomorphs after amphiboles. Skel. lx is dominant oxide.

5741-5890' (1749.9-1795.3 m) - Chl. unit 9 with m-cg skel. lx or il. Weakly sheared.

5890-5920' (1795.3-1804.4 m) - Biotite-bearing unit 9 with m-cg skel. lx. Weakly sheared.

5920-6180' (1804.4-1883.7 m) - Fairly altered unit 9 containing epidote, chlorite, some biotite and mg to m-cg skel. lx and il.

6180-6181' (1883.7-1884.0 m) - Banded rock; probably a shear zone.

6181-6284' (1884.0-1915.4 m) - Altered unit 9? containing mg to m-fg il and, in places, mte.

6284-6286' (1915.4-1916.0 m) - Mineralized quartz-carbonate stringer.

6286-6422' (1916.0-1957.4) - Chl. unit 9? with mg to f-mg skel. il and fg il needles.

6422-6446' (1957.4-1964.7 m) - Brecciated fg unit 9?

6446-6459'3" (1964.7-1968.8 m) - Sheared chl. G.M.D. with fg il.

6459'3"-6460'6" (1968.9-1969.2 m) - Contact of G.M.D. with black shales of Boulder Dyke; highly disrupted with considerable quartz/carbonate veining.

6460'6"-6461'2" (1969.2-1969.4 m) - Mixture of G.M.D. and black shale; mixing is at least partially tectonic; primary relationships not distinguishable.

6461'2"-6468' (1969.4-1971.5 m) - Black, pyritic shale with slatey cleavage developed; contains some fragments of a massive grey sediment.

6468-6495' (1971.5-1979.7 m) - Massive silicified grey sediment containing pseudomorphs after sulphates.

6495-6498' (1979.7-1980.6 m) - Sheared black shale containing fragments of a light grey rock.

6498-6500' (1980.6-1981.2 m) - Highly sheared G.M.D.

6500-6510' (1981.2-1984.3 m) - Sheared, chl. unit G.M.D. with m-cg skel. lx.

6510-6539' (1984.3-1993.1 m) - Chl. unit 9 G.M.D. (not logged in detail).

6539-6555' (1993.1-1998.0 m) - Amphibolitic unit 9 with m-cg skel. il.
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0-36'9" (0-11.2 m) - Chl. unit 9 with mg to cg skel il. Minor shears at 33' (10.1 m) and 36' (11.0 m) - Slightly bleached sheared rock with weak pyrite mineralization from 2' to 2'6" (0.6-0.8 m).
36'9"-37'4" (11.2-11.4 m) - Chl., very slightly bleached unit 9 with skel. ilx.
37'4"-38'7" (11.4-11.8 m) - Partly bleached sheared unit 9 with siliceous patches; contains abundant pyrite throughout as fine grained anhedral aggregates; tourmaline developed in siliceous rock.
38'7"-39' (11.8-11.9 m) - Slightly bleached unit 9 with m-cg skel. ilx.
39'-45' (11.9-13.7 m) - Chl. unit 9 with mg skel. Ilx becoming less prominent towards end of section and grading into unit 8.
45-144' (13.7-45.9 m) - Amphibolitic unit 8? with prominent blades of amphibole and minor fine grained oxides.
144-171' (43.9-52.1 m) - Transitional chl. unit 8/9 with mg to fg ilx. Very weakly sheared.
171-178' (52.1-54.3 m) - Chl. unit 9 with mg ilx, weakly sheared with irregular carbonate veins parallel to shearing.
178-200' (54.3-61.0 m) - Mod. sheared chl. unit 9 with mg il. Numerous small carbonate stringers.
200-223' (61.0-68.0 m) - Mod. sheared unit 9 with mg il.
223-226' (68.0-68.9 m) - Mod. sheared zone 9 with sheared mg ilx.
226-232' (68.9-70.7 m) - Chl. unit 9; sheared 1" mineralized zone at 226' (68.9 m). Quartz leader at 226'6" (69.0 m).
232-243'6" (70.7-74.2 m) - Mod. to strongly sheared unit 9 becoming progressively more and more bleached as mineralized zone is reached. Later foot (30 cm) contains disseminated pyrite.
243'6"-245' (74.2-74.7 m) - Strongly sheared, bleached lode. First 8" (20 cm) are highly bleached, siliceous and contain abundant fg Py concentrated along shear planes. Later part is less bleached, contains more disseminated Py and minor tourmaline.
245-248'2" (74.7-75.7 m) - Mod. bleached, sheared rock with dissem. Py; mg skel. il, lx.
248'2"-251'4" (75.7-76.6 m) - Strongly sheared pyritic lode. First foot mod. bleached, remainder strongly bleached. Strongly bleached section consists mainly of carbonate and quartz with finely dissem. Py. Py is more abundant and coarser in mod. bleached section.
251'4"-254' (76.6-77.4 m) - Mod. bleached sheared unit 9 with dissem. Py.
254-257' (77.4-78.3 m) - Mod. sheared, bleached unit 9 with abundant irregular carbonate stringers many of which are mineralized. Dissem. Py throughout.
257-258'6" (78.3-78.8 m) - Unit 9 showing good ophitic texture; chloritic to slightly bleached.
263-265'6" (79.9-80.9 m) - Strongly sheared, mod. bleached unit 9 with abundant fine grained Py. Traversed by narrow quartz stringers.
265'6"-267' (80.9-81.4 m) - Bleached, strongly sheared lode with abundant f-mg Py aligned along shear planes.
267-269' (81.4-82.0 m) - Mod. bleached, sheared rock with disseminated sulphides and mineralized shears.
269-270' (82.0-82.3 m) - Bleached, strongly sheared lode with mod. abundant dissem. Py.
270-273' (82.3-83.2 m) - Mod. bleached, mod. sheared unit 9 with mg lx, sparsely dissem. Py.
273-286' (83.2-87.2 m) - Chl. to weakly bleached, mod. sheared unit 9 with mg lx, sparsely dissem. Py.
286-286'6" (87.2-87.3 m) - Bleached, strongly sheared unit 9 with fg dissem. Py.
286'6"-288' (87.3-87.8 m) - Mod. bleached, sheared rock with dissem. Py.
288'-304' (87.8-92.7 m) - Chl. to weakly bleached unit 9 traversed by numerous irregular carbonate stringers some of which are mineralized; 3" (10 cm) zone of quartz/carbonate stringers at 298' (90.8 m).
304-310' (92.7-94.5 m) - Slight to mod. bleached unit 9 traversed by numerous irregular carbonate stringers.
310-310'3" (94.5-94.6 m) - Bleached, sheared, pyritic lode.
310'3"-313'6" (94.6-95.6 m) - Mod. bleached, sheared rock with dissem. sulphides.
313'6"-314' (95.6-95.7 m) - Bleached, pyritic sheared lode.
314-316' (95.7-96.3 m) - Mod. bleached, sheared rock with dissem. sulphides.
316-317' (96.3-96.6 m) - Bleached, strongly sheared; abundant fine grained Py along shear planes.
317-320' (96.6-97.5 m) - Mod. bleached to bleached rock with dissem. Py.
320-322' (97.5-98.1 m) - Mod. bleached to bleached rock with Py along shears.
323-327' (98.1-99.7 m) - Strongly sheared pyritic bleached lode.
327-328' (99.7-100.0 m) - Very siliceous core of lode; very finely dissem. Py within silica.
328-335' (100.0-102.1 m) - Bleached, strongly sheared pyritic lode with minor siliceous and carbonate rich patches; traversed by thin irregular carbonate stringers.
335-338' (102.1-103.0 m) - Mod. bleached unit 9 with dissem. Py; weak to mod. sheared; contains m-cg il.
338-341' (103.0-103.9 m) - Chl. to weakly bleached unit 9.
341-359' (103.9-109.4 m) - Chl. unit 9 traversed by numerous thin irregular carbonate stringers some of which are mineralized.
359-359'3" (109.4-109.5 m) - Sheared, bleached zone with finely dissem. Py and euheiral mte xls.
359'3"-367' (109.5-111.9 m) - Relatively unsheared unit 9 with ophitic texture. Occasional thin carbonate stringers.
367-372' (111.9-113.4 m) - Mod. sheared unit 9 with abundant carbonate stringers which are quite strongly mineralized at 370' (112.8 m). Chl. to weakly bleached.
372-402' (113.4-124.1 m) - Chl. unit 9 with mg skel. il. Rare carbonate veins.
407-409' (124.1-124.7 m) - Weakly bleached unit 9; mod. sheared with some Py in shears. mg skel. lx.
409-611' (124.7-125.3 m) - Chl. unit 9 with mg skel. il.
411-413' (125.3-125.9 m) - Slightly bleached unit 9 with mg lx.
413-414' (125.9-126.2 m) - Slightly bleached unit 9 with coarse grained Py.
414-416'6" (126.2-126.9 m) - Slightly bleached unit 9 with mg skel. lx.
416'6"-421'6" (126.9-128.5 m) - Chl. unit 9 with m-cg skel. il.
421'6"-425' (128.5-129.5 m) - Weakly sheared chl. unit 9 with minor carbonate stringers.
425-441' (129.5-134.4 m) - Chl. unit 9 with mg skel. il.
441-445' (134.4-135.6 m) - Weak to mod. sheared unit 9 with minor carbonate stringers; sparsely dissem. Py.
445-445'6" (135.6-135.8 m) - Quartz/carbonate vein at 45° to core; weakly mineralized at edges.
445'6"-448' (135.8-136.6 m) - Weakly to mod. bleached, weakly to mod. sheared unit 9 with mg il alternating with fine grained ilmenitic rock.
448-452' (136.6-137.8 m) — Chl. unit 9 with mg il.
452-452'2" (137.8-137.83 m) — Py bearing carbonate vein at 80° to core.
452'2"-464' (137.83-141.4 m) — Chl. to weakly bleached unit 9 with f-mg il
traversed by numerous irregular carbonate veins including one 6" (15 cm) vein
at 459-459'6" (139.9-140.1 m) which shows colloidal textures and minor Py
mineralization especially at the edges. Contains dissemin. Py throughout.
464-465'6" (140.1-141.9 m) — Strongly sheared mod. bleached carbonate rich
lode with mod. abundant dissemin. Py. Lode contains purple anhydrite.
465'6"-467' (141.9-142.3 m) — Chl. unit 9 with mg il; minor carbonate
stringers.
467-467'3" (142.3-142.4 m) — Strongly sheared, carbonate rich zone with
dissemin. Py; only slightly bleached.
467'3"-474' (142.4-144.5 m) — Weakly bleached unit 9 traversed by numerous
carbonate stringers.
474-475' (144.5-144.7 m) — Weakly to mod. bleached; strongly sheared zone
traversed by carbonate veins; mod. abundant dissemin. Py.
475-481'6" (144.7-146.8 m) — Mod. sheared, weakly to mod. bleached rock tra-
versed by numerous small carbonate filled shears; dissemin. Py throughout;
pale purple anhydrite is present in the veins at 475'4" (144.9 m) and 478' 
(145.7 m).
481'6"-483' (146.8-147.2 m) — Strongly sheared, bleached pyritic lode charac-
terized by carbonate and siliceous concentrations; fg and mg Py; first 6" 
(15 cm) are more carbonate rich.
483-491' (147.2-149.7 m) — Mod. sheared, weakly to mod. bleached unit 9 with
mg il traversed by numerous carbonate stringers with associated minor Py
mineralization.
491-500' (149.7-152.4 m) — Chl. unit 9 with m-cg il.
500-507'6" (152.4-154.7 m) — Weakly bleached, mod. sheared unit 9 with abund-
ant minor carbonate veins, weakly mineralized at 501' (152.7 m) where purple
anhydrite is present in vein.
507'6"-509' (154.7-155.1 m) — Sheared, bleached carbonate rich rock with
finely dissemin. Py.
509-513' (155.1-156.4 m) — Mod. bleached sheared rock with dissemin. Py.
513-514' (156.4-156.7 m) — Strongly bleached, carbonate rich rock with coarse
grained pyrite.
514-515' (156.7-157.0 m) — Mod. bleached quartz rich dolerite with f-mg il
and dissemin. Py.
515-583'2" (157.0-177.8 m) - Chl. unit 9 with mg 1x.
583'2" (177.8 m) - End of hole.

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0-98' (0-29.9 m) - Chl. unit 9 with m-cg skel. il. Ophitic texture well
developed up to 40' (12.2 m) after which it becomes less obvious as patchy
carbonate and epidote develop.

98-102' (29.9-31.1 m) - Mod. bleached unit 9 with cg skel. il; mod. sheared
at 60° to core.

102-104' (31.1-31.7 m) - Mod. sheared, mod. bleached lode traversed by quartz
leaders bearing mg euhedral Py and mte.

104-107' (31.7-32.6 m) - Mod. bleached, sheared unit 9.

107-109' (32.6-33.2 m) - Mod. to strongly bleached lode traversed by quartz
leaders.

109-110' (33.2-33.5 m) - Mod. bleached, sheared unit 9.

110-110'8" (33.5-33.7 m) - Mod. bleached lode with mg Py.

110'8"-120' (33.7-36.6 m) - Slightly to mod. bleached unit 9 with mg skel.
il.

120-122' (36.6-37.2 m) - Bleached to mod. bleached lode traversed by quartz
stringers and pink carbonate veins where cg Py is concentrated.

122-142' (37.2-43.3 m) - Chl. unit 9 with 2" (5 cm) pink carbonate vein at
124' (37.8 m) and ½" (1 cm) pink carbonate vein at 129' (39.3 m); many
minor white carbonate veins all approx. perpendicular to the core; contains
cg skel. il.

142-142'6" (43.3-43.4 m) - Mod. sheared at approx. 45° to core; mod.
bleached mineralized zone with fg Py about quartz/carbonate stringer.

142'6"-147' (43.4-44.8 m) - Mod. bleached unit 9.

147-164' (44.8-50.0 m) - Weakly sheared, chl. unit 9 with some epidote
developed. Numerous thin carbonate veins; m-cg skel. il.

164-268'6" (50.0-81.8 m) - Chl. unit 9 with weak irregular shears. Skel. il
varies from m-cg at beginning of section to m-fg at about 200' (61.0 m) and
increasing to mg at end of section.

268'6"-269' (81.8-82.0 m) - Weakly mineralized shear zone.

269-301' (82.0-91.7 m) - Chl. weakly sheared zone 9 with mg skel. il.

301-305' (91.7-93.0 m) - Weakly sheared unit 9 (40° to core) with a ½" (1 cm)
quartz vein at 301'9" (92.0 m) and a 2" (5 cm) carbonate vein at 304' (92.7 m);
mg il, 1x.
305-315' (93.0-96.0 m) - Sheared chl. unit 9 with epidote developed; f-mg skel. lx.
315-319' (96.0-97.2 m) - Chl. unit 9 with mg-cg skel. il; minor carbonate filled shear zones.
319-320'3" (97.2-97.6 m) - Slightly bleached, sheared unit 9 with mg skel. il.
320'3"-322' (97.6-98.1 m) - Strongly sheared bleached lode traversed by irregular carbonate veins. Central portion is highly bleached with abundant f-mg Py along shear planes.
322-330' (98.1-100.6 m) - Slightly bleached unit 9 with irregular carbonate filled shear zones. The section 323-326' (98.5-99.4 m) has long branching amphibole? pseudomorphs; mg skel. il.
330-334'6" (100.6-102.0 m) - Mod. bleached, sheared lode traversed by occasional carbonate veins. Rather weakly dissem. fg Py.
334'6"-336' (102.0-102.4 m) - Mod. bleached, sheared unit 9.
336-354' (102.4-107.9 m) - No. 4 lode. From 336-339' (102.4-103.3 m) the lode is mod. bleached and sheared at approx. 45° to core. Fg Py is developed along shear planes. From 339-347' (103.3-105.8 m), the lode is very siliceous and is almost pure silica at around 345-346' (105.2-105.5 m).
Lenses of carbonate and quartz give the lode a porphyritic appearance at 341'6"-342'6" (104.1-104.4 m) Py is sparsely dissem. in siliceous lode but is more abundant from 346-347' (105.5-105.8 m). From 347-349' (105.8-106.4 m) the lode is bleached and mod. to strongly sheared with abundant mg aggregates of Py. From 349-354' (106.4-107.9 m), the lode is mod. bleached with dissem. Py and euhedral mte; there is a quartz patch at 351'6" (107.1 m).
354-356' (107.9-108.5 m) - Weakly bleached unit 9 with mg skel. il, lx.
356-382' (108.5-116.4 m) - Chl. weakly to mod. sheared unit 9 with mg skel.
il, lx.
382-383'3" (116.4-116.8 m) - Bleached to mod. bleached lode traversed by narrow quartz stringers. Strong shearing at 40° to core.
383'6"-401' (116.8-122.2 m) - Chl. to weakly bleached unit 9 with mg skel.
il, lx.
401-419'6" (122.2-127.9 m) - Weakly to mod. bleached unit 9 with very sparsely dissem. Py. Numerous irregular carbonate filled shears.
419'6"-424'6" (127.9-129.3 m) - 4 E-br lode. Strongly sheared and bleached with siliceous patches between 422'6" (128.8 m) and 424' (129.2 m) traversed by numerous narrow quartz veins. Quartz patch at 420' (128 m).
424'6"-427' (129.3-130.1 m) - Slightly bleached unit 9.
427-453' (130.1-138.1 m) - Chl. unit 9 with mg skel. il.
453-555' (138.1-169.2 m) - Chl. to epidotitic unit 9 with mg-cg skel. 1x.
(c core is not in order at the end of the hole but the lithology remains
essentially the same). There is some m-cg skel. il at 553-555' (168.6-169.2 m).
555' (169.2 m) - End of hole.

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0-88' (0-26.8 m) - Chl. unit 7 with coarse grained patches at 6''-15'' (0.2-
4.6 m), 27'3''-33'' (8.3-10.1 m), 51-56' (15.5-17.1 m), 66-76'6'' (20.1-23.3 m),
becoming generally coarser grained towards end of section. Minor shears at
approx. 80° to core. Quartz veins at 5' (1.5 m) (1'' (2 cm) weakly mineralized),
69'6'' (21.2 m) (5'' (1 cm) at 60° to core).
88-88'6'' (26.8-27.0 m) - Weakly mineralized shear zone.
88'6''-92'6'' (27.0-28.2 m) - Chl. transitional unit 7/8.
92'6''-93' (28.2-28.3 m) - Mod. bleached, sheared rock with aggregate of
massive Py and some dissem. Py.
93-98' (28.3-29.9 m) - Slightly bleached unit 7/8.
98-100'6'' (29.9-30.6 m) - Mod. bleached unit 7/8; mod. sheared with minor
dissem. Py.
100'6''-122' (30.6-37.2 m) - (Only about 5' (1.5 m) of core are actually present-
the missing core is probably that immediately preceding 120' (36.6 m)).
Sheared bleached lode with aggregates of mg Py. Carbonate segregation at
102' (31.1 m). Quartz patch at 120'2''-120'4'' (36.6-36.7 m). Narrow segrega-
tions of mte xls at 120'6'' (36.8 m), siderite veins at 121' (36.88 m),
121'2'' (36.93 m).
122-124'2'' (37.2-37.8 m) - Mod. bleached unit 7/8.
124'2''-126' (37.8-38.4 m) - Bleached, sheared lode with dissem. Py; Quartz
rich patch in centre of lode contains mg euhedral Py.
126-129'3'' (38.4-39.4 m) - Mod. bleached sheared unit 7/8 with sparsely
dissem. sulphides.
129'3''-131' (39.4-39.9 m) - Bleached lode traversed by quartz/carbonate vein
and leaders in first foot of section.
131-132'3'' (39.9-40.3 m) - Mod. bleached unit 7/8.
132'3''-132'9'' (40.3-40.5 m) - Bleached pyritic lode with minor siderite vein.
Py is fg and anhedral.
132'9''-134' (40.5-40.8 m) - Mod. bleached lode showing development of
siliceous and siderite patches containing mg euhedral mte. xls., fg dissem. Py.
134-163' (40.8-49.7 m) – Mod. to strongly bleached unit 7/8.
163-165' (49.7-50.3 m) – Sheared pyritic lode with cg euhedral Py along carbonate filled shear zones.
165-166'6" (50.3-50.7 m) – Mod. bleached sheared unit 7/8.
166'6"-167'6" (50.7-51.1 m) – Bleached sheared lode with f-mg anhedral Py about carbonate vein.
167'6"-176' (51.1-53.6 m) – Mod. bleached sheared unit 7/8 with some Py along shear planes.
176-184' (53.6-56.1 m) – Sheared pyritic lode traversed by ¼" quartz vein at 183'6" (55.9 m).
184-207' (56.1-63.1 m) – Weakly mineralized sheared lode; mod. bleached with quartz/carbonate vein at 193'6" and a purple anhydrite vein at 206' (62.8 m). Veins are associated with more intense mineralization.
207-226' (63.1-68.9 m) – Weakly to mod. bleached zone 8? traversed by narrow carbonated shears with fg il becoming coarser towards end of section.
226-243' (68.9-74.1 m) – Chloritic unit 8.
243-245' (74.1-74.7 m) – Transitional unit 8/9.
245-254' (74.7-77.4 m) – Unit 9 with cg skel. il.
254-260' (77.4-79.2 m) – Chl. transitional unit 8/9 with variation from cg to fg to mg to m-cg through section.
260-268' (79.2-81.7 m) – Mod. sheared unit 8/9.
268-270' (81.7-82.3 m) – Strongly sheared lode with abundant purple anhydrite and fg Py along shears.
270-274' (82.3-83.5 m) – Slightly bleached mod. sheared unit 8.
274-323' (83.5-98.5 m) – Chl. unit 8/9 with generally mg skel. il. Traversed by narrow carbonate filled shears and a 2" (5 cm) pink carbonate filled shear at 314' (95.7 m).
323-324'6" (98.5-98.9 m) – Narrow quartz/carbonate leader with abundant aggregates of mg Py, Po.
324'6"-362' (98.9-110.3 m) – Chl. unit 8/9 with minor carbonate filled shear zones.
362-378' (110.3-115.2 m) – Chl. slightly bleached unit 9 with ophitic texture and mg il.
378-379' (115.2-115.5 m) – Bleached, strongly sheared lode with fg to mg Py.
379-394' (115.5-120.1 m) – Chl. unit 9 with ophitic texture becoming less obvious towards end of section; mg skel. il.
394-394'8" (120.1-121.5 m) – Sheared mod. bleached lode with massive aggregates of fg Py associated with carbonate.
394'8"-432' (121.5-131.7 m) – Chl. unit 8/9 transitional with fg to m-cg skel. il but otherwise like unit 8.

432-435' (131.7-132.6 m) – Mod. bleached unit 8 with fg il; has stockwork of thin mineralized carbonate veins.

435-438'? (132.6-133.5? m) – (Position uncertain as core out of order).
Strongly sheared bleached lode traversed by purple anhydrite veins and irregular patches. Py is fg.

438'?-443'9" (133.5?-135.2 m) – Slightly bleached unit 8/9 with f-mg skel. il.

443'9"-444' (135.2-135.3 m) – 1" (2 cm) carbonate vein strongly mineralized on either side.

444-465' (135.3-141.7 m) – Chl. unit 8/9 with mg skel. il, lx.

465-489' (141.7-149.0 m) – Chl. unit 9 with ophitic texture, mg skel. il.
Minor carbonate filled shear zones slightly mineralized at 478'6" (145.8 m).

489-490'6" (149.0-149.5 m) – Slightly sheared, mod. bleached lode with dissem. mg Py.

490'6"-493' (149.5-150.3 m) – Mod. bleached, sheared unit 9 with mg skel. il.

493-494' (150.3-150.6 m) – Bleached sheared lode with fg Py.

494-498' (150.6-151.8 m) – Mod. bleached, sheared unit 9 with mg skel. il.

498-501'6" (151.8-152.5 m) – Sheared bleached lode with dissem. fg Py.

501'6"-505' (152.5-153.9 m) – Slightly bleached unit 9 with ophitic texture, mg skel. il.

505-506' (153.9-154.2 m) – Mod. bleached, sheared lode with fg Py.

506-508' (154.2-154.8 m) – Slightly bleached, ophitic unit 9 with mg skel. il.

508-511' (154.8-155.8 m) – Sheared, bleached lode with abundant fg Py.

511-525' (155.8-160.0 m) – Weakly bleached, chl. unit 9 with ophitic texture and mg skel. il.

525-530' (160.0-161.5 m) – Sheared, bleached lode with quartz stringers, carbonate patches and f-mg Py aggregates.

530-547' (161.5-166.7 m) – Chl. unit 9 with mg skel. il. Minor carbonate filled shear zones.

547-548' (166.7-167.0 m) – Carbonate filled shear zone with mg euhedral Py along shear planes.

548-642' (167.0-195.7 m) – Chl. unit 9 with dominantly mg skel. il and lx; occasional cg patches; becoming generally finer grained towards end of section.

642-672'6" (195.7-205.0 m) – Chl. unit 9 becoming finer grained towards end of section. Traversed by carbonate filled shear zones.
672'6"-676' (205.0-206.0 m) - Bleached, sheared lode with occasional small quartz/carbonate patches; one of these at 674'6" (205.6 m) contains visible tellurides.

676-695' (206.0-211.8 m) - Chl. unit 9 with m-fg skel. il.

695-697' (211.8-212.4 m) - Fg unit 9 traversed by numerous carbonate stringers some of which are mineralized.

697-704' (212.4-214.6 m) - Chl. unit 10 with well developed variolitic texture.

704-713' (214.6-217.3 m) - Chl. unit 10 with weakly developed variolitic texture.

713-719' (217.3-219.2 m) - Chl. very fg unit 10.

719-722' (219.2-220.1 m) - Mod. bleached very fine grained unit 10.

722-732' (220.1-223.1 m) - Strongly sheared bleached lode marking fault zone between unit 10 and unit 9. From 722' to 726' (220.1-221.3 m), the lode is extremely fine grained and probably sheared unit 10; shearing reaches its greatest intensity at 723-725' (220.4-221.0 m). Throughout this section, the Py is extremely fg but there are occasional mg aggregates. There is a quartz vein at 723' (220.4 m). From 725-726' (221.0-221.3 m), the lode is very siliceous with very fg dissem. Py. From 726-732' (221.3-223.1 m) the lode is coarser grained with f-mg Py. F-mg skel. il first appears at 731' (222.8 m).

732-750' (223.1-228.6 m) - (Core not in order). Bleached unit 9 with mg skel. il. Strongly bleached and mineralized at 738-739'? (224.9-225.5 m ?), 749-750'? (228.3-228.6 m ?).

750-762' (228.6-232.3 m) - Unit 9 with mg skel. il becoming finer grained.

762-763' (232.3-232.6 m) - Carbonate rich slightly bleached unit 9 with fg skel. il.

763-789' (232.6-240.5 m) - Unit 10 with prominent variolitic texture becoming less obvious towards end of section.

789' (240.5 m) - End of hole.

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KE53

0-11' (0-3.4 m) - Chl. unit 9 with cg skel. il.

11-26'11" (3.4-8.2 m) - Sheared unit 9 with skel. il, lx. Minor carbonate veins, some transgressive to shearing; others parallel to it.

26'11"-27'5" (8.2-8.4 m) - Sheared semibleached rock with small carbonate veins and weakly dissem. Py.

27'5"-28.4" (8.4-8.6 m) - Sheared Chl. unit 9 with m-cg skel. il.
28'4"-48' (8.6-14.6 m) - Strongly sheared (approx. 45° to core) unit 9 with m-cg skel. il, 1x. Narrow minor quartz carbonate veins cutting shear planes at approx. 70°. Also irregular carbonate veins parallel to shearing - many associated with weak disem. sulphides as at 28'4"-29' (8.6-8.8 m), 40'6"-41' (12.3-12.5 m).

48-65' (14.6-19.8 m) - Slightly to mod. sheared unit 9 with mg skel. il.

65-73' (19.8-22.3 m) - Chl. unit 9 with well developed ophitic texture and mg il.

73-99'7" (22.3-30.4 m) - Mod. sheared chl. unit 9 becoming more carbonate rich towards end of section. Contains occasional coarse-grained Py xls.

99'7"-100'6" (30.4-30.6 m) - Dark chl. unit 9 traversed by quartz leaders and containing abundant very fg to mg Py. No sign of bleaching about leader which cuts core at approx. 20°.

100'6"-102'3" (30.6-31.2 m) - Mod. sheared chl. unit 9.

102'3"-103'11" (31.2-31.7 m) - Chl. unit 9 traversed by quartz leaders and abundant fg Py.

103'11"-114' (31.7-34.7 m) - Chl. unit 9 showing well developed ophitic texture. mg skel. il becoming finer grained towards end of section.

114-118'7" (34.7-36.1 m) - Mod. sheared transition between unit 9 and unit 8 with f-mg to mg il.

118'7"-128'4" (36.1-39.1 m) - Semibleached unit 8 traversed by quartz leaders at 119' (36.3 m), 119'6" (36.4 m), 120'6" (36.7 m), 121' (36.9 m); by a series of carbonate and chlorite stringers between 124'6" and 125'3" (37.9 and 38.2 m) and by narrow carbonate veins between 127' and 128' (38.7 and 39.0 m). Fine grained Py is disem. throughout this sheared rock and is especially abundant about the quartz leaders, though it is also abundant in the interval 126'4"-127' (38.5-38.7 m) where there are no visible veins.

128'4"-165' (39.1-50.3 m) - Weakly sheared unit 8 with f-mg il; weakly bleached; epidote developed in places with minor disem. sulphides between 154' and 161' (46.9 and 49.1 m).

165-222'11" (50.3-67.9 m). Mod. sheared weakly bleached unit 8 traversed by irregular quartz and carbonate stringers which are generally barren; some mg euhedral Py at 189-190' (57.6-57.9 m).

222'11"-223'3" (67.9-68.0 m) - Irregular quartz carbonate patch with associated Py mineralization.

223'3"-225'11" (68.0-68.9 m) - Mod. sheared, weakly bleached unit 8.
225'11"-227'3" (68.9-69.3 m) – Strongly sheared weakly bleached rock traversed by narrow carbonate stringers. Abundant f-mg Py.
227'3"-253' (69.3-77.1 m) – Weakly sheared unit 8.
253' (77.1 m) – End of hole.

KE54
0'-24' (0-7.3 m) – Chl. unit 8 with f-mg il.
24'-25' (7.3-7.6 m) – Weakly bleached unit 8 with minor fg Py about thin carbonate vein.
25-27'2" (7.6-8.3 m) – Mod. bleached unit 8 with only very narrow quartz leaders. f-mg dissem. Py.
27'2"-32' (8.3-9.8 m) – Weakly bleached unit 8 with minor quartz leader at 29' (8.8 m).
32-62' (9.8-18.9 m) – Chl. unit 8.
62-64'6" (18.9-19.7 m) – Mod. bleached unit 8.
64'6"-65'6" (19.7-20.0 m) – Bleached, pyritic unit 8; py is mg.
65'6"-70'9" (20.0-21.6 m) – Mod. bleached rock traversed by quartz/carbonate (siderite) vein with minor associated Py.
70'9"-105' (21.6-32.0 m) – Chl. unit 8 with f-mg il.
105-106'8" (32.0-32.5 m) – Strongly sheared unit 8 with Py along shear planes.
106'8"-120' (32.5-36.6 m) – Chl. unit 8 with fg il.
120-124' (36.6-37.8 m) – Transitional unit 8/7.
124-134' (37.8-40.8 m) – Fg chl. unit 7.
134-135' (40.8-41.1 m) – Coarse grained patch in unit 7.
135-138' (41.1-42.1 m) – fg chl. unit 7.
138-139' (42.1-42.4 m) – Coarse grained patch.
139-140'3" (42.4-42.7 m) – Strongly sheared unit 7.
140'3"-156' (42.7-47.5 m) – fg chl. unit 7.
156-159'6" (47.5-48.6 m) – Coarse grained patch in unit 7 with very cg skel. il.
159'6"-163'6" (48.6-49.8 m) – Chl. fg unit 7.
163'6"-165' (49.8-50.3 m) – Epidotitic unit 7.
165-168' (50.3-51.2 m) – Chl. unit 7.
168-175' (51.2-53.3 m) – Epidotitic unit 7 becoming coarser grained towards end of section.
175-188'6" (53.3-57.5 m) – Unit 8? with f-mg il and occasional patches with cg skel. il – possibly just a coarser grained version of unit 7; massive Py aggregate at 185'.
188'6"-199' (57.5-60.7 m) – Chl. fg unit 7.
199'-203'9" (60.7-62.1 m) - Coarse grained patch in unit 7 with mg skel. il.
203'9"-223' (62.1-68.0 m) - Fg chl. unit 7.
223'-224' (68.0-68.3 m) - Coarse grained patch.
224'-226' (68.3-68.9 m) - Fg chl. unit 7.
226'-230' (68.9-70.1 m) - Coarse grained patch.
230'-232' (70.1-70.7 m) - Fg chl. unit 7.
232'-233'6" (70.7-71.2 m) - Coarse grained patch.
233'6"-260'6" (71.2-79.4 m) - Fg chl. unit 7.
260'6"-262'8" (79.4-80.1 m) - Coarse grained patch.
262'8"-265' (80.1-80.8 m) - Fg chl. unit 7. Quartz stringer and minor dissem. Py at 265' (80.8 m).
265'-268' (80.8-81.7 m) - Coarse grained patch.
268'-268'6" (81.7-81.8 m) - Finer grained unit 7 with zone of cg il at end of section.
268'6"-269' (81.8-82.0 m) - Typical Charlotte type vein approx. ¼" wide at approx. 30° to core with narrow bleached halo.
269'-271' (82.0-82.6 m) - Fg chl. unit 7.
271'-272'6" (82.6-83.1 m) - Coarse grained patch.
272'6"-322' (83.1-98.1 m) - Fg chl. unit 7 with minor irregular carbonate veining.
322'-333' (98.1-101.5 m) - Highly bleached and sheared carbonate rich rock with dissem. Py.
333' (101.5 m) - End of hole.

KE55
0-34'9" (0-10.6 m) - Chl. unit 8.
34'9"-35'8" (10.6-10.9 m) - Sheared, mod. bleached unit 8 with fg dissem. Py.
35'8"-45' (10.9-13.7 m) - Weakly to mod. bleached unit 8.
45-51' (13.7-15.5 m) - Chl. unit 8.
51-59'1" (15.5-18.0 m) - Weakly to mod. bleached unit 8 transitional to unit 7 at end of section.
59'1"-65'7" (18.0-20.0 m) - Mod. to strongly bleached unit 7/8 with 2" (5 cm) quartz vein at 62'8" (19.1 m) and an 8" (20 cm) irregular quartz/feldspar vein containing a dark greenish black material at 63-63'8" (19.2-19.4 m); dissem. fg Py throughout especially along shear planes.
65'7"-83'6" (20.0-25.5 m) - Fg chl. unit 7.
83'6"-84' (25.5-25.6 m) - Typical Charlotte type vein approx ¼" (1 cm) wide with narrow bleached halo.
84-87' (25.6-26.5 m) - Fg chl. unit 7.
87-87'6" (26.5-26.7 m) - Mineralized shear zone with shearing at 45° to core.
98-144' (29.7-34.5 m) - Fg chl. unit 7.
114-121' (34.5-36.9 m) - Coarse grained patch.
121-131' (36.9-39.9 m) - Fg chl. unit 7.
131-134' (39.9-40.8 m) - Coarse grained patch; mod. bleached with Py along minor stringers towards end of section.
134-138' (40.8-42.1 m) - Mod. bleached fg unit 7.
138-160'8" (42.1-49.0 m) - Fg chl. unit 7.
160'8"-161'6" (49.0-49.2 m) - Coarse grained patch.
161'6"-167'6" (49.2-51.1 m) - Fg chl. unit 7.
167'6"-168'6" (51.1-51.4 m) - Sheared unit 8? or coarse grained patch in unit 7.
168'6"-183' (51.4-55.8 m) - Chl. fg unit 7.
183-185' (55.8-56.4 m) - Coarse grained patch.
185-205' (56.4-62.5 m) - Strongly bleached unit 7 traversed by quartz/carbonate stringers at 200'8" (61.2 m) and 201' (61.3 m). Mainly fg il but cg patches at 200'8" (61.2 m) and 202-203' (61.6-61.9 m). Fg Py about stringers; coarser dissem. Py elsewhere.
205-223'4" (62.5-68.1 m) - Fg chl. unit 7.
223'4"-223'10" (68.1-68.2 m) - Bleached pyritic rock traversed by quartz leader.
223'10"-225'6" (68.2-68.7 m) - Fg chl. unit 7.
225'6"-228'6" (68.7-69.6 m) - Coarse grained patch.
228'6"-230' (69.6-70.1 m) - Fg chl. unit 7.
230-232' (70.1-70.7 m) - Coarse grained patch.
232-246' (70.7-75.0 m) - Fg chl. unit 7.
246-247' (75.0-75.3 m) - Coarse grained patch.
247-252'8" (75.3-77.0 m) - Chl. fg unit 7, with minor irregular carbonate veins.
252'8" (77.0 m) - End of hole.
Appendix 3

ELECTRON-MICROPROBE TECHNIQUES

All analyses were performed on a JEOL JXA-5A electron probe microanalyser fitted with three spectrometers and an energy dispersive analyser. The take off angle for all analyses was 40°. Other operating conditions were varied to suit the minerals being analysed.

Analyses were corrected for background, dead time, drift, absorption, fluorescence and atomic number effects using the computer program of Mason et al. (1969).

(i) Tellurides
Accelerating voltage: 20 kV.
Beam current: $\sim 4 \times 10^{-8}$ amps.
No. of peak measurements: 3.
No. of background measurements: 1.
Counting time: 15 seconds.
Lines and crystals used: Te $L\alpha_1$ (PET), Au $L\alpha_1$ (LIF), Ag $L\alpha_1$ (PET), Fe $K\alpha_1$ (LIF), S $K\alpha$ (PET), Co $K\alpha$ (LIF), Cu $K\alpha_1$ (LIF), Sb $L\alpha_1$ (PET), Pb $M\alpha$ (PET).
Standards: Pure metals except for Fe and S, for which troilite was used.
Interferences: Pure Te gives 0.58% Sb – all analyses were corrected for this by hand prior to computer processing by assuming a linear relationship between the interference and the amount of Te present.

(ii) Gold
Accelerating voltage: 20 kV.
Beam current: $\sim 4 \times 10^{-8}$.
No. of peak measurements: 3.
No. of background measurements: 1.
Counting time: 15 seconds.
Lines and crystals used: Au $L\alpha_1$ (LIF), Ag $L\alpha_1$ (PET), Te $L\alpha_1$ (PET), Fe $K\alpha_1$ (LIF), S $K\alpha$ (PET), Co $K\alpha$ (LIF), Cu $K\alpha_1$ (LIF).
Standards: Pure metals except for Fe and S, for which troilite was used.
Special conditions: For gold grains less than 10 microns across, a method employed by the late R. Britten (pers. comm.) for analysing very small grains in lunar rocks, was used. This involved displaying the grain with the probe in the scanning mode, centering a very small beam on the sample and analysing the grain with the beam in spot mode. During analysis, the position of the
beam was adjusted to keep the absorbed current to a minimum since gold has a much lower absorbed current than sulphides or silicates. For very small grains it was impossible to stop the beam wandering onto the surrounding minerals. In most cases, pyrite was the surrounding mineral and by measuring Fe and S simultaneously, it was possible to correct for the Fe contributed by the pyrite but the corrected analyses are obviously less accurate than the analyses made on large gold grains. Where Fe-bearing silicates or carbonates were in contact with the grains, it was impossible to measure the Fe in the gold and analyses have been corrected to 100% on the basis of their Au, Ag and Cu values alone.

(iii) Tetrahedrite
Accelerating voltage: 20 kV.
Beam current: $\sim 4 \times 10^{-8}$ amps.
No. of peak measurements: 3.
No. of background measurements: 1.
Counting time: 15 seconds.

Lines and crystals used: As Kα (LIF), Sb Lα₁ (PET), Cu Kα₁ (LIF), Te Lα₁ (PET), Fe Kα₁ (LIF), S Kα (PET), Zn Kα₁ (LIF), Ag Lα₁ (PET).
Standards: Pure metals except for Fe and S, for which troilite was used.
Interferences: Cu interferes with the Zn background at the normal background distance of 300 steps; to avoid this, a background distance of 350 steps was used.

(iv) Sulphides
Accelerating voltage: 15 kV.
Beam current: $\sim 2 \times 10^{-8}$ amps.

No. of peak measurements: 3 for major elements; 10 for trace elements.
No. of background measurements: 1 for major elements; 5 for trace elements.
Counting time: 15 seconds for major elemental analyses; 60 seconds for minor element analyses.

Lines and crystals used: Fe Kα₁ (LIF), S Kα(PET), As Kα (LIF), Ag Lα₁ (PET), Co Kα (LIF), Ni Kα₁ (LIF), Cu Kα₁ (LIF), Ti Kα₁ (LIF).
Standards: Pure metals except for Fe and S, for which troilite was used.
Special conditions: Long counting times were used to measure trace amounts of Co, Ni, As and Ti in pyrite. In order to prevent cracking of the carbon coating, the specimen position was changed every sixty seconds.
The lower limits of sensitivity for Co, Ni, As and Ti can be calculated using the formula:

\[
P = \frac{4.7 \sqrt{C_B}}{C_S \sqrt{T}}
\]

where \( C_B \) = counts on background
\( C_S \) = No. of counts/unit time/% element in standard
\( T \) = counting time

These limits are 0.01, 0.01, 0.07 and 0.008 respectively.

(v) *Silicates*

Accelerating voltage: 15 kV.
Beam current: \( \sim 2 \times 10^{-8} \) amps.
No. of peak measurements: 3.
No. of background measurements: 1.
Counting time: 15 seconds.

Lines and crystals used: Ti \( \text{K}\alpha_1 \) (LIF), V \( \text{K}\alpha_1 \) (LIF), Cr \( \text{K}\beta \) (LIF), Fe \( \text{K}\alpha_1 \) (LIF), K \( \text{K}\alpha \) (PET), Ca \( \text{K}\alpha \) (PET), Ba \( \text{L}\alpha \) (PET), Mn \( \text{K}\alpha \) (PET), Na \( \text{K}\alpha \) (KAP), Mg \( \text{K}\alpha \) (KAP), Al \( \text{K}\alpha \) (KAP), Si \( \text{K}\alpha \) (KAP).

Standards: Ti - rutile, V - V metal, Cr - Cr metal, Fe - anandite, K - potassium tantalate, Ca - wollastonite, Ba - anandite, Mn - Mn metal, Na - jadeite, Mg - MgO, Al - \( \text{Al}_2\text{O}_3 \), Si - \( \text{SiO}_2 \). A standard kaersutite crystal was used as an internal standard.
Appendix 4

NEUTRON ACTIVATION TECHNIQUES

Sample Preparation

After cleaning and crushing, about 300 mg of each sample were weighed into clean vials prepared from high purity silica tubing. After sealing, the samples were wrapped in aluminium foil and placed in an irradiation can together with standards. In most experiments, two irradiation cans containing five or six samples and two standards each were used.

Standard Preparation

Stock standard solutions of appropriate concentrations (0.5-10 mg) were all prepared from "Specpure" chemicals obtained from Johnson, Mathey and Company with the exception of Sb which was prepared from "Analar" potassium antimony tartrate. The Te, Se, Ag and Tl standards were all prepared by dissolving the "Specpure" elements in concentrated HNO₃ and then diluting to 6N HNO₃ in the case of Te, Se and Tl, and 1N in the case of Ag. The Au and Pd standards were prepared by dissolving the "Specpure" metals in aqua-regia, evaporating several times with concentrated HCl to drive off the HNO₃, and taking up in 2N HCl. The Co standard was prepared by dissolving Co sponge in concentrated HCl, then diluting to 2N HCl with distilled H₂O. The Ir standard was prepared by dissolving "Specpure" ammonium chloridirate in 2N HCl. The As standard was prepared by dissolving the metal in concentrated HNO₃, evaporating to incipient dryness and taking up in 1N NaOH. Potassium antimony tartrate was dissolved in water for the Sb standard.

Aliquots of various combinations of stock standard solutions were diluted for use as sub-standard solutions. The combinations used were: Au-Ir-Pd-Co; Te-Se-Ag-Tl; and As-Sb. The concentrations of these sub-standard solutions were about 3 µg Au/g; 20 µg Ir/g; 100 µg Pd/g; 5 µg Co/g; 150 µg Te/g; 150 µg Se/g; 150 µg Ag/g; 150 µg Tl/g; 150 µg As/g; and 200 µg Sb/g.

Standard ampoules were prepared from high purity "Supersil" silica tubing (0.3 mm O.D., 0.2 mm I.D.). After addition of about 20 mg of "Specpure" silica each tube was accurately weighed to ± 0.03 mg. About 30 mg of the Au-Ir-Pd-Co sub-standard was then added to each tube, weighed accurately and spun in a centrifuge. The standards were then evaporated to dryness in an oven at 100°C. The Te-Se-Ag-Tl standard solution was then added to the ampoules and dried down at 65-70°C. Finally, the As-Sb solution was added, evaporated at 65-70°C and when dry, the ampoules were sealed.
Carrier Preparation

Carrier solutions were prepared from analytical or laboratory grade chemicals in the same way as the standard solutions with the following exceptions: Au was taken up in 1N HNO₃ instead of 1N HCl (since As, Sb, Se and Te are highly volatile in the presence of chloride ion); Co was dissolved in concentrated HNO₃ and diluted to 1N HNO₃; As₂O₃ was used instead of As metal, and was dissolved in 2N NaOH; Tl COOH was used instead of Tl metal, and was dissolved in 6N HNO₃; AgNO₃ was used instead of Ag metal, and was dissolved in water. Concentrations of the carrier solutions were approximately 10 mg/ml for Au, Pd, Te, Se, Tl and Sb; 2 mg/ml for Ir and 20 mg/ml for As, Co and Ag; the weight of element used was about 20 mg for Au, Co, Pd, Te, Se, Tl, As and Sb, 4 mg for Ir and 40 mg for Ag.

Irradiation of Samples and Standards

Samples and standards were wrapped in aluminium foil, placed in an aluminium irradiation can and sent to the A.A.E.C. nuclear reactor, HIFAR, at Lucas Heights, Sydney for irradiation at a thermal neutron flux of about 6 x 10¹² neutrons/cm²/sec. Irradiation times were varied to suit the samples being analysed: samples with low Te levels were irradiated for 6 to 7 days; ore-grade samples were irradiated for only 1 or 2 days. Because of the large amount of short-lived radioactivity (principally ²⁴Na), it was necessary to "cool" the samples for 3 to 4 days before commencing work on them.

The following radionuclides were chosen for determination: ¹⁹⁸Au (t½ = 2.69 d), ¹¹⁰Ag (t½ = 255 d), ¹²³Te (t½ = 117 d), ⁷⁵Se (t½ = 120 d), ²⁰⁴Tl (t½ = 3.4 yr), ⁷⁶As (t½ = 26.5 hr), ¹²⁴Sb (t½ = 60.3 d), ⁶⁰Co (t½ = 5.26 yr), ¹⁹²Ir (t½ = 74.2 d) and ¹⁰⁹Pd (t½ = 13.46 hrs).

Radiochemical Processing

(i) Au-Ag-Te-Se-As-Sb-Co + Tl

Prior to the return of the irradiated samples, nickel crucibles for both standards and samples were cleaned in hot aqua regia. A little distilled water and about 6 pellets of NaOH were added to each crucible. Then 2 ml of Au, Ag, Te, Se and if required, Tl, carriers were pipetted into each crucible. Approximately 1 ml of Co carrier was then added using a disposable pipette. If the solution turned blue, this indicated that it was still basic; if not, further pellets of NaOH were added until the solution did turn blue. These carriers were then evaporated to dryness on a hot plate.
When dry, the crucibles were cooled, 2 mls of Sb and 1 ml of As carrier were added and the crucibles replaced on the hot plate to gently evaporate the carriers.

Immediately prior to fusion of each sample, approximately 3 g of NaOH was added to the dried crucible, the irradiated sample powder was carefully transferred from the silica vial to the crucible, approximately 3 g of Na₂O₂ was sprinkled over the contents of the crucible and the crucible was covered with a lid.

The crucible was then heated very gently over a Meker burner until all water was driven off and the contents were blackened. Then the heat was gradually raised until the contents were a dull red colour and the sample was fused for a further five minutes with intermittent swirling of the melt to aid in dissolution and mixing. The crucible was then allowed to cool.

Standards were treated in exactly the same way as the samples except that after the contents of the silica vial had been tipped into the crucible, the broken vial was added to the crucible and fused with the carriers.

After the samples had been fused and cooled, the fusion cakes were leached with distilled water in covered, pre-weighed 250 ml beakers. When reaction subsided, the contents of each crucible were washed into the beakers using about 50 ml of water and finally about 2 ml of 6N HCl.

Concentrated HCl was carefully added to the fusion solution until it was just acid (this is indicated by the disappearance of the grey-black precipitate around the sides of the beaker and the formation of a cloudy white precipitate in a green solution). The fusion solution was then diluted to 100 to 150 ml using distilled water and an aliquot of 10 to 15 ml weighed into a polythene bottle for Co determination.

Initial Separation

The following method was used to separate different groups of elements:

1. Centrifuge the fusion solution to remove the precipitate of AgCl and SiO₂ (CAB-O-SIL may be added to nucleate the SiO₂). Wash the precipitate twice with 0.75N HCl and add the washings to the supernatant. Retain the precipitate for purification of AgCl.

2. Filter the supernatant onto an ion exchange column containing 3 g BIO-RAD AG-1 X8, 100-200 mesh anion exchange resin (chloride form) which has been pre-treated with dilute HCl. Au (and Tl) are retained on the resin. The eluant contains the As, Sb, Se and Te.
3. Bubble H₂S through the eluant until saturated (~5 minutes) in order to precipitate the As and Sb as sulphides. Heat gently to aid precipitation and coagulation of these sulphides. Leave to stand for several hours, then centrifuge. Retain the supernatant for precipitation of Te and Se. Wash the As/Sb sulphide precipitate and add the washings to the supernatant.

4. Wash the columns with 10 ml of concentrated HCl, 20 ml of 2N HCl, 20 ml of 0.2N HCl and 20 ml of H₂O.

5. If Tl is being determined, elute with 60 ml of 1M H₂SO₄ saturated with SO₂. Then wash columns with 10 ml of water, add 10 ml of aqua regia to redissolve Au, wash with 20 ml 2M HCl and two 5 ml aliquots of water.

6. Elute the Au with 150 ml of 0.1M thiourea in 0.1M HCl.

**Purification of As and Sb**

The method used is that of Ewers (1975) and Ewers and Keays (1977).

1. Dissolve the As and Sb precipitate in 3 ml of 10% ammonium sulphide solution, dilute with H₂O to 10 ml and centrifuge down any insoluble material.

2. Acidify the supernatant cautiously with 6N HCl until a flocculent precipitate of As and Sb sulphides is obtained.

3. Centrifuge and discard the supernatant after testing it with several drops of HCl for any further precipitation.

4. Dissolve the precipitate in 10 ml of 3N NaOH and 1 ml of H₂O₂.

5. Heat the solution at about 70°C (in a waterbath) for 5-10 minutes with dropwise addition of H₂O₂ until most of the sulphur has been oxidized.

6. Cool the solution and then add 15 ml of 10N HCl and 0.5 g of Na ClO₃.

7. Continue heating with further dropwise addition of H₂O₂.

8. Filter and add about 4 g of NaH₂PO₄ to the filtrate.

9. Heat the solution in a covered beaker on a hot plate until the As metal precipitates and coagulates.

10. Centrifuge down the As and retain the supernatant for further purification of Sb.

11. Wash the As precipitate with water, then boil the precipitate in 10 ml of hot 5% tartaric acid to remove any soluble Sb, centrifuge and rewash the precipitate with water.

12. Filter the As onto a weighed fibreglass filter pad, wash with water and then alcohol. Dry in an oven for 15 minutes at 105°C, cool for 10 minutes, weigh and mount on an Al tray.

13. To the supernatant obtained in step 10, add 10 mls of saturated oxalic acid solution and saturate the solution with H₂S.
14. Gently warm the precipitate to coagulate the orange precipitate of Sb₂S₃.
15. Dissolve the Sb₂S₃ in 5 mls of concentrated HCl and centrifuge to remove any sulphur.
16. Add the supernatant to 20 ml of saturated oxalic acid solution and bubble H₂S through the solution until Sb₂S₃ commences to precipitate (H₂S should not be bubbled through the solution for too long or sulphur will precipitate).
17. Filter the Sb₂S₃ onto a weighed fibreglass filter pad, wash with water and then alcohol. Dry in an oven for 15 minutes at 50°C, cool for 10 minutes, weigh and mount on an Al tray.

Purification of Au

The method used is modified after that of Crocket et al. (1968) and Keays et al. (1974).
1. Evaporate the Au fraction (from step 6 of the initial separation) to dryness on a hot plate using heat lamps.
2. Add dropwise 10 ml 50% HNO₃ and warm gently (violent reaction).
3. When reaction ceases, add 15 ml concentrated HCl and digest for 20 minutes on hot plate.
4. Remove watchglass and evaporate to near dryness.
5. Drive off HNO₃ with two 5 ml aliquots of concentrated HCl.
6. Take up in 10 ml of 2N HCl and heat to dissolve all salts.
7. Filter, add 15 ml of H₂O and 15 ml of 1N HCl saturated with SO₂ and warm on hot plate to precipitate Au.
8. Swirl to aggregate the Au, decant supernatant and wash with 6N HCl, then water.
9. Carefully filter the Au onto a weighed fibreglass filter pad, wash with water, then alcohol. Dry in an oven at 110°C for 15 minutes, cool and weigh. After weighing, fix the Au to the filter pad using collodion in ether and mount on an Al tray.

Purification of Tl

The method used is modified after that of Hughes (1978).
1. Evaporate the Tl fraction (from step 5 of the initial separation) to near dryness (marked by fumes of H₂SO₄).
2. Add 5 ml distilled H₂O, 5 ml of H₂O saturated with SO₂, 2 ml La carrier (10 mg) and 1 ml Ca carrier (in dilute HNO₃) (10 mg).
3. Add 2 drops of universal indicator.
4. Make basic with NaOH (1–2 ml).
5. Centrifuge to remove the La and Ca precipitate.
6. To the supernatant add acetic acid dropwise until acid (~ ½ ml).
7. Immediately prior to filtering each sample, add 5 ml 10% KI to precipitate Tll.
8. Filter the Tll onto a weighed fibreglass filter pad, wash with water, then alcohol. Dry in an oven at 100°C for 15 minutes, cool for 10 minutes, weigh and mount on an Al tray.

Purification of Ag

1. Dissolve AgCl precipitate from step 1 of the initial separation in NH₃.
2. Centrifuge to remove SiO₂, transfer supernatant to a clean tube.
3. Precipitate AgCl from the supernatant with 1 ml of concentrated HCl, then concentrated HNO₃ to methyl orange end point.
4. Wash precipitate with 0.2N HCl, then H₂O.
5. Dissolve the precipitate in NH₃.
6. Add 1 ml Fe carrier while swirling tube.
7. Centrifuge Fe(OH)₃ down. Transfer supernatant to a clean tube.
8. Repeat steps 3 to 5 twice.
9. Immediately prior to filtering each sample, add 1 ml of concentrated HCl, then concentrated HNO₃ to methyl orange end point.
10. Filter the AgCl precipitate onto a weighed fibreglass pad before it has time to coagulate, wash with water, then alcohol. Dry in an oven at 110°C for 15 minutes, cool for 10 minutes (away from direct sunlight), weigh and mount on an Al tray.

Purification of Se and Te


1. Gently warm the supernatant from step 3 of the initial separation scheme on a hotplate (~ 80°C) with periodic additions of water saturated with SO₂ to reduce the Se VI and Te VI to Se O and Te O. The reduction takes 1 to 2 days; the solution goes grass green at the end point.
2. Heat for a further 2 hours to drive off all SO₂.
3. Remove the beakers from the hot plate, cool for 10 minutes then add 10 ml of 10% SnCl₂ in 6N HCl to precipitate most of the Se and Te.
4. Centrifuge down the Se and Te; retain the supernatant; wash the Se and Te twice with water and add washings to the supernatant.
5. Leave the supernatant overnight to test for further precipitation of Se and Te.
6. Next morning, add a further 5 ml of 10% SnCl₂ in 6N HCl; if there is any further precipitation, centrifuge and add precipitate to that from step 4.
7. Repeat steps 5 and 6 until there is no further precipitation of Se and Te (this may take 3 days).
8. Wash the Se-Te precipitate twice with H₂O.
9. Dissolve the precipitate in 2 mls of concentrated HNO₃ and evaporate to dryness over a bunsen burner.
10. Cool and add 5 ml of concentrated HCl and about 10 mg of NaClO₃.
11. Add to an ion exchange column containing 3 g BIO-RAD AG-1X8, 100-200 mesh anion exchange resin (chloride form) which has been conditioned with 20 ml of 6N HCl and two 5 ml aliquots of concentrated HCl.
12. Elute Se from the column with 40 ml of 3N HCl.
13. Elute Te from the column with 45 ml of 0.3N HCl.
14. To the Se eluant, add holdback carriers for Ir, Co, Cr and Cu (~1 mg each) in concentrated HCl.
15. Precipitate red Se with about 50 mg of NaHSO₃; warm gently to coagulate the precipitate.
16. Centrifuge, wash and dissolve the precipitate in 50% HNO₃ and evaporate to near dryness.
17. Add 5 ml of 5N NaOH to the residue, add about 2 mg of Fe carrier, centrifuge down Fe(OH)₃ and transfer supernatant to a clean tube; wash solid with 5 ml H₂O and add washings to supernatant.
18. Acidify the supernatant to about 5N HCl.
19. Add about 50 mg of NaHSO₃ to the solution to precipitate red Se.
20. Centrifuge and wash the Se precipitate with H₂O.
21. Filter the Se onto a weighed fibreglass filter pad, wash with water, then alcohol. Dry in an oven for 15 minutes at 105°C to give grey Se as the stable allotrope, cool for 10 minutes, weigh and mount on an Al tray.
22. To the Te eluant, add holdback carriers for Ir, Co, Cr and Cu (~1 mg each) in concentrated HCl.
23. Add about 50 mg of NaHSO₃; warm gently on hotplate; add 5 ml of saturated N₂H₄.H₂SO₄ solution and 10 ml of H₂O saturated with SO₂ to aid precipitation and coagulation of the Te.
24. Centrifuge, wash precipitate with two 5 ml aliquots of water and dissolve precipitate in 50% HNO₃. Add 2 mg of Se in 50% HNO₃ as a holdback carrier.
25. Heat over a bunsen burner; add a few drops of 10% KI to remove iodine, then evaporate to near dryness.

26. Dissolve the residue in 5 ml of concentrated HCl and add 5 ml of cold concentrated HCl saturated with SO₂.

27. Allow the solution to stand for approximately one hour until the red Se has settled.

28. Filter off Se and dilute solution to 25 ml.

29. Heat the solution on a hot plate, add 10 ml of H₂O saturated with SO₂, then 5 ml of saturated N₂H₄.H₂SO₄ solution and finally, 10 ml more of H₂O saturated with SO₂. Heat until the precipitate settles in an easily filterable form.

30. Centrifuge and wash precipitate with H₂O.

31. Filter the Te onto a weighed fibreglass filter pad, wash with water, then alcohol. Dry in an oven for 15 minutes at 105°C, cool for 10 minutes, weigh and mount on an Al tray.

(ii) Au-Ir-Pd-Co

Carriers for Au, Ir, Pd and Co were pipetted into clean crucibles and evaporated to dryness prior to the return of the irradiated samples.

The samples were fused, together with carriers and about 2 gm of NaOH and 2 gm of Na₂O₂ at bright red heat for 10 minutes over a Meker burner. When cool, the crucible, and lids were transferred to weighed 250 ml beakers and the fusion cake leached with water. The contents of the crucibles were then washed into the beakers using about 40 ml of water, then 5 ml 6N HCl followed by several ml of hot aqua regia. The fusion solutions were then acidified with about 30 ml of concentrated HCl and 5 ml of concentrated HNO₃. When the solutions were cool, aliquots were weighed out into poly-bottles for determination of Co.

Initial Separations

The methods used were essentially those of Keays and Kirkland (1972).

1. Take fusion solution to dryness on a hot plate under heat lamps.

2. Take up residue in 0.75N HCl - 0.004M Ce(IV) solution.

3. Centrifuge down silica residue; wash twice with 0.75N HCl and add washings to supernatant.

4. Filter supernatant into anion exchange columns containing 1.6 g BIO-RAD AG-1X8, 200-400 mesh (chloride form) which has been pre-treated with 20 ml of 0.75N HCl - 0.004M Ce(IV) solution.
5. Wash resin with 20 ml of 0.2N HCl.
6. Wash resin with 15 ml of 2N HCl.
7. Wash resin with 20 ml of concentrated HCl.
8. Elute Pd with 80 ml of concentrated HCl into 250 ml beaker containing holdback carriers for Au, Ir, Co, Cr and As (~10 mg of each) and retain for Pd determination.

**Procedure for Au and Ir**

Wash anion-exchange resin into a 12 dram plastic vial for counting.

**Purification of Pd**

1. Evaporate the Pd fraction to a volume of 5 ml.
2. Add 20 ml of H_2O saturated with SO_2.
3. Cover with watch glass and heat for 5 minutes to precipitate the Au holdback carrier.
4. Filter solution, while still hot, into a glass centrifuge tube containing 5 ml of 1% dimethylglyoxime in alcohol to form Pd(C_4H_7O_2N_2)_2.
5. Centrifuge and wash precipitate twice with 0.2N HCl. If precipitate appears dirty (i.e. is not bright yellow, dissolve in aqua regia, drive off the HNO_3 with HCl and add 20 ml of H_2O, then 5 ml of 1% dimethylglyoxime in alcohol to reprecipitate Pd.
6. Filter onto a weighed fibreglass filter pad, wash with water, then alcohol. Dry in an oven for 15 minutes at 100°C, cool for 10 minutes, weigh and mount on an Al tray.

**Counting Procedures**

Au, As, Sb, Se, Co, Ir and all high-level Ag samples were γ-counted on a 50 cc Canberra Industries Ge[Li] detector; low level Ag samples were γ-counted on a 3" x 3" Bicron NaI(Tl) detector; Te was X-ray counted on a Princeton Gamma Tech LEPS; and Tl and Pd were β-counted on a Beckman Widebeta counter. For all elements counted on the Ge[Li] and LEPS detectors, determinations were made by summing the counts in the six channels containing the major photopake and subtracting the background from the three channels on either side of the peak and comparing the countrate with that of the standards after making yield and decay corrections. To minimize statistical errors, counting times were adjusted where possible to obtain a net peak count of 5000 counts. For some very low level samples, less counts had to suffice and the number of significant figures on these samples has been reduced accordingly. For the samples counted on the NaI(Tl) detector,
peak height above background was used rather than the area under the peak because of the lower resolution of this detector. For Tl and Pd, the count rate was determined by subtracting the blank from the given count. In the case of Pd, the decay curve was followed for at least 3 half lives to check for impurities. In addition to decay and yield corrections, self-absorption corrections must also be made for these β-counted samples (see Ewers, 1975).

Sensitivity limits in grams are approximately: Au: \(1 \times 10^{-12}\), Ag: \(1 \times 10^{-9}\), As: \(1 \times 10^{-11}\), Sb: \(5 \times 10^{-11}\), Se: \(1 \times 10^{-8}\), Te: \(3 \times 10^{-9}\), Tl: \(5 \times 10^{-9}\), Co: \(1 \times 10^{-10}\), Pd: \(1 \times 10^{-9}\) and Ir: \(1 \times 10^{-10}\).
Appendix 5

METHODS FOR THE DETERMINATION OF BASE METALS, K, Na, S AND CO₂

(i) Atomic Absorption and Flame Photometry

Atomic absorption was used for the determination of Cu, Ni, Pb, Zn, Cr and V and flame photometry for K and Na on a single sample aliquot.

Preparation of Sample

1. Accurately weigh out about 1 gm of sample powder onto a watch glass.
2. Transfer the sample to a PTFE crucible.
3. Add 10 ml of concentrated hydrofluoric acid, 1 ml of concentrated nitric acid and 2 ml of concentrated perchloric acid.
4. Cover the crucible with a lid and place in an asbestos hot box on a small hotplate (12 samples per hot box). Leave to evaporate overnight.
5. The following morning, remove the hot box and crucibles from the hot plate and allow to cool for 20 minutes.
6. Wash down the lids of the crucibles with distilled water, and 3/4 fill each crucible with distilled water.
7. Add 2 ml of perchloric acid and stir to mix.
8. Replace on hot plate and heat gently for at least one hour to dissolve the various oxides.
9. Remove from the hotplate and allow to cool.
10. Transfer the contents of the crucibles to a 50 ml standard flask, dilute to 50 ml with 2N HCl.
11. Pipette 5 ml of this solution into a 200 ml standard flask, and add 40 ml of Al(NO₃)₃ solution. Make the solution up to 200 ml with 1% v/v perchloric acid for determination of K and Na.

Preparation of Standards for K and Na

1. Weigh exactly 2.5418 g AR NaCl and 1.9090 g AR KCl which has been dried to constant weight into a 100 ml beaker.
2. Dissolve in 1% v/v perchloric acid, transfer quantitatively to a litre flask and make up to mark with 1% v/v perchloric acid. This gives a stock solution which is 1000 ppm in Na and K.
3. Prepare 100 ppm and 50 ppm standard solutions from the stock solution; all standards must be 1% v/v perchloric acid and 20% v/v Al(NO₃)₃ (Al is used to suppress the flame emission of Ca). From the 100 ppm solution prepare 25, 20, 10, 5, 2 and 1 ppm standard solutions in 1% v/v perchloric acid and 20% v/v Al(NO₃)₃.
Preparation of Standards for Cu, Ni, Pb, Zn, Cr and V

1. Prepare 1000 ppm stock solutions from analar or 99.9% purity metals or stoichiometric salts.
2. From the stock solutions prepare 100 ppm standard solutions in 2N HCl for each element.
3. From the 100 ppm standard solutions, prepare mixed Cu-Ni-Pb-Zn standards with concentrations of 25, 20, 15, 10, 5, 2.5, 1 and 0.5 ppm, separate V standards of 25, 10 and 2.5 ppm and Cr standards of 20, 10, 5, 2.5 and 1 ppm, all in 2N HCl.

Determination of Base Metals

A Varian Techtron AA5 AAS spectrometer was employed. For Cu, Ni, Pb, Zn and Cr an air/acetylene gas mixture was used but in order to increase the sensitivity for V, a nitrous oxide/air/acetylene mixture was necessary.

The instrument was set at the correct wavelength for each element using a concentrated standard to maximize sensitivity. The solutions from step 10 were then sprayed in turn to determine the absorbance; the standard giving the reading closest to the sample was sprayed immediately after the sample for comparison. A H₂ lamp was used to determine non-atomic absorption but this was only found to be significant for Pb.

The amount of metal (M) in the sample is given by the formula:

\[
ppm \, M = \frac{(\text{Abs}_\text{sample} - \text{non-atomic Abs}) \times \text{concentration of standard}}{\frac{\text{Abs}_\text{std}}{\text{mass sample (g)}} \times \text{volume (ml)}}
\]

Determination of Na and K

Na and K were determined with an EEC flame photometer. The instrument was set to give full scale deflection at a convenient concentration (generally 50 or 100 ppm) and to read zero with distilled water. Because the calibration curves for Na and K are not exactly linear, the instrument was calibrated with a series of standards of varying concentrations. The deflections of the sample solutions were then recorded and the concentration calculated using the standard with the nearest deflection. Matrix effects were checked for in all types of sample analysed by mixing equal quantities of a standard solution and the solution being determined. In all cases, the resultant deflection was close to half the sum of the two indicating that matrix effects were negligible.
(ii) **S Determinations**

S was determined with a Leco model 532 automatic S titrator. The sample was mixed with metallic tin and iron and combusted in an induction furnace. SO₂ evolved from the sample during combustion was carried through a stream of oxygen and bubbled through weak HCl solution containing starch, KI and I₂ (which gives a blue colouration). The evolved SO₂ reacted with the I₂ according to the equation: \( \text{SO}_2 + \text{I}_2 + 2\text{H}_2\text{O} \rightarrow \text{H}_2\text{SO}_4 + 2\text{HI} \) which causes a reduction in the intensity of the blue colouration. As this occurred, the Leco automatically titrated KIₐ₃ into the solution to release further I₂ according to the equation: \( \text{KIO}_3 + 5\text{KI} + 6\text{HCl} \rightarrow 6\text{KCl} + 3\text{I}_2 + 3\text{H}_2\text{O} \) until the end point (determined by a photoelectric cell) was reached. The amount of KIO₃ titrated is directly proportional to the amount of SO₂ evolved.

(iii) **CO₂ Determinations**

Total CO₂ was determined by a combustion technique described by Riley (1958).

Carbonate CO₂ was determined by a volumetric method using a Hg manometer. The manometer simply measured the increase in pressure due to CO₂ evolved when about 3 ml of orthophosphoric acid was added to the powdered sample and heated in a closed reaction vessel. The amount of carbonate C is given by the formula:

\[
\% \text{CO}_3 = \frac{v(p-m)}{W(T+t)} \times 0.0961
\]

where
- \( v \) = change in volume of the Hg
- \( p \) = room pressure in mm Hg
- \( m \) = a pressure correction in mm Hg for the temperature of the water used to cool the Hg
- \( W \) = weight of sample in g
- \( T \) = average of room temperature (in °C) before and after determination
- \( t \) = 273.16
Appendix 6
LEAD ISOTOPE TECHNIQUES

Mineral Separation

Rough pyrite separations were made on a micro-sluice. This separate was washed ultrasonically in water, then alcohol. When dry the sample was sieved using a nylon screen held in a plastic holder. The -70+135 size fraction was handpicked under a binocular microscope using a technique developed by R. Rudowski, A.N.U. (pers. comm.). 3 to 100 mg of Py were used for analysis depending on the expected Pb content.

Pyrrhotite was separated using a strong handmagnet and then washed ultrasonically. It was not possible to handpick the pyrrhotite because of its magnetism.

Other minerals were handpicked from massive specimens.

Experimental Procedure for Sulphides

Method of Mrs. M. Aso (pers. comm.). All chemistry carried out in laminar flow clean air laboratory.

1. Weigh sample containing ~1 µg Pb into a 50 ml teflon beaker.
2. For pyrite: wash sample with hot 3N HCl for 10 minutes, then twice with hot 4 x distilled water. For pyrrhotite: wash twice with hot 4 x distilled water to remove surface contamination.
3. Add 3–5 drops Br₂ water and swirl sample into solution.
4. Cover beaker and cautiously add 5 ml of 7N HNO₃, then 2 ml of 6N HCl.
5. Let stand 10 minutes.
6. Swirl sample until float material sinks to bottom and is in a slurry.
7. Add 2 ml of concentrated HF, then 2–3 drops of HClO₄.
8. Let stand about 2 hours cold.
9. Place on hotplate at ~230°C for several hours until sample is in solution or there is just a white precipitate on the bottom of the beaker.
10. Remove cover and dry down at ~150°C.
11. Take up in ~5 ml of 1.2N HBr.
12. Cool and split sample by weight into 2 parts. To 1, add 2 γ 7/4 spike (~0.2 gm of solution), 3 γ 235 spike (~0.3 gm of solution) and 1/3 g of H₂SO₄.

Column Preparation

Use DOWEX 1 x 8, 200–400 mesh for both U and Pb.
For Pb: Wash resin with 3-4 column volumes of 6N HCl. Then put 4 x distilled sub boiling water in reservoir and stir with glass rod to resettle resin. Wash with 3-4 column volumes of 4 x distilled sub boiling water. Prepare resin with 2 column volumes of 1.2N HBr.

For U: Wash resin with 2-3 column volumes of 4 x distilled water. Prepare resin with 2 column volumes of 10N HCl.

Pb Chemistry
1. Put 1N HBr sample on column; collect eluant for U determination.
2. Wash column with 4 x 2½ ml 1.2N HBr (retain 1st washing for U), 1 x 2½ ml 1.2N HCl. Elute Pb with 2 x 2½ ml 6N HCl.
3. Evaporate to dryness.
4. Take up in 1.2N HCl.
5. Wash columns with 2 column volumes 4 x distilled sub boiling water; prepare columns with 2 column volumes of 1.2N HCl.
6. Put 1.2N HCl sample on column. Wash with 1 column volume of 1.2N HCl. Elute Pb with 2 x 2½ ml 6N HCl.
7. Evaporate to dryness; add 1 drop of 7N HNO₃; evaporate to dryness; add another drop of HNO₃ and evaporate to dryness.

U Chemistry
1. Take Pb eluant + 1st 1.2N HBr rinse and evaporate to dryness.
2. Dissolve in 10N HCl.
3. Put on column. Wash with 4 x 5 ml 10N HCl.
4. To reduce and remove Fe: wash with 2 x 5 ml 0.5M (NH₄)₂SO₄ saturated with SO₂, then 2 x 5 ml 10N HCl.
5. Elute U with 10 ml of 0.3N HBr.

Blank
A blank was processed with every set of samples.
Use ~0.15 g Fe solution and 1 γ Pb²⁰⁸ spike.
Treat as for Pb sample but do not split.

Spikes
Pb⁷/₄ spike:  
8/6 = 1.8745
7/6 = 36.995
6/4 = 0.26481
²⁰⁷Pb = 0.04677 μ mole/g
235\textsubscript{U} spike : $\frac{235}{238} \approx 6,000$

$235\textsubscript{U} = 37.90 \mu$ mole/g

208\textsubscript{Pb} spike : $\frac{8}{6} = 458.3$

$\frac{7}{6} = 0.269$

$\frac{6}{4} = 252.0$

$208\textsubscript{Pb} = 0.04773 \mu$ mole/g

**Experimental Procedure for Mercury, Silver and Lead Tellurides**

1. Weigh sample into a 50 ml teflon beaker.
2. Wash twice with hot 4 x distilled water.
3. Dissolve by heating in 3 ml of 7N HNO\textsubscript{3}.
4. Evaporate to dryness on hotplate at -150\textdegree C.
5. Take mixture up in 1.2M HBr.
6. Divide solution into two parts as rapidly as possible to ensure even division of AgBr colloid (if coloradoite is present, the Hg keeps most of the Ag in solution); add spikes as for sulphides.

If no coloradoite present, treat as for calaverite.

**Pb Chemistry**

1. Put 1N HBr sample on column; collect eluant for U determination.
2. Wash column with 4 x 2$\frac{1}{2}$ ml of 1.2N HBr, 1 x 2$\frac{1}{2}$ ml of 1.2N HCl. Elute Pb with 2 x 2$\frac{1}{2}$ ml of 6N HCl.
3. Evaporate to dryness.
4. Take up in 2 ml of 1.2M HBr.
5. Wash columns with 2 column volumes 4 x distilled sub boiling water; prepare columns with 2 column volumes of 1.2N HBr.
6. Put 1.2N HBr sample on column. Wash with 1 x 2$\frac{1}{2}$ ml of 1.2N HBr, 1 x 2$\frac{1}{2}$ ml of 1.2N HCl. Elute Pb with 2 x 2$\frac{1}{2}$ ml H\textsubscript{2}O (Ag remains on column).
7. Evaporate to dryness; add 1 drop of 7N HNO\textsubscript{3}; evaporate to dryness; add another drop of HNO\textsubscript{3} and evaporate to dryness.

**U Chemistry**

As for sulphides.

**Experimental Procedure for Calaverite and Hg-free Tellurides**

1. Weigh sample into a 50 ml teflon beaker.
2. Wash twice with hot 4 x distilled water.
3. Dissolve in 4 ml of 7N HNO\textsubscript{3} and 2 ml of 10N HCl.
4. Evaporate to near dryness.
5. Take salt up in 2 ml of 6N HCl.

**Pb Chemistry**

1. Put 6N solution on column which has been treated with 2 column volumes of 6N HCl.
2. Take eluant to dryness.
3. Take up in 1.2N HBr.
4. Complete processing following steps 1 to 7 of the mercury, silver and lead telluride procedure.

**U Chemistry**

As for sulphides.

**Experimental Procedure for Galena**

Chemistry must be carried out in a separate laboratory from the low level samples.

1. Weigh a small cleavage fragment of galena into a 50 ml teflon beaker.
2. Dissolve in 2 ml of 6N HCl and evaporate to dryness.
3. Redissolve in 2 ml of 7N HNO₃ and evaporate to dryness.
4. Take up in 8 ml of 1.2N HBr and weigh.
5. Take an aliquot of 1-2 drops in a separate beaker, weigh and dilute to 2 ml with 1.2N HBr.

**Pb Chemistry**

1. Put 1N HBr sample on column.
2. Wash with 2 x 2½ ml of 1.2N HBr, 1 x 2½ ml of 1.2N HBr.
3. Elute Pb with 2 x 2½ ml of 6N HCl.
4. Evaporate to dryness; add 1 drop of 7N HNO₃; evaporate to dryness; add another drop of HNO₃ and evaporate to dryness.

**Experimental Procedure for Whole Rocks**

1. Weigh sample into a 50 ml teflon beaker.
2. Add 5 drops of Br-water, 3 ml of 7N HNO₃, 1 ml of 6N HCl and leave to stand for 10 minutes.
3. Add 2 ml of concentrated HF and 2-3 drops of HClO₄.

**Pb Chemistry**

As for sulphides

**U Chemistry**

As for sulphides
Mass Spectrometry

The Pb was mounted as PbI$_2^+$ and determined by the method of Oversby et al. (1970).
### Appendix 7

**SUMMARY OF TELLURIDE OCCURRENCES AT KALGOORLIE**

<table>
<thead>
<tr>
<th>Telluride</th>
<th>Locality</th>
<th>Depth (m)</th>
<th>Method of Identification</th>
<th>Reference</th>
</tr>
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<tbody>
<tr>
<td>Galaverite</td>
<td>Block 45 mine</td>
<td>69</td>
<td>Chem. analysis</td>
<td>Simpson (1912)</td>
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<td>Associated mine</td>
<td>311</td>
<td>X.R.D.</td>
<td>Markham (1960)</td>
</tr>
<tr>
<td></td>
<td>&quot;</td>
<td>?</td>
<td>Chem. analysis</td>
<td>Simpson (1902)</td>
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<td>X.R.D.</td>
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<td>501</td>
<td>Microchem., etching</td>
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<td>Probe analysis</td>
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<td>Oroya mine</td>
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<td>Perseverance mine</td>
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<td>Stillwell (1931)</td>
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Author/s: 
Golding, Lee Yvonne

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