EXPERIMENTAL STUDIES OF
FAULT ZONE DEVELOPMENT IN A POROUS SANDSTONE

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DECLARATION:

I declare that this thesis has been written by myself and is the result of my own research, except where contributions have been stated and duly acknowledged.

Karen Mair.
ABSTRACT

This study investigates the processes involved in the formation and evolution of faulting in high porosity sandstone using laboratory triaxial compression testing. Faults in highly porous sandstone significantly affect the porosity and permeability of the rock, and typically occur as anastamosing compound bands of damage. Previously only the individual unit of these deformation band structures had been re-produced in the laboratory, possibly due to limitations on sample size. Now by deforming large specimens, I have not only produced zones of deformation bands, but also observed their hierarchical development as a function of strain - for the first time. A series of dry tests were carried out on initially intact 100 mm diameter cores of Locharbriggs sandstone, at a constant confining pressure of 34.5 MPa, a constant axial strain rate of 5x10E-6 /s and increasing amounts of axial strain. Samples were driven over their failure curves and then subjected to differing amounts of post failure sliding. A second series of tests were carried out at increasing amounts of confining pressure (in the range of 13.8 MPa to 55.2 MPa) and constant amounts of axial strain. The response of the rock to the loading conditions was continuously monitored throughout the tests by recording the axial load, the axial strain and the volumetric strain. Deformation structures and gouge material produced were studied by hand specimen, thin section and laser particle size analyses. The samples exhibited essentially brittle behaviour with small amounts of strain hardening and softening occurring near peak stress. Dynamic failure was accompanied by a measurable stress drop, which systematically decreased in magnitude as a function of confining pressure. Both compaction and dilatancy are observed in the volumetric strain curves. The amount of dilatancy decreased systematically with increasing confining pressure, until at high confining pressures, compactive behaviour dominated. The ultimate strength of the samples increases systematically as a function of confining pressure. The fault zones generated consist of sets of strands of pale granulated material, separated by lenses of apparently undamaged host rock. Sets of strands were sub-parallel in longitudinal section (parallel to shear direction), but showed anastamosing in perpendicular cross section. The widths of the strands are variable, but the spacing between bands is always greater than the band width itself. Axially oriented microcracks are prevalent in the lenses separating the gouge strands. The individual strands showed a reduction in grain size, porosity and sorting compared to the host rock. The number of strands in a fault zone are found to be linearly correlated to the applied axial strain. Mean grain size, however, reached a steady value irrespective of the applied axial strain. Deformation at low confining pressure led to development of a single shear plane whereas high confining pressure resulted in an isotropic, radially symmetric pattern of damage comprising many gouge strands. All the deformation studied was brittle at a microscopic scale involving microcracking and shear band development but an increase in confining pressure strongly affected the style of structures, how they interacted, and promoted a change towards macroscopically ductile deformation. A study of natural fault zone structures was carried out in the Hopeman sandstone of Lossiemouth, Scotland. Both the fabrics and spatial relations observed in field are strikingly similar to laboratory fault zone characteristics. In particular, a quantitative analysis of individual strands showed the same reductions in grain size, sorting and porosity seen in the laboratory specimens. In summary, the observations presented here are the first realistic evidence of zones of deformation bands to be produced in the laboratory.
dedicated to the memory
of my Dad
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Preface

Introduction
As is the case with most research, the emphasis of this study has changed significantly since its conception in 1993. I see this as a very positive aspect, since it has allowed me to concentrate on the most productive and rewarding paths. On analysis of the first laboratory-deformed specimen, which yielded exciting new results, it was clear that the project should be driven mainly by experimental work, to investigate how and why these features formed, but should also encompass an element of fieldwork. In the next few sections I will explain how time was divided between experimental work, analyses and fieldwork. I go on to indicate where I have previously presented aspects of this thesis, and introduce the first chapter.

Experimental Work Programme
Setbacks invariably plague experimental work especially where new apparatus is concerned. This study was no exception and was interrupted on several occasions by major equipment failure. Rig ‘down time’ was used, where possible, to analyse test samples and process the data collected to date. Although this is not the perfect scenario, valuable quantitative data was collected during these times. Progression was far from linear, but the constant feedback between analyses and laboratory experiments provided opportunities to fill data gaps as they arose, and promoted focus into areas of particular interest e.g. concentrating tests near or at dynamic failure once the values of stress and strain expected at failure had been recognised. Appendix A provides a log of major equipment repairs for reference.

Field Programme
Original field work was carried out in the Muddy Mountains Thrust Zone, South-east Nevada, U.S.A.. This area was found to be unsuitable due to a complex secondary phase of deformation which overprinted much geological information. The difficulties in decomposing the two phases of deformation into an initial thrusting event and subsequent normal re-activation acting along the same orientation of plane were deemed to beyond the scope of this study. Also, the main features of interest - possible evidence for the mobility of fault gouge into the upper plate due to enhanced pore fluid pressures along the fault plane during thrusting - were only observed in areas where convincing evidence for secondary phase fault re-activation was indicated. This implied that the features were more strongly
linked to the secondary deformation phase than the initial thrusting and hence obliterated any evidence for enhanced fluid pressures at the time of thrusting. Although the field area itself proved unsuitable, extremely valuable field techniques (specific to geological areas with 99% outcrop) were rapidly acquired, along with a sobering appreciation for the complexities of natural fault zones.

My introduction to the peculiar features, known as deformation bands, came whilst field assisting Dr Patience Cowie in the Entrada and Navajo sandstones at Arches National Park, Moab, Utah. My interest in these features was fuelled by their occurrence in a rock-type very similar to that of my laboratory studies and therefore during this exercise, I noted their main characteristics and how their apparent interaction with each other. On my subsequent return to Edinburgh I began to draw comparisons with my initial laboratory tests. It was clear that this was a worthwhile task but more field samples were required for a quantitative comparison. As a return to Utah was not possible, a local field area was chosen for its good exposure along a coastal section and proximity to an Edinburgh base. The deformation bands occurring in the Hopeman sandstone of Cummingstown, near Elgin, North-east Scotland, therefore form the basis of the fieldwork presented here.

Time Frame
This Ph.D. has included two separate, three month periods of official suspension due to contract employment. The period from July until October 1995 was spent 'beta-testing' a multi-channel acoustic emission system. Between June and September 1996 I carried out a suite of creep experiments on small plugs of sandstone with brine as a pore fluid. A further one month period of field assisting was carried out in April 1994. Together, these suspensions resulted in a Ph.D. of 39 months duration.

Presentations
Elements of the work presented in this thesis have been previously presented by myself in the following locations: Statoil, Stavanger, Norway 4/97 (invited talk); Department of Earth, Atmospheric and Planetary Sciences, MIT, Cambridge, MA 1/97 (invited talk); Shell International, Rjiswick, Holland 12/96 (invited talk); ‘Faulting, fault seal and fluid flow in hydrocarbon reservoirs’, University of Leeds, Leeds, UK 9/96 (conference talk); Fall Meeting of the American Geophysical Union, San Francisco, CA 12/95 (conference talk); UKGA, Manchester, UK 4/95 (conference talk); EAPG, University of Aberdeen,
3/95 (conference talk). I have also been an active participant of the departmental seminar series at Edinburgh and have therefore presented my work extensively in this environment.

A Users Guide

The subject of experimental rock mechanics sits at the boundary of several different disciplines which can often lead to ambiguities in terminology. A short Glossary (Appendix B) of technical terms and abbreviations is provided. This list is by no means exhaustive and in particular ignores geological terms in common usage. The reader from a ‘non-geological’ background is referred to one of the many geological dictionaries e.g. Whitten and Brooks (1988).

The next chapter introduces the rationale for this study, previous related work, and explains the novel contribution this study makes. At the end of Chapter 1, a thesis plan is presented for the ensuing chapters.
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CHAPTER 1: INTRODUCTION

[1.1] Rationale
The permeability of a faulted rock, and hence its ability to transmit fluids, is strongly
dependent on its existing fault network, and how this develops in response to changing
stresses in the brittle upper crust. The influence of faults on fluid flow through porous rocks
has become an area of intense study due to many practical applications in several areas, e.g.
groundwater flow, hydrothermal fluids migration, leaching and disposal of radioactive
waste, and hydrocarbon production. Of particular interest to this study is the relationship
between faulting, permeability and fluid flow. There are now recognised to be several
distinct mechanisms which can effectively reduce rock permeability during faulting in
sandstones, resulting in a potential for faults to act as a barrier or seal to fluid flow.
Whether an individual fault acts as a conduit, encouraging preferential fluid flow along itself
with respect to the host rock, or as a barrier, preventing or buffering fluid flow, depends on
the micro-mechanisms involved in fault development, how these mechanisms interact and
how far they have evolved.

[1.2] Aims
The aims of this project are to improve understanding of the micro-mechanisms that operate
during the formation and evolution of macroscopic fractures in porous sandstone; the
environment (e.g. confining pressure, loading conditions, strain), that controls fault
development; and the interaction between the environment and the resulting deformation
structure. From this basis, inferences can then be drawn as to how the faults influence the
petrophysical properties of the bulk rock in a range of applications.

[1.3] Methodology
In this thesis I present results from a series of laboratory compression experiments designed
to create faults in previously intact rock and study their mechanical and structural evolution.
Experimental work offers the opportunity of real time monitoring of mechanical parameters
during deformation and a post test structural analysis of samples which can be directly
compared with field studies of fault systems. If similar "end products" are observed in both
the laboratory and field, then laboratory work may be used to infer the evolution stages of
field structures, or to predict their future behaviour in response to further natural or man- 
made changes to the stress field.

In this chapter I provide a brief introduction to rock and fault mechanics and the 
current state of rock deformation experiments. In addition to the external forces, the 
intrinsic properties of the rock, e.g. porosity and permeability, are crucial factors in 
determining how faults develop. Also, as seen in section [1.1], the development of faults 
themselves significantly affect the porosity and permeability of rocks. The following 
sections discuss these aspects, providing a background to this study, the basis for the work 
detailed in this thesis, and the project objectives.

[1.4] Background - Fault Mechanics

[1.4.1] Stress and strain

Tectonic and gravitational forces in the Earth which act upon rocks, will tend to displace 
and deform an element of rock. How the rock responds to these forces depends on the 
magnitude and loading rate of the forces applied, the temperature and chemical 
environment, the absence or presence of pore fluids, and of course the intrinsic properties of 
the rock itself. In order to quantify the magnitude and orientation of forces applied and the 
response of the rock, the concept of stress and strain is introduced. The stress state acting 
on a body is generally represented by forces acting on the faces of an elemental cube (see 
Figure: 1.1) where stress is defined as the force per unit area on any one of the cube faces. 
Normal stresses act perpendicular to the faces, whereas shear stresses act parallel to the cube 
faces. The stresses can be defined explicitly as a nine component stress tensor (equation: 
1.1)

$$\sigma = \begin{bmatrix} \sigma_{11} & \sigma_{12} & \sigma_{13} \\ \sigma_{21} & \sigma_{22} & \sigma_{23} \\ \sigma_{31} & \sigma_{32} & \sigma_{33} \end{bmatrix}$$ equation: 1.1

in which the diagonal terms represent normal components of stress, and the off diagonal 
components represent shear components of stress (see Figure: 1.1). The forces capable of 
exerting torques must cancel out, hence the matrix is symmetric $\sigma_{12} = \sigma_{21}$, $\sigma_{13} = \sigma_{31}$, $\sigma_{23} = \sigma_{32}$ 
so the tensor has six degrees of freedom. Further simplification can occur if a matrix 
rotation is used to reduce the stress tensor to the diagonal form with trace $\sigma_1$, $\sigma_2$, $\sigma_3$ 
representing the principal stresses. Conventionally, the normal components of stress are 
referred to as the maximum ($\sigma_1$), intermediate ($\sigma_2$) and minimum ($\sigma_3$) principal stresses
Chapter 1 - Introduction

(Anderson, 1954). The differential stress is defined as the difference between the maximum and minimum stresses (i.e. $\sigma_1 - \sigma_3$).

Strain represents the visible response of a rock to the applied load and is normally expressed as a relative or dimensionless change in the size and shape of the rock. The strain tensor takes a similar form to the stress tensor (equation: 1.2)

\[
\varepsilon = \begin{bmatrix}
\varepsilon_{11} & \varepsilon_{12} & \varepsilon_{13} \\
\varepsilon_{21} & \varepsilon_{22} & \varepsilon_{23} \\
\varepsilon_{31} & \varepsilon_{32} & \varepsilon_{33}
\end{bmatrix}
\]

and strain is defined as the change in volume (or length) divided by initial volume (length). The relationship between the stress and strain depends on the intrinsic properties of the rock. If the material is elastic and isotropic, stress and strain can be related by the equation:

\[
\sigma = E \cdot \varepsilon
\]

known as Hooke's law, where $E$ is Young's modulus of elasticity. In this case, the response to applied stress is instantaneous and any deformation occurring in the rock is recoverable when the load is removed. The deformation of a material is normally illustrated in the form of a plot showing axial stress as a function of axial strain (e.g. figure: 1.2). The main mechanisms of failure are defined in the following section in terms of these stress-strain curves.

[1.4.2] Macroscopic failure and microscopic processes

Materials subjected to a differential stress ($\sigma_1 - \sigma_3$) can undergo failure by two mechanisms: (i) Discrete deformation (or fracture) along a well defined fracture surface, or (ii) irrecoverable continuous deformation (plastic flow) where the material fails without loss of cohesion and continuity. Fracture is defined as being brittle when it is preceded by little or no permanent damage, however it is common for a small amount of inelastic strain to occur before failure (Paterson, 1978), and a material is often described as ductile if it fails by plastic flow (Ranalli, 1995). In ideal brittle behaviour, stress goes to zero at brittle yield (Figure: 1.2a). Ideal plastic behaviour results in stress remaining constant after attaining the material's yield stress, although, experimentally, stress can also either increase (strain hardening) or decrease (strain softening) with respect to increasing strain after the yield point. These three types of plastic behaviour are illustrated in Figure: 1.2b.

An increase in depth, and hence an increase in confining pressure and temperature, promote a transition from macroscopic brittle behaviour to macroscopic ductile behaviour.
Figure: 1.1 Diagram showing the stresses acting on the faces of an elemental cube (after Hobbs et al., 1976).

Figure: 1.2 Stress-strain curves for a) ideal brittle failure; and b) ductile failure (after Ranalli 1995). The curve in b) shows ideal plastic behaviour and hence constant stress (straight line); an increase in stress i.e. strain hardening (upward curve); and a decrease in stress i.e. strain softening (downward curve).
Figure: 1.3 illustrates schematically the macroscopic appearance of samples as they pass from the brittle to ductile failure regimes. The main attributes and variables associated with the brittle, semi-brittle (transitional) and fully plastic fields are also summarised in Figure: 1.3, and the figure highlights that the ductile field encompasses both the semi-brittle and plastic regimes. Brittle behaviour is often characterised by dilatancy (see [1.4.6]) and shear localisation which usually involve cataclastic mechanisms including grain crushing and grinding. Ductile behaviour is generally homogeneously distributed throughout the material volume as opposed to being localised on a plane. The micro-mechanisms responsible for macroscopic observations associated with brittle or ductile behaviour depend on the initial porosity and composition of the rock. Low porosity quartzo-feldspathic rocks may fail by crystal plasticity processes whereas highly porous silicate materials are much more likely to fail by local grain scale brittle cracking. This grain scale brittle behaviour, when distributed throughout the material volume, is referred to as cataclastic ‘flow’ illustrating the concept that macroscopic ‘ductility’ can be achieved by distributed cataclastic (microscopically brittle) mechanisms. Moreover, macroscopic brittle and ductile regimes, although resulting in very different deformation, often involve very similar micro-mechanisms. Individual micro-mechanical processes associated with brittle and ductile failure are described in more detail in sections [1.4.8] and [1.4.9]. This study concentrates on the deformation of porous sandstones under upper crustal conditions, where macroscopically brittle failure mechanisms are most likely.

[1.4.3] Coulomb-Navier macroscopic failure criterion

A failure criterion is an empirical construction which can be used to predict the macroscopic conditions at which a rock is most likely to fail without regard to the microscopic processes involved. Conversely, the Griffith theory, discussed in Paterson (1978), is based on the microscopic physical processes involved in the failure of an ideal homogeneous continuum, but it does not, in general, predict the macroscopic strength of heterogeneous composite materials such as rocks. Mohr’s macroscopic criterion for shear fracture states that: “fracture occurs across the plane (or planes) where the shear stress first reaches a value that depends on material parameters and normal stress on that plane”. The Mohr circle is a geometric construction to illustrate the stress state within a rock (Figure: 1.4). The relationship between shear stress and normal stress is determined by experimental studies which define the locus of stress conditions at the point of failure (Figure: 1.5). Sample failure will occur when the Mohr circle, characterising the stresses within a rock, intersects
Phenomenology of the Brittle Ductile Transition
of Low Porosity Crystalline Rocks under Compressive Loading

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<tr>
<th>Failure Mode</th>
<th>&lt; -------------- Brittle-------------&gt;</th>
<th>&lt; -------------- Semibrittle-------------&gt;</th>
<th>&lt; -------------- Ductile-------------&gt;</th>
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<tr>
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<td>&lt; ---------------------------------- Cataclasis---------------------------------- &gt;</td>
<td>&lt; ---------------------------------- Plastic---------------------------------- &gt;</td>
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| Permanent Strain before Failure | < <3% > < 3% > < 5% > |
| Work Softening               | < > |
| Possible Stress Drop          | < > |
| Loss in Cohesion              | < > |
| Microcracking                 | | |
| Dilatancy                     | | |
| Acoustic Emission             | | |
| Pressure Dependence of Strength | | |
| Temperature Dependence of Strength | | |
| Deformation Mechanisms        | Distributed and Localized Microcracking | Distributed Microcracking, Local Plasticity | Fully Plastic |

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<th>Macroscopic Appearance</th>
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<th>Typical Axial Stress-Strain Curves</th>
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Figure: 1.3 Schematic diagram illustrating the phenomenology of the brittle-ductile transition (after Evans et al. 1990).
Figure: 1.4 Illustration of the Mohr circle construction (after Scholz, 1990) and the Coulomb fracture criterion showing the relation between parameters at failure. Graph shows shear stress as a function of normal stress. The angular relationship between the fracture planes and the principal stresses is shown on the right.

Figure: 1.5 Coulomb-Navier criterion for shear fracture (after Ranalli, 1995) showing equation of the linear envelope.
the failure envelope. Experimental studies reveal a failure curve which is close to linear, hence the straight envelope on Figure: 1.5 as defined by equation: 1.4

$$\tau = S + \mu \sigma$$  

equation: 1.4

where $\tau$ is the shear stress, $S$ is the cohesive strength of the material and is equivalent to the shear stress required for fracture in the absence of a normal stress, $\mu$ is the coefficient of internal friction which is an empirical measure of the increase in shear stress brought about by an increase in normal stress (in general $\mu = 0.5-0.8$), and $\sigma$ is the normal stress. Equation: 1.4 is generally referred to as the Coulomb-Navier Failure Criterion. Although this construction has limitations (e.g. it ignores the effects of the intermediate principal stress which is thought to influence failure - see Crawford et al., 1995), it gives a reasonable empirical account of failure as determined from laboratory and field studies.

Pore fluids influence the stresses required to deform a rock, and this is defined by the effective stress. The law of effective stress describes the physical behaviour of a fluid saturated porous medium (equation: 1.5)

$$\sigma_{ij}^e = \sigma_{ij} - P \delta_{ij}$$  

where $\sigma_{ij}^e$ is the effective stress, $\sigma_{ij}$ is the applied stress, $P$ is the pore fluid pressure, $\delta_{ij}$ is the Kronecker delta (see Jaeger and Cook, 1979). Pore fluid pressures tend to work against the applied stress, hence reducing it.

A sample can be made to approach failure in the following ways illustrated in Figure: 1.6: (a) By increasing the differential stress which increases the size of the Mohr circle; (b) Increasing pore fluid pressure (equivalent to a reduction in effective normal stress); this maintains the same size of Mohr circle, but displaces it to the left towards the failure envelope; (c) A reduction in the friction coefficient; this shallows the slope of the envelope; or (d) A reduction in sample cohesion, which maintains the same slope but moves the envelope down towards the x-axis. In contrast, a system will move away from failure under the opposite conditions (arrows reversed in Figure: 1.6): e.g. (a) A reduction in differential stress reduces the size of the Mohr circle; (b) A reduction in pore fluid pressure moves the Mohr circle to the right; (c) An increase in internal friction increases the slope of failure envelope; and (d) An increase in cohesion moves the failure envelope upwards.

[1.4.4] Experimental rock deformation

Previous rock deformation experiments representing a variety of stress conditions have been used to study rock mechanics problems e.g. uniaxial, hydrostatic, biaxial, triaxial,
Figure 1.6 Schematic diagram using the Coulomb-Navier failure criterion and Mohr circle to illustrate the main ways in which a sample can approach failure: a) increase $\sigma_1 - \sigma_3$; b) reduce effective stress; c) reduce friction; d) reduce cohesion.
polyaxial, 'true' triaxial. Each have strengths and weaknesses in simulating crustal conditions whilst introducing as few experimental artefacts as possible. A full account of each of these methods is given in Jaeger and Cook (1979). Perhaps the most popular procedure used to impose a triaxial stress state such that all principal stresses are non-zero (see Figure: 1.7) is referred to as the triaxial test. Triaxial tests are the most geologically significant methods of experimental rock deformation as they effectively simulate both overburden, and the normal and shear stresses, which are known to significantly influence sample behaviour (see Paterson, 1978). The simplest form of this test involving a differential stress ($\sigma_1 - \sigma_3$), requires the procedure of applying a uniaxial stress to a sample already under hydrostatic confinement (see figure: 1.7). In this arrangement, the intermediate principal stress ($\sigma_2$) must equal one of the other principal stresses, usually the minimum stress ($\sigma_3$). The confining pressure, which is hydrostatic, is applied by means of a hydraulic medium (usually oil or an inert gas) surrounding the sample under pressure. The axial load is applied to the sample ends, typically using a ram.

[1.4.5] Synoptic stress-strain curve for brittle failure

During experimental rock deformation the stresses applied to a rock and the resulting deformation can be monitored prior to, during, and after failure. This yields important information regarding the evolution of failure which is generally presented graphically as a plot of axial stress versus axial strain (shortening). Figure: 1.8 is a synoptic stress-strain curve for a compressive test, illustrating the main stages of loading (phases I-VI on the diagram) which have been determined from previous experimental work (modified after Paterson, 1978).

Phase I, which is most evident in uniaxial tests but also apparent in some triaxial tests, is the upwardly concave initial phase of the stress strain curve (Figure: 1.8). This phase is generally attributed to the closing of pre-existing cracks under stress and may be totally reversible. In porous rocks, this phase may also involve irreversible pore collapse which will continue into the next phase. In phase II, elastic deformation dominates and the sample responds to load in a linear way, determined by its intrinsic material properties, i.e. elastic constants. Phase III marks a departure from linear elasticity, and involves non-linear strain hardening up to peak stress. This stage typically commences at approximately 2/3 of peak stress. The departure from elasticity is usually ascribed to the growth and development of microcracks distributed fairly randomly throughout the sample. One important characteristic of this phase is that cracks propagate in a stable way, i.e. they have a limited
Figure: 1.7 Diagram showing the geometry of the triaxial deformation test on cylindrical specimens. Diagram shows the application of firstly a hydrostatic pressure (top) where $\sigma_1 = \sigma_2 = \sigma_3$, then a superimposed uniaxial stress (bottom) resulting in a differential stress ($\sigma_1 - \sigma_3$) where $\sigma_1 > \sigma_2 = \sigma_3$. 
growth and do not run on unstably to form macroscopic fractures. Phase IV is a short phase of quasi-static strain softening deformation, which occurs immediately prior to dynamic sample failure, and is thought to correspond to a localisation of microcracking (e.g. Scholz 1990). At this point the deformation becomes inhomogeneous and the stress and strain values are only meaningful as averages as the rock specimen itself is also non-uniform. Phase V marks the onset of dynamic shear localisation which normally corresponds to the appearance of macroscopic shear fracture. An abrupt stress drop back to the frictional level is identified at dynamic failure, corresponding to a loss of both cohesion and the ability to bear load. The main points punctuating the stress-strain curve in Figure: 1.8 are defined, as follows: a) end of crack closure phase; b) yield stress; c) peak stress; d) onset of dynamic failure; e) residual stress; f) final stress. All these phases in stress-strain evolution have been identified on one or all of the rock deformation tests reported in this thesis.

[1.4.6] Synoptic volumetric strain curve for brittle failure

In addition to monitoring axial strain, volumetric strain is often monitored during deformation tests. A measurement of volumetric strain is useful because of the tendency of the sample to compact or dilate, and hence changes in porosity can be inferred. Figure: 1.9 is a synoptic graph of stress difference as a function of volumetric strain for a typical laboratory test on a low porosity crystalline material (redrawn after Brace et al., 1966). The term dilatancy is used here (following Brace et al. 1966) to describe an increase in volume relative to the behaviour (linear compaction) anticipated for a linear elastic material (shown by the dotted line in Figure: 1.9). Phase I shows a concave tail indicating a greater volume compaction than would be expected for an elastic material. This will only be discernible if the rock initially consists of cracks which can close under strain. This initial volume compaction may be missing under triaxial compression tests due to earlier closure of cracks under the period of hydrostatic loading prior to the application of a differential stress (see Figure: 1.7). Throughout phase II, the rock exhibits linear elastic compaction as a function of loading, corresponding to phase II in the stress-strain curve (Figure: 1.8). Phase III, in figure: 1.9, marks a departure from linear behaviour where the rock is observed to increase in volume i.e. dilate with respect to the behaviour expected for an elastic material. This is generally attributed to either the opening of microcracks with axes parallel to maximum compressive stress (Brace et al., 1966), but in the case of porous granular rocks, may also be due to grains riding over each other (see section [1.4.8]). Phase IV signifies macroscopic fracture, which may be associated with an abrupt dilation corresponding to the point at
Figure: 1.8 Synoptic stress-strain curve for a compressive laboratory test (modified after Paterson 1978) illustrating the main stages of loading (phases I-VI explained in main text section [1.4.5]) and the significant points punctuating the curve: a) end of crack closure; b) yield stress; c) peak stress; d) onset of dynamic failure; e) residual stress; f) final stress.

Figure: 1.9 Synoptic graph of volumetric strain as a function of stress difference (redrawn after Brace et al. 1966) showing the main characteristics stages of volumetric response (Phase I-V explained in main text section [1.4.6]). A dotted line shows the predicted volumetric response of a linear elastic material.
which the rough surfaces of the fault are first separated - due to a sudden loss of cohesion on
the point of failure. Following failure, in phase V, the volume of the rock may stay
essentially constant (if the sample slides frictionally on a single surface), or may undergo
dilatancy or compaction depending on a range of factors including the porosity, grain size
and effective pressure (Wong et al., 1997).

[1.4.7] The influence of confining pressure: the brittle-ductile transition
Laboratory deformation of sandstones and quartzites (e.g. Hirth and Tullis, 1989; Wong,
1990) has shown that an increase in confining pressure (simulating increased burial depth)
or an increase in porosity promotes a change in the style of deformation from the brittle to
macroscopically ductile regime. Although the individual micro-mechanical processes are
dominantly brittle grain cracking in both regimes, deformation changes from localisation
along a single plane in the brittle regime to distributed pervasive damage throughout the
sample in the ductile regime.

Perhaps the most obvious characteristic distinguishing brittle from ductile
deformation modes is the magnitude of the stress drop at failure. Brittle materials have a
significant and often audible stress drop (e.g. Figure: 3.1 this study), whereas the stress drop
in ductile materials is observed to decrease in magnitude with increasing confining pressure
(e.g. Figure: 3.7 this study). One of the main criteria for defining the transition into the
ductile field is an absence of a recognisable stress peak and the lack of a significant stress
drop.

The volumetric strain prior to failure in the brittle and ductile fields can show very
similar trends to each other. However, the behaviour at, and beyond failure commonly
varies significantly in the two deformation fields. Samples deforming in a brittle regime
undergo initial compaction which is generally followed by dilatancy. In contrast, the extent
of dilatancy following failure gradually decreases with increased confining pressure until
compaction is the dominant mechanism throughout in the ductile regime. Micro-mechanical
models offering an explanation for dilatancy and compaction, based on laboratory
microstructural analysis, are briefly presented in the next section.

[1.4.8] Shear-induced dilatancy
*Shear-induced dilatancy* (Menendez et al., 1996) is a mechanism that potentially explains
the dilatation observed just prior to failure in the brittle regime as presented in Figure: 1.10.
The stages of evolution are as follows: a) Shear stress increases and the rock reacts
Figure: 1.10 Schematic diagram showing the evolution stages of shear-enhanced dilatancy (see main text section [1.4.8] for details).
elastically; b) At a critical stress, grain-grain contacts begin to rupture; c) Grains are then free to move relative to each other and ride over each other, causing an overall increase in volume i.e. *dilation*. This process is sometimes referred to as *particulate flow*; d) Tensile stresses are then concentrated on fewer points and hence increase locally; e) An increase in tensile stresses results in the tensile fracture of grains. The fractured grains associated with this tensile fracture then have a reduced grain size, and are further *granulated*, under further stress, to produce a gouge layer. Frictional sliding localises along this gouge layer, and the volume of the gouge layer remains essentially constant. Small increases and decreases in volume occur due to the roughness of fracture surfaces. The gouge layer forms a shear band and continued slip along this shear band induces tensile stresses which lead to fracturing within, and at the walls, of the band.

1.4.9 Shear-enhanced compaction

At higher confining pressures, samples under triaxial compression are observed to undergo compaction throughout loading (e.g. Figure: 3.19e this study). This is referred to as *shear-enhanced compaction* (e.g. Curran and Carroll, 1979; Wong et al., 1992). From observations of the microstructures produced by a series of laboratory tests at a range of pressure conditions, Menendez et al. (1996) suggested the following micro-mechanical model to explain the observations (see Figure: 1.11): a) At high confining pressure, relative movement between grains (as proposed in [1.4.8] above) is inhibited as they are unable to roll over each other due to lateral constraints; b) In order to accommodate the increasing shear strain, pore space collapses instead, and at this point the contacts between grains may rupture. Pore space collapse results in strain hardening, thereby inhibiting shear localisation; c) Any further shear movement must then be accommodated through the fracture of grains (in a clay-rich material this stage would not occur as the grains would simply roll and slip over each other due to the lower friction coefficient of the clay minerals); d) Further strain leads to pervasive grain crushing resulting in grain size reduction and a broader range of grain sizes; e) The range of grain sizes pack together more efficiently than a more uniform size distribution, hence reducing sample volume. In this model, a *critical pressure* is required to promote a change from shear localisation to cataclastic flow. This critical pressure reduces systematically with both increased porosity and a larger grain size, implying that highly porous, coarse-grained materials are more likely to undergo cataclastic flow (Wong et al., 1997) than brittle failure. Although both models (described here and in [1.4.8]) describe the microfracture of grains under shear stress as a mechanism of grain size
Figure: 1.11 Schematic diagram illustrating the stages of evolution during shear-enhanced compaction (see main text section [1.4.9] for details).
reduction, the manner by which microfracture occurs is significantly different (i.e. dilatant or compactive) with a difference in confining pressure.

[1.4.10] Elastic crack model
The mechanisms described above accept the inhomogeneous, granular nature of rocks. However, previous attempts to derive failure criteria have assumed that rocks can be treated as an effective homogeneous continuum (e.g. Griffith criterion). A good review of elastic crack theory is presented in Pollard and Segall (1989). This continuum elastic crack model predicts how the growth of a single crack will influence its surroundings when subjected to a uniform elastic load. An increase in the maximum shear stress promotes a tendency for shear deformation. By quantifying the amount of deformation, the magnitude of the maximum shear stress on and around a shear crack (in either mode II or mode III displacement modes, see Figure: 1.12) can be calculated (e.g. Pollard and Segall, 1989; Kostrov and Das, 1982). Figure: 1.13a shows the geometry of the problem, and Figure: 1.13b shows an example of this continuum approach in the form of a contour plot of shear stress normalised to stress drop along the pure shear (mode II) crack after Das and Scholz (1981). The shaded areas indicate an increase in stress due to the crack development and negative contours indicate areas of reduced stress both along the crack itself and in the adjacent area. This plot illustrates a relatively simple stress field pattern, with an expected increase in two lobes at the crack tips but also a significant increase in shear stress at a perpendicular distance of approximately half a crack length from the crack (see Figure: 1.13c). Stress is decreased immediately adjacent to the crack. This implies a potential for shear deformation a finite distance away from the initial crack, albeit diminishing with distance.

[1.5] Background - Structural and hydraulic characteristics of faults
[1.5.1] Introduction
So far, I have concentrated on the mechanical basis for examining rock deformation and faulting. In this section, I summarise the type of information that can be extracted from the structural examination of natural and laboratory faults. I briefly consider the structural influence on the hydraulic properties of a rock which is subsequently discussed in further detail in section [1.6].
Figure 1.12 Sketch illustrating the three fundamental modes of fracture (after Guéguen et al. 1990). I, opening mode or tensile; II, sliding mode or in-plane shear; III, tearing mode or anti-plane shear.

Figure 1.13 a) see caption overleaf.
Figure: 1.13b)

Figure: 1.13 Diagram showing the stress changes resulting from a shear rupture (after Das and Scholz, 1981). a) Geometry of the problem indicating the crack and co-ordinates; b) shear stress changes brought about by the crack (according to the elastic crack model) as a percentage of stress drop on the crack. Shaded area indicates regions of enhanced stress; c) normalised shear stress as a function perpendicular distance from the crack. Arrows show position of maximum shear stress. Shear stress at the site of the crack is clearly reduced.
Sealing faults

As indicated in the opening section of this chapter, faults in sedimentary rocks are now recognised to have ‘sealing potential’, i.e. the ability to form barriers or buffers to fluid migration. The three most well established types of sealing faults are juxtaposition, clay-smear, and sand-sand seals (see e.g. North, 1985). Juxtaposition seals occur in interbedded sedimentary sequences when faulting offsets a porous unit, e.g. sand, bringing it in direct contact with a unit of lower permeability, e.g. shale (Figure: 1.14). The lower permeability unit causes a physical barrier or buffer to fluid flow. Clay smear seals are found in sandstones which contain a small amount of clay in the matrix or as interbeds. When faulting occurs, the weak clays are injected from the host rock into the fault plane. As well as aiding localisation of faulting onto a narrow discrete surface, this surface is also a buffer to fluid flow due to the small particle size and water-retaining properties of the clay (see figure: 1.15). More recently it has been recognised that sealing most commonly occurs across sand-sand fault interfaces in very clean sands with no clays present. Two thirds of fault sealing is thought to occur across sand-sand interfaces (Knipe 1996, pers. comm.) The granulation of sand grains forms a rock powder or gouge (see Figures: 1.16 and 1.11), which reduces the pore volume and hence the permeability. Some field examples of such faults, involving sand-sand contacts in highly porous quartz rich sandstone are presented in the next section.

Observations of natural examples of faults in porous sandstone

Faults occurring in highly porous sandstone units have been studied in several field areas. They have been variously referred to in the literature as "deformation bands" (Aydin 1978, Aydin and Johnson 1978), "microfaults" (Jamison and Stearns 1982), "granulation seams" (Pittman 1981), and simply "faults" (Underhill and Woodcock 1987). To avoid confusion, the terminology of Aydin and co-workers, described below, will be adopted throughout this thesis.

The above studies showed that faulting in highly porous quartz-rich sands typically occurs as complex zones consisting of a set of distinct pale bands forming an anastamosing web of damage, separated by lenses of apparently undamaged host rock (e.g. Figure: 1.17a). In the plane parallel to the direction of shear, these same deformation bands have approximately straight traces which lie sub-parallel to the shearing direction (Figure: 1.17b). These zones are often more resistant to weathering than the host rock, and hence stand proud of their surroundings allowing easy identification in the field. The individual
Figure: 1.14 Diagram of a juxtaposition seal. A shale/sand sequence is faulted such that the shale and sand horizons become juxtaposed. The shale forms a barrier seal against the more permeable sand.

Figure: 1.15 Diagram of a clay smear seal. Clays occur either as narrow beds between the sand layers or as a percentage constituent in the sand. When faulting occurs, the more mobile clays are pushed into the fault zone where they become concentrated. Again this forms a physical low permeability barrier to fluid flow.

Figure: 1.16 Diagram of a sand-sand seal. Where no clays are present, faulting causes granulation along a sand-sand interface. Grain breakage and subsequent size reduction results in a dense well packed layer of reduced permeability.
Figure: 1.17 Photographs of a naturally occurring deformation bands showing a) anastamosing of pale white strands; b) distinct bands lying sub-parallel to the shearing direction.
deformation bands are zones of intense cataclasis, a few mm wide which accommodate very small amounts of offset, and are composed entirely of the host rock mineralogy. They are sites of concentrated damage where grain size, the degree of sorting, and porosity are all reduced with respect to host rock. Aydin and Johnson (1983) categorised the features in terms of their displacement measured from bedding offset: (i) single deformation bands which have mm-scale offset; (ii) zones of deformation bands which occur as a series of distinct bands accumulating a total of cm to dm scale offsets; (iii) slip surfaces with tens of meters offset, localised onto a single plane.

The individual deformation band units are observed to occur both as isolated bands and in groups. Slip surfaces, in contrast, never occur as isolated features but are always closely associated with zones of deformation bands (Aydin 1978, Aydin and Johnson 1978). From these observations of the spatial relations in the field, Aydin and Johnson (1983) proposed that deformation bands develop sequentially as shown in Figure: 1.18: a) The first band to form causes a very small offset of a horizontal bed; b) Two bands will cause a greater offset; c) Several bands (or a zone of bands) cause significant offset; d) and finally, deformation localises onto a slip surface which accommodates much greater displacement. This model appears to be consistent with the majority of other field observations (e.g. Jamison and Stearns, 1983; Underhill and Woodcock, 1987; Antonellini et al., 1994). However, the hierarchical development of deformation bands associated with offset has not previously been observed under laboratory conditions. Accordingly, one of the prime aims of this thesis is to test this hypothesis in the laboratory.

Underhill and Woodcock (1987) highlighted some of the microstructural characteristics of deformation bands, such as grain size reduction, an increase in angularity of grains, and pore space reduction by thin-section analysis. These observations are all consistent with cataclastic granulation which is often assumed to be the sole mechanism for deformation band formation. However, Antonellini et al., (1994) showed that features often described in the field as deformation bands were not exclusively cataclastic granulation strands, but could take the form of either dilatant zones of enhanced porosity (see Figure: 1.10 terminated at stage c), compaction bands of reduced porosity, concentrated clay smears, or some combination of these. They sub-divided so-called deformation bands into categories, depending on their structural characteristics and implied that these characteristics may depend on the initial composition and porosity of the host rock. This study focuses on the cataclastic deformation bands, but it is important to be aware that other types of deformation bands do occur.
Figure: 1.18 Series of block diagrams showing the sequential development of deformation bands from a single band to a slip surface. a) single deformation band; b) two deformation bands; c) zone of deformation bands; d) slip surface (Aydin and Johnson 1983).
Previous observations of laboratory-induced faults in porous sandstone

Previous laboratory work on porous sandstones has succeeded in producing either single or conjugate sets of shear fractures, characterised by a reduced grain-size, reduced sorting, and adjacent zones of microfracturing (e.g. Dunn et al., 1973). Engelder (1974) observed the generation of layers of gouge by the cataclasis of previously intact sandstone, again in single shear zones. Friedman and Logan (1973) produced multiple parallel compaction bands referred to as Luders bands (see Figure: 1.19). At first glance these bands, oriented parallel to shear fractures, and associated with some microfracturing, appear to resemble deformation bands. On closer inspection, however, they show little or no grain size reduction, hence they do not possess the main characteristic of cataclastic deformation bands, and generally appear as surface features only (although on occasion they do traverse the sample).

The influence of confining pressure and hence burial depth on deformation structures as determined from laboratory tests (e.g. Hirth and Tullis, 1989; Wong, 1990) has been introduced in section [1.4.7]. In summary, at low confining pressures, deformation is localised along a single plane, whereas at high confining pressures, deformation is distributed and pervasive throughout the sample. At intermediate confining pressures transitional behaviour between these two extremes is observed, and this observation may offer an explanation for the complex style of many sets of deformation bands observed in the field (as suggested by Antonellini et al., 1994).

Hirth and Tullis (1989) observed in their experiments on quartzite (see section [1.4.7]) that cataclastic flow was a transient strain hardening phenomenon which only lasted as long as pores were collapsing. However, such a transient would occur preferentially in initially highly-porous rocks at higher confining pressures. At high strains, Hirth and Tullis (1989) found shear localisation replaced cataclastic flow as the dominant deformation style. If deformation bands are regarded as a form of transient cataclastic flow which is replaced by shear localisation at high strains, this laboratory observation may offer one explanation to the switch from the formation of zones of deformation bands (Figure: 1.18c) to deformation localised on a slip surface (Figure: 1.18d).

Conceptual models for the development of deformation bands

Various conceptual models have been suggested to explain the evolution of deformation band structures from field observations of the end-products. The main theories for the formation of single bands, zones of bands, and slip surfaces are introduced below.
The initiation of a single deformation band can be explained by two generally accepted models (e.g. Aydin and Johnson 1983): (i) reduction of pore space due to compaction leads to an increase in the number of contact points and contact areas, hence increasing the friction of the mass. The grains interlock strongly and further deformation requires grain fracture; (ii) shear motion tends to dilate the rock mass leading to stress being concentrated at fewer contact points i.e. local high stress concentrations which induce the fracture of grains and hence grain size reduction. In both cases, as the grains become smaller, more angular and better packed, the number of contact points within the band increase and friction therefore increases.

The existence of closely-spaced multiple bands in essentially the same orientation, with each band accommodating only a small displacement, leads to the idea that each band undergoes a stage of strain hardening, possibly caused by the increase in grain contact friction (Aydin and Johnson 1983, Jamison and Stearns 1982, Underhill and Woodcock 1987, Fowles and Burley 1994). Strain hardening on the band makes it energetically more favourable for subsequent deformation to localise on the neighbouring virgin host rock than continue to slip along a pre-existing, but now stronger, fault. This process continues until a zone of discrete deformation bands develops and the bulk deformation is accommodated across a distributed zone made up of a set of extremely localised parallel bands (e.g. Figure: 1.18c).

Slip surfaces have straight traces, striations and grooves, and accommodate large amounts of displacement indicating localised strain along a single plane. All of these characteristics are fundamentally different from those occurring in deformation bands, and imply a strain softening mechanism of formation (Aydin 1978, Jamison and Stearns 1982, Underhill and Woodcock 1987) similar to that inferred for most brittle faults. Therefore it appears that the active deformation mechanisms for formation of single bands (i.e. strain hardening) are different from those invoked for the development of slip surfaces (i.e. strain softening). In an attempt to explain the switch in deformation mechanisms, Aydin and Johnson (1983), following the theories of Rudnicki and Rice (1975), treated zones of deformation bands as a relatively stiff inclusion in a soft matrix, the deformation bands being stiffer due to grain crushing and consolidation. (This geometry is the opposite to a Griffith crack where the inclusion is softer). When an instability develops (due to local stress in the inclusion exceeding the applied stress experienced in the matrix), a runaway instability can occur making strain increments very large. This mechanism would explain the development of slip surfaces in or near to the ‘relatively stiff inclusion’ of deformation
Figure: 1.19 Photograph of a) multiple parallel superficial bands formed in Solenhofen limestone under triaxial compression (Friedman and Logan, 1973); b) a cut sample showing that the bands occur only on the surface of the sample.

Figure: 1.20 a) Map of aftershocks (crosses) of 9 April 1968 Borrego Mountain Earthquake for 3 months following the main shock. b) vertical cross section. (from Das and Scholz, 1981; after Hamilton, 1972).
bands. Fowles and Burley (1994) offer an alternative explanation, proposing that the lack of cohesion (reduction of shear strength) on the slip surfaces may be induced instead by an increase in strain rate as observed in the experimental work of Morrow and Byerlee (1989). This however begs the question of what causes the increased strain rate in the first place.

Whatever the mechanisms responsible for the development of these peculiar features, it is clear that each new deformation episode is strongly influenced by the history of the rock, implying that the mechanical characteristics of a rock are strongly controlled by the total amount of strain and the type of damage it has sustained during deformation (Jamison and Stearns 1982).

[1.5.6] Off-fault deformation and aftershocks

In a similar manner to how dynamic failure in rock deformation experiments produces faults associated with a significant and often audible stress drop, dynamic shear fracture on a geological scale results in earthquakes and, in a heterogeneous Earth, aftershocks. Studies of the spatial distribution of seismicity following an earthquake reveal that aftershocks do not necessarily occur on the main fault, and that long range 'triggering' can also occur. In several cases (cited by Das and Scholz, 1981) off-fault aftershocks related to strike-slip earthquakes have locations that are parallel to the main fault but are isolated a finite perpendicular distance (see Figure: 1.20). Das and Scholz (1981) recognised that these off-fault aftershocks cluster in certain locations where an elastic crack model for pure shear (see section [1.4.10]) predicts increased shear stress in far-field location due to the initial fault. The spatial correlation between observed slip and predicted stress increased led Stein and Lisowski (1983) to infer that triggered rupture can occur where the increases stresses cause favourably-oriented zones of weakness to fail such that the rupture is not connected to the main shock fault trace at depth. Das and Scholz (1981) calculate that the increase in shear stress in the far-field location approximately 10% of the stress drop on the main fault. Therefore, for new faulting to occur, the host rocks must either already be at a stress comparable to that where slip can occur, or the initial and main rupture must have a high stress drop. These ideas are discussed with reference to laboratory-induced deformation band features in Chapter 6.
[1.6] Structural and Mechanical Influences on Porosity and Permeability

[1.6.1] Introduction
In the preceding sections, I have focused on the mechanics of faulting and the structural characteristics of fault structures. Now I introduce the concept of how these structures actually influence the rocks capacity for fluid and more importantly fluid flow. Sections [1.6.2] and [1.6.3] below summarise the main influences on porosity and permeability in porous rocks. Much of the material is taken from the textbook of Guégén and Palciauskas (1994), which is a very good introduction to the main (and generally accepted) structural influences on the hydraulic properties of rock. Also see Means (1976) if required. Section [1.6.3] introduces some important aspects of permeability derived from laboratory studies.

[1.6.2] Porosity
Porosity is defined as the pore volume per unit rock volume, and is commonly denoted the symbol $\phi$. The initial porosity (primary porosity) of a granular medium is a function of three main variables: grain size; grain shape; and the distribution of grain sizes (i.e. degree of sorting).

Grain size: For a uniform grain size in a randomly packed medium, porosity tends to increase with particle size reduction. This is because the load required to overcome frictional and cohesive forces (i.e. on grain surfaces), increases systematically with surface area. Surface area is inversely proportional to grain size, and hence smaller particles reach a stable state at a larger porosity than large grains. For regular (e.g. face-centred or close-packed cubic) packing of ideal spheres of a single grain size, a reduction in radius has no effect on pore space.

Grain shape: The influence of grain shape is not yet well understood. However, studies using idealised non-spherical particles of equal size yield porosities that are systematically greater than those for corresponding sized spheres. In this case the porosities are also independent of size (Wyllie and Gregory 1955).

Grain size distribution: A mixture of grain sizes will invariably cause a reduction in porosity with respect to that of a unimodal size distribution. The extreme cases describing a matrix of uniform large grain-size and a matrix of a uniform small grain-size are illustrated in Figures: 1.21a and 1.22a respectively. The addition of fine particles ('fines') to a uniform large grained matrix (figure: 1.21b) will reduce porosity for a given rock volume as the fines will sit in the pore space between the large grains. The structural style associated with this matrix would be referred to as grain-supported (Figure: 1.21c). The addition of large grains
Figure: 1.21 Schematic diagram showing a) framework of large grains; b) influx of fine particles; c) grain supported fabric of reduced porosity (with respect to the initial state).

Figure: 1.22 Schematic diagram showing a) framework of fine grains; b) addition of a few large grains; c) resulting in a matrix supported fabric of reduced porosity (with respect to the initial state).
into a uniform small-grained rock (Figure: 1.22b) will also decrease porosity for a given volume, as volume which was voids is now replaced by an equal volume of solids (figure: 1.22c). The structural texture here would be defined as matrix-supported (Figure: 1.22c). Beard and Weyl (1973) measured initial mean porosity for an unconsolidated sand as a function of the degree of sorting. They observed a monotonic increase in % porosity for an increase in the degree of sorting from 27.9% (for very poorly sorted) to 42.4% (for extremely well sorted). This implies that the maximum porosity will occur for perfectly sorted sediments. Any deformation which results in the generation of fine particles and poorer sorting will therefore systematically reduce the porosity.

The porosity of a granular material changes due to sample consolidation and compaction which may result in grain fracturing, grain dissolution or recrystallisation of grains. This porosity formed during diagenetic or permanent deformation processes is commonly termed secondary porosity.

[1.6.3] Permeability
Permeability is a measure of the ease with which fluids can flow through rocks. Fluid flow is often described by Darcy’s law:

\[ q = -\frac{k}{\eta} \frac{dp}{dx} \]  equation: 1.6

where \( q \) is the Darcy velocity, \( \frac{dp}{dx} \) is the hydraulic pressure gradient across the sample, \( \eta \) is fluid viscosity and \( k \) is the permeability coefficient. No simple relation exists between porosity and permeability, since permeability does not depend solely on primary porosity but also on the nature of the porosity microstructure, in particular its degree of connectivity in 3-dimensions. Small-scale heterogeneities in pores e.g. rough surfaces and the effective path length that a fluid must follow (termed the tortuosity) can significantly alter permeability even though these heterogeneities have little effect on porosity. When secondary porosity is taken into account, the scatter in the correlation between porosity and permeability increases and permeability often becomes strongly directional. Similar to porosity, permeability also reduces with poor sorting. However, the permeability reduction depends on pore geometries and pore throat size, rather than the size of the large pores. Hence permeability is strongly controlled by grain shape, grain geometry and grain packing.
Several models have been derived to give approximations of permeability in rocks by idealising pathways into tubes or cracks. Although such analytical theoretical models can never accurately predict the absolute permeability of a ‘real rock’, they are very useful in highlighting the controlling geometric factors. One example is a tube model (Guéguen and Diennes 1989) comprising a network of capillary tubes of different radii (Figure: 1.23). Inside the tubes of radius \( r \), length \( d \), and separation distance \( l \), fluid has an average velocity \( v \) given by Poisseuille’s law (sometimes known as the ‘r-squared’ law):

\[
v = -\frac{r^2 dp}{8\eta dx}
\]  \hspace{1cm} \text{equation: 1.7}

With some assumptions and manipulation, this leads to the equation

\[
k = \frac{\pi r^4 d}{8 l^3}
\]  \hspace{1cm} \text{equation: 1.8}

(derived in Guéguen and Palciauskas, 1994) showing that the permeability depends on the variables \( d \), \( r \), and \( l \). As a consequence it is predicted that small pores or pore throats (hence small \( r \)) will rapidly reduce permeability.

An added complication is that the permeability of rocks depends not only on the micro-pore structure but the ambient conditions the rock is subject to (e.g. pressure and temperature). In addition, time dependent effects are important e.g. healing of microcracks through precipitation of silica from hydrothermal or diagenetic circulating fluids (Lloyd and Knipe 1992). The question of whether permeability increases or decreases with deformation is not a trivial one. Deviatoric stress will reduce porosity and permeability through kinetic compaction processes (see section [1.4.9]) but may also be responsible for introducing both new microcracks and shear-induced dilation (see section [1.4.8]), which will increase permeability.

[1.6.4] Laboratory studies of permeability

Laboratory studies have provided an important insight into the evolution of permeability with deformation. Early efforts were concentrated on crystalline rocks (where deformation is more straightforward since porosity is mainly due to fractures), and this established a fairly good picture of permeability evolution in such rock types. Porous sedimentary rocks, however, have a much more complex behaviour due to the existence (and interplay) of both pores and microfractures. The evolution of permeability in deforming sedimentary rocks has not yet been fully explained, although several important characteristics have been observed by previous workers. For example, Zoback and Byerlee (1975) observed that the
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Figure: 1.23 Capillary tube model for permeability: $r$ is the radius, $d$ is the length, and $l$ the average distance between two tubes (Guéguen and Diennes, 1989). See main text section [1.6.3] for calculation of permeability.
permeability of an intact sandstone was inversely related to confining pressure under hydrostatic loading conditions. In addition, an unconsolidated sand, deformed under increasing hydrostatic compression, underwent a gradual decrease in porosity and permeability with increased pressure, which accelerated markedly at a critical pressure corresponding to the onset of grain crushing and skeletal collapse (Zoback and Byerlee 1976). This acceleration is more marked under conditions of true triaxial stress ($\sigma_1 > \sigma_2 > \sigma_3$), (Crawford et al., 1995) and active pore fluid chemistry (Main et al., 1996). During tests of deformed layers of borehole gouge, Morrow et al. (1984) recognised, that gouge (or rock flour) may be as effective as clay gouges in reducing permeability. Chester and Logan (1986) observed that the permeability of wall rock, which showed dilatancy prior to failure, was higher than fault gouge, which exhibited a continuous reduction in porosity and permeability throughout deformation.

Recently, Zhu and Wong (1997) carried out a comprehensive experimental programme to investigate the evolution of permeability with deformation in a series of porous sandstones in triaxial compression. Conditions covered a range from brittle faulting to cataclastic flow, and their main conclusions were as follows. Permeability consistently decreased with increasing strain whether failure occurred by shear localisation or cataclastic flow. There is a positive correlation between porosity and permeability for porous sandstone throughout deformation in the cataclastic flow regime. However, in the brittle faulting regime, porosity and permeability show a positive correlation only in the initial stages of loading. In the dilatant phase, porosity increases but permeability still decreases (showing a short lived negative correlation), then after failure, permeability decreases rapidly with relatively little change in porosity (i.e. indicating no correlation at all).

1.7 What’s new about this project and which gaps will it fill?

The deformation mechanisms involved in the formation and evolution of deformation bands in sandstone are not yet completely understood. Laboratory experiments have offered good explanations for some stages of the process, and have produced features which exhibit many of the characteristics of natural faults, in particular the formation of single deformation bands. However, to the authors knowledge no previous experiments have reproduced geologically-realistic faults showing the hierarchical development of sets of deformation bands side by side, (e.g. Figure: 1.18). This may be due to the limited size of cylindrical rock samples (in general 1-1.5” diameter) used for previous experiments. Traditionally, the ratio of 10:1 between smallest sample dimension and grain size was thought sufficient to
make measurements which would represent those of the bulk rock, hence 1-1.5" samples of medium grained rock were deemed perfectly adequate. More recently the importance of a process zone in crack propagation has been recognised, and the idea that one fixed grain : sample size ratio can be ascribed is now doubtful. However, it is still generally assumed that a 1"-1.5" diameter core sample will have a certain area at its centre where deformation is not significantly affected by edge effects, and therefore this area can reasonably be used to approximate the bulk material. A larger sample will presumably have a much larger sample volume where edge effects are minimal, and hence will give a better approximation to natural conditions. It may seem puzzling that larger samples have received relatively little attention, however, they do pose some technical difficulties, which have most likely limited their use. The main draw-back in carrying out rock mechanics tests on very large samples is the relation shown in equation: 1.9

\[ \sigma_1 = \frac{F}{A} = \frac{F}{\pi r^2} \] equation: 1.9

where \( \sigma_1 \) is axial stress, \( F \) is axial force acting on the sample faces, \( A \) is the cross-sectional area of the sample which in the case of a cylindrical sample is proportional to \( r^2 \). This clearly indicates that the force required to apply a given stress to a sample increases as the square of the sample radius. Hence a much larger (and more expensive) deformation apparatus is required to produce the force needed to deform large samples.

In order to maximise the area where structures may develop, and in which displacements may be observed, I have deformed significantly larger samples (4" diameter cylindrical cores) in a new large capacity triaxial deformation rig within this study. The ratio of sample diameters in the two set-ups (4:1) may not appear very significant at first. However, the ratios of grain size to sample size in each case can be calculated assuming mean grain size of the host rock to reveal the ratios 150:1 (for 1" diameter tests) as opposed to 600:1 (for 4" diameter tests). This implies there are 150 and 600 grains across the diameters of a 1" and 4" diameter cylinder respectively which clearly is significant if one considers the typical spacing of a set of deformation bands observed in the field as approximately 10-20 grain diameters.

Due to the limited absolute displacement and relatively small fracture surface area produced in 1-1.5" diameter samples, little gouge material is produced during fracture and sliding. In general, previous experiments investigating gouge characteristics required the deformation of simulated gouge packed along pre-existing saw-cut fractures. As a result, much experimental work has been concentrated on either the failure process itself or sliding
on a pre-existing artificial fault and relatively few studies have encompassed both (c.f. Bernabé and Brace, 1990; Wong et al., 1997). Using larger samples, a much larger surface area of fracture is produced for a given axial strain, and hence more gouge material is produced. Therefore, it is possible to study both the fracture process and the end-product gouge material. An additional advantage of the large-scale tests is that a larger absolute volume of the sample is free from edge effects, and therefore it is possible to construct a standard thin section entirely free of edge effects, unlike the smaller-sample tests.

The rationale in this study is to create faults within intact material and continue sliding deformation on the faults, thereby monitoring the pre-, syn- and post-failure stress-strain characteristics, in addition to the larger scale structures and the characteristics of the gouge material produced.

In the tests described in this thesis, initially intact dry samples were deformed to increasing amounts of axial strain at conditions simulating a few kilometers depth. Stress and strain during loading was monitored, and the samples were subsequently unloaded and subjected to post test geological analysis including laser particle size analysis, thin section analysis and SEM. Sets of deformation bands were developed side by side within deformed samples, with an increasing number of bands developing with increasing strain.

[1.8] Thesis Organisation
Following this introductory chapter, which has detailed the concepts necessary to appreciate this study, previous relevant studies, along with a brief outline of the project aims and method, the remainder of this thesis is organised in the following way: Chapter 2 describes the experimental apparatus, monitoring equipment and test procedure used for triaxial compression experiments; Chapter 3 presents the mechanical data monitored during the deformation tests; Chapter 4 describes the structural data acquired at the end of the tests; Chapter 5 summarises the fieldwork carried out in Lossiemouth, North-east Scotland and compares structural data collected to the laboratory data presented in Chapter 4; Chapter 6 presents a synthesis and discussion of the results presented in the preceding chapters; and Chapter 7 states my conclusions and recommendations for future work.
CHAPTER 2: EXPERIMENTAL APPARATUS AND METHODOLOGY

[2.1] Introduction

Rocks in the Earth's crust experience a triaxial stress state normally described in terms of three principle compressive stresses: $\sigma_1$ (maximum stress); $\sigma_2$ (intermediate stress); and $\sigma_3$ (minimum stress). The relative magnitudes of these three stresses (coupled with its intrinsic material properties) determine how a rock will deform. Therefore this stress state is simulated in the laboratory during rock mechanics testing to investigate the development of deformation. When deforming cube shaped samples, different stresses can be applied independently to each face (polyaxial loading). However in the case of a cylindrical sample, this is normally done by what is termed triaxial loading. Here an endload (hence $\sigma_1$) is applied, by means of a hydraulic ram, to the flat end faces, and $\sigma_2 = \sigma_3$ are applied by squeezing the cylindrical sides of a sample sheathed in a non permeable jacket. Tests are generally conducted by first applying a hydrostatic stress (i.e. $\sigma_1 = \sigma_2 = \sigma_3$ ) to the sample then adding a differential stress by increasing axial load to the sample ends (i.e. $\sigma_1 > \sigma_2 = \sigma_3$). This chapter introduces the experimental apparatus used in this study to deform cylindrical samples, the systems which monitor deformation during a test and finally the test procedure employed.

The experimental work carried out in this study is designed to investigate the development of faulting in initially intact porous rock. The two main variables considered are the influence of increasing axial strain (i.e. amount of deformation accommodated) and the influence of confining pressure (i.e. simulating burial depth). Two series of experiments are carried out to investigate the way in which these parameters affect the mechanical and structural development of deformation structures. Series 1 tests investigate the influence of axial strain alone by keeping other parameters (e.g. confining pressure) constant whilst applying increasing amounts of axial strain. Individual tests are terminated when different amounts of strain have been achieved. Samples are then unloaded and analysed to investigate the structures formed during progressively greater amounts of deformation. Series 2 tests are designed to isolate the influence of confining pressure. In this series, individual samples are deformed under different amounts of confining pressures whilst the axial strain applied to all tests is constant. Also, the influence of sample bedding orientation on mechanical and structural data is considered by deforming samples with bedding perpendicular to, and parallel to maximum stress.
[2.2] Large capacity deformation rig

A large-capacity servo-controlled triaxial deformation rig is used to apply axial load and radial confining pressure to 100 mm diameter right cylindrical samples of ~230 mm length (see Figures: 2.1, 2.2). Here maximum stresses are applied axially, and minimum and intermediate stresses are equal, and are applied by radial confining pressure. The apparatus, shown in figure: 2.1, consists of: a) a linear actuator to provide axial load; b) a basal plinth where the sample sits centrally on the bottom steel platen; c) a bell shaped pressure vessel, within which the sample sits and confining pressure is achieved; d) a confining pressure intensifier which supplies high pressure confining fluid to the vessel; and e) a very stiff solid loading frame designed to minimise elastic stretching of the rig during deformation experiments. A large motor pump unit supplies pressurised oil to the main hydraulic circuit which feeds the actuator and intensifiers via servo valves (marked 'S' in figures: 2.1a, and d respectively). An electronic control panel (Figure: 2.1f) controls these servo valves and a set of transducers monitors pressures and piston displacements continuously during a test.
Figure 2.1 Photograph of the large-capacity deformation rig, showing: a) the vertical piston used to apply axial load to the ends of the sample; b) the basal plinth with lower platen (located directly above internal load cell) clearly shown in the centre; c) confining pressure vessel surrounded by the furnace and insulation which is bolted down onto the base shown in b); d) the servo controlled confining pressure intensifier; e) the extremely stiff loading frame which prevents the rig itself stretching during a rock deformation test; and f) the electronic control panel. 'S' indicates servo-valves in a) and c).
Figure: 2.1 a

Figure: 2.1 b
Chapter 2 - Experimental Apparatus and Methodology

Figure: 2.1c

Figure: 2.1d
Figure: 2.1e

Figure: 2.1f
Figure: 2.2 is a schematic cross section drawing, indicating the relative positions of the elements which make up the large capacity deformation apparatus.

In the following sections I describe the rock sample characteristics, the deformation rig hardware and control systems, the equipment which monitor physical quantities during a test, and show how this data is acquired and processed in preparation for subsequent presentation and interpretation in chapter 3. Finally, the test procedures followed during the series of deformation experiments which form the basis of this study are described.

[2.3] Sample description and sample stack

[2.3.1] Sample rock type

The sample rock type used in all tests carried out in the large capacity rig was Locharbriggs sandstone, a lower Permian aeolian sandstone which is highly porous and quartz rich. This rock-type has been previously characterised by Ngwenya et al. (1990) see table: 2.0. This sandstone is analogous to the Penrith sandstone studied in the fieldwork of Fowles and Burley (1994). No clay is present, but hematite cement is present as a coating on grains and there is only a small amount of feldspar. Fresh (non-weathered) samples were obtained from a working quarry in Dumfries, South West Scotland. 100 mm diameter cylinders were cored from blocks of Locharbriggs in directions perpendicular and parallel to the bedding lamination using a diamond coring drill. The sample ends were then lathed one at a time to produce precise right angled cylinders of the appropriate length. All sample preparation was carried out at room temperature.

[2.3.2] Sample stack

Each sample was jacketed in a viton rubber sleeve to isolate it from the confining fluid. The sample stack consisted of two steel mushroom shaped platens, which contacted the upper and lower ends of the sample (see Figure: 2.3). The steel platens are exactly the same size as the samples. According to Jaeger and Cook (1979), this arrangement eliminates the effect of a bending interface and diminishes any lateral constraint on the sample. The lower platen contacted the sample via a spacer (if required) and a Vaseline Petroleum Jelly layer which coated a thin melinex disc (Melinex is a very strong polyester with low friction properties). This arrangement reduced end piece friction, which is thought to inhibit shear development by preventing the sideways movement of the sample necessary for localisation along a plane (Scott et al., 1994). Self-amalgamating tape was used to bond the rubber sleeve to the platen, in order to prevent leakage of confining fluid into the sample during the initial stages
Figure: 2.2 Schematic diagram of a cross section of the large scale deformation rig, showing the relative positions of the load frame, actuator, confining pressure vessel, sample and jacket in place.
Table: 2.0 Rock sample properties

<table>
<thead>
<tr>
<th>Description</th>
<th>Lower Permian, aeolian sandstone rounded - well rounded</th>
</tr>
</thead>
<tbody>
<tr>
<td>Grain shape</td>
<td>fine-medium</td>
</tr>
<tr>
<td>Grain size</td>
<td>hematite grain coatings (+ some silica)</td>
</tr>
<tr>
<td>Cement</td>
<td>lamellae ~ 1mm thick</td>
</tr>
<tr>
<td>Bedding</td>
<td>Locharbriggs North Quarry, Baird and Stevenson, Dumfries</td>
</tr>
<tr>
<td>Location</td>
<td></td>
</tr>
<tr>
<td>Porosity</td>
<td>22.2%</td>
</tr>
<tr>
<td>Quartz*</td>
<td>88.1%</td>
</tr>
<tr>
<td>K-Feldspar*</td>
<td>5.9%</td>
</tr>
<tr>
<td>Kaolinite*</td>
<td>0.0%</td>
</tr>
<tr>
<td>Siderite*</td>
<td>0.0%</td>
</tr>
<tr>
<td>Dolomite*</td>
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</tr>
<tr>
<td>SiO2**</td>
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<tr>
<td>Al2O3**</td>
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<tr>
<td>MgO**</td>
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<tr>
<td>Na2O**</td>
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<tr>
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<tr>
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<td>0.1%</td>
</tr>
<tr>
<td>MnO**</td>
<td>0.0%</td>
</tr>
<tr>
<td>P2O5**</td>
<td>0.0%</td>
</tr>
<tr>
<td>LOI**</td>
<td>0.2%</td>
</tr>
</tbody>
</table>

* Semi-quantitative modal mineralogy (%) data using XRD from Ngwenya et al. 1990.
** Major element (%) data using XRF from Ngwenya et al. 1990.
Figure: 2.3 Schematic diagram of rock sample and steel end piece arrangement showing 100mm diameter rock sample, surrounded by rubber sleeve. The lower end platen meets the sample via a melinex disc and Vaseline jelly, in order to lubricate the sample end and facilitate lateral movement during deformation.

Figure: 2.4 Calculated axi-symmetric stress distributions in a cylinder under uniaxial compression (from Paterson 1978 ch3), showing the most highly stressed areas as shaded. Note the stress concentrations at the corners of the sample, and in two blocks neighbouring the centre of the sample. These are the areas where cracking would be expected to initiate. The contours show that non uniform stresses would be expected for a distance into the sample which depends on the sample dimensions. If samples have l:d >>1, then a uniform stress field can be achieved in the centre part of the sample. As the l:d ratio increases, a larger proportion of the sample is experiencing a uniform stress field. In triaxial tests a ratio of 2:1 is sufficient, in uniaxial tests the ratio is 3:1.
of loading (tests run7 to run16). Once the confining pressure started to rise the rubber sleeve would be squeezed tight against the end platens and no leakage of confining fluid is expected. In early tests (run1 to run6) copper wire was wound round the top and bottom of the rubber jacket to prevent leaks of confining fluid into the sample, but failure of the wire did allow some oil leakage so this method was subsequently abandoned. Viton rubber can change its properties (becoming stiffer at high pressures), but at the test conditions experienced here, this change is very unlikely to occur. I assumed that negligible load can be supported by the rubber jacket and the self-amalgamating tape, due to their relative weakness when compared to the rock (after Weeks, 1980).

[2.3.3] Sample size and aspect ratio considerations

The length : diameter ratio for the samples used here is approximately 2:1. Studies (e.g. Paterson, 1978; Jaeger and Cook, 1979) have shown that this aspect ratio is acceptable as the central part of the samples is only slightly affected by platen end effects. Samples with l : d ratios smaller than 2:1 encounter problems with end effects. In triaxial tests the effects are less severe than in uniaxial tests, where 3:1 aspect ratio samples are advisable. As the dimensions of a sample increase, the fracture strength has been measured to decrease (e.g. Paterson, 1978). This is an important consideration if the large scale tests are compared to smaller sample sizes. Figure: 2.4 (redrawn from Paterson, 1978) shows a theoretical calculation of the elastic stress distributions within a cylindrical sample loaded under uniaxial compression.

[2.4] The Hardware

[2.4.1] Introduction

This section introduces the various components of the rock deformation press hardware. The principles of servo-feedback and a process-response loop are now normally used in the control systems of rock mechanics apparatus. The deformation rig used in this study is no exception, and where relevant these concepts are introduced. For a more detailed account see an introductory engineering text e.g. Haslam et al. (1981).

[2.4.2] Linear Vertical Actuator - Axial Load

Axial load is provided by a 2000 kN vertical linear actuator which can be driven in compression or tension (see figures: 2.5, 2.1a, 2.2). Pressurised oil is pumped from the main hydraulic circuit at a pressure of 3500 psi (24.1 MPa) through a MOOG-76 electro-hydraulic
Figure 2.5 Schematic diagram showing the hydraulic circuit and independent servo-control of the main actuator and the confining pressure intensifier of the large scale deformation rig. The locations of load cells and linear variable displacement transducers used to monitor load and axial strain during a test are also shown. (Load cell 1 is fed back to the servo control loop in the same way as load cell 2, but this is not shown on the diagram due to lack of space. The pore pressure intensifier, also because its circuit is similar to the confining pressure, is not shown here for the same reason.)

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servo valve (see ‘S’ in figure: 2.1a) to the actuator, which moves a piston, causing an axial load to be applied to the top end piece, and hence to the sample end. The movement of the actuator is constantly monitored by a series of transducers which can convert either a movement or pressure into an electrical signal. A Linear Variable Differential Transformer (l.v.d.t.) monitors position of the piston, known as its stroke, and two Load Cells (see Figure: 2.5) monitor the load applied to the sample. Transducer outputs are scaled and demodulated in the Amplifier then appear as digital readouts on the digital peak meter (d.p.m.).

[2.4.3] Load cell, Main ram
The actuator provides the downward force but the transmission of this force to the sample is controlled by the main ram. A design drawing of the main ram is presented in Figure: 2.6 and is described in the section [2.4.4] below.

[2.4.4] Confining Pressure Intensifier - Radial Confining Pressure
The rock sample is housed inside a bell shaped steel pressure vessel rated to 10000 psi (69 MPa). Radial confining pressure is achieved by surrounding the sample, sleeved in an impermeable Viton rubber jacket, with a confining fluid under pressure. The confining pressure piston cylinder intensifier (Figure: 2.7) is charged with confining fluid, in this case low viscosity hydraulic oil (Tellus 68) which is extremely incompressible. The intensifier piston is then extended and a confining pressure is achieved. The piston is extended by applying pressurised oil from the main hydraulic circuit via a servo valve into the lower half of the intensifier. The piston can only be driven forward (unlike the main actuator) and relies on the pressure of the confining fluid to force it backwards. The servo valve can control the piston travel and therefore the application, reduction or maintenance of confining pressure. The pressure of the confining fluid leaving the top of the intensifier is constantly measured using a pressure transducer, and the movement of the piston is monitored using an l.v.d.t.

Pore fluid under pressure may also be forced into the porous rock sample itself through a hole in the lower endpiece and pressurised using a separate piston cylinder intensifier, similar in design to the confining pressure intensifier described above. However, pore fluids were not present in any of the tests described in this thesis.
Figure: 2.6 Schematic diagram showing the design of the pressure balanced ram within the confining pressure vessel. The ram is balanced by applying confining fluid both to the sample sides and to the end piece via the volumes shown in dark grey. The route taken by confining fluid in order to balance the ram and equalise pressure on all sides of the sample is shown by the dark grey line connecting the confining fluid reservoir and volume behind the ram. The air bleed at the front side of the ram is also shown. Cross sectional areas A and B referred to in the main text are highlighted, together with the locations of the cap piece, rock sample and pore fluid access.
Figure: 2.7 Diagram of a cross section of the confining pressure intensifier. The confining fluid is located in the shaded region above the piston. The pressure of the oil here is monitored by the pressure transducer at the top of the intensifier. Displacement of the piston is monitored by the L.V.D.T. underneath the piston. The MOOG servo valve which controls flow of oil from the hydraulic circuit into the intensifier to drive the piston is on the lower right. The pore pressure intensifier is analogous to this except that it has a lower volume capacity for pore fluid.
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[2.4.5] Pressure Balanced Ram

The rig has a pressure balanced ram which means that when confining pressure is applied to the circumference of samples, it is also applied to the back side of the ram through a connection port bored in the top of the pressure vessel (see Figure: 2.6). This creates a downward (compressional) force on the platens and sample ends, hence applying a near hydrostatic load to the sample. One essential feature of the ram is that the cross sectional area of the back side of the ram (area A) is identical to the cross sectional area of the sample endpiece (area B). This results in an equal pressure being applied to the sample as is applied to the back side of the ram by the confining oil. Another important feature is that as the ram advances into the sample during a test, confining fluid will be displaced from the pressure vessel via the connection port into the space created at the back of the ram. The cross sectional areas and the vertical movement of the ram and the piston endpiece are identical, so a constant confining fluid volume is maintained in the pressure vessel during loading. Volumomentry measurements can therefore be carried out without the need to consider the effects of the advancing ram. Without such a pressure balanced ram, the volume available for fluid in the pressure vessel would gradually decrease due to the extra volume taken up by the ram and would make confining fluid volumomentry more difficult. It is assumed that the flow of oil from the pressure vessel to the back of the ram will be more or less instantaneous which is probably a fair assumption considering the pressure differential which will promote rapid flow. The axial load applied is required to be only the sum of the differential load + the seal friction i.e. the confining pressure does some of the work.

[2.5] Hardware Control System

[2.5.1] Electronic Control System

The main actuator, the confining pressure and pore fluid pressure intensifiers are independently operated by an electronic control panel (see Figures: 2.1f, 2.8). The hydraulic control unit operates the motor on the hydraulic power pack, which generates pressure for the main hydraulic circuit, and also the solenoid valve, which moves to allow high-pressure hydraulic oil to enter the circuit and reach the servo valves. Safety interlock relays monitor overload conditions in the power pack, and automatically shut down the hydraulic power
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pack, preventing operation of the hydraulic power in the event of such a condition. A set of six limit detectors can be set manually for load, strain and stroke feedback signals. Signals received which exceed these limits will shut off hydraulic power immediately, and the relevant error will be indicated by a light on the relevant limit detector. Error pressure level trips for the confining pressure and pore fluid pressure servo controllers are located on the appropriate racks see figure: 2.8.

The actuator is driven under either stroke, strain or load control by the appropriate servo controller. Only one servo controller may be in the control loop at any one time, the others are isolated from the servo valve amplifier but are connected to transducers to monitor their output. The servo controller compares the signal fed back by the transducers (through an amplifier) to the command signal requested by the operator, and attenuates or amplifies the resulting error. This error signal is then sent to the servo valve amplifier which converts this error from a voltage into a control current to regulate the flow rate of hydraulic oil through the servo valve. The servo valve opens or closes to affect hydraulic oil flow, and therefore move the actuator in order to reduce the error (the difference between the desired boundary condition and the measured one) to zero.

Confining pressure is controlled by a servo controller which compares the feedback transducer signal (via an amplifier) to the command signal. The error between these is passed to the confining servo amplifier which modifies the signal to control the flow rate of oil in the servo valve in order to move the intensifier piston and subsequently reduce the error to zero. As the intensifier piston can be driven only in one direction i.e. forward to apply confining pressure, if oil pressure being applied is higher than that requested, the piston must simply be allowed to float backwards under the pressure of the confining fluid until zero error has been achieved. Pore fluid pressure is controlled independently but an identical way to confining pressure.

[2.5.2] Computer Control

The servo hydraulic control system can be run either under manual or computer control. Computer control is achieved by turning off the hydraulic pressure to the rig circuit, turning a key in the hydraulic control unit to ‘computer’ then bringing up the hydraulic circuit pressure again. The Tandon Personal Computer uses custom-designed testing software provided by the manufacturer (E.S.H. Testing, Birmingham, UK) to operate the rig. A simple block programme must be created in order to drive the equipment, the blocks being a series of timed ram sequences. The rig will perform the actions requested in each block in
Figure: 2.8 Line drawing of the electronic control panel. Digital panel meters showing feedback signals from all transducers are located on the top rack. The second rack has stroke strain and load amplifiers for the actuator. Servo controllers for the actuator are next then the hydraulic control unit and limit detectors are located on the fourth rack. Below is the waveform signal generator. On the sixth and seventh racks are found the confining pressure controllers and pore fluid pressure controllers respectively. The furnace controller is located on the bottom rack.
Figure: 2.8 (continued)
sequence, until it reaches the final block, or an error is tripped and hydraulic power is cut off. The following parameters must be selected for each block:

**Select** (operation) **STROKE, LOAD, WAIT or END**

**Duration** (time required to move from present position to new required position) in s

**Mean level** (new final position required) in **mm**

**Confining Pressure** (required fluid pressure) in **p.s.i.**

**Pore Pressure** (required fluid pressure) in **p.s.i.**

**Acquire data** (a disc is required) **Y, N**

The mixture of Imperial and S.I. units common to engineering applications in the UK is a fundamental design of the rig. The conversion here is $1 \text{ MPa} = 145 \text{ psi}$. Figure: 2.9 shows an example of a block programme used to control the rig.

The hardware amplifier ranges selected on the electronic control panel must also be programmed into the computer. These are used in reducing transducer outputs in Volts to physical quantities. The test can then be started by typing RUN and the computer will automatically carry out the whole test. Caution must be practised when controlling the rig under computer control, as the computer assumes all mean level potentiometers on the control panel are nulled. If this is not actually the case it can lead to unexpected and dangerous movements of the actuator.

Problems have been detected with the ESH Testing software time counter. Although the actual clock time is accurate, the seconds counter used to countdown each block in the programme is inaccurate. The ESH counter seconds are approximately 13% shorter than standard seconds. This results in each loading block being slightly shorter than anticipated leading to slightly different strain rates from those requested. Table: 2.1 presents the differences in requested and actual applied strain rates. Fortunately the time information logged during a test is accurate, so these values can be used in data processing without the need for corrections.

[2.6] **Data Monitoring Systems**

[2.6.1] **Data Monitoring Equipment**

A series of pressure and displacement transducers are used to continuously record the pressures in the intensifiers, the load at the sample and the movement of the pistons.
block which has no operation but is required to satisfy condition A. Block C is a rest period to allow equilibration of the sample. Block D is the unload block where the actuator is pulled off of the sample and returned to the zero load position in order to move (extend) the top of the sample. Block E is the load block. If the actuator is in position 0, the top of the sample is unloaded. Block F reverses the load block to allow equilibration of the sample. Block G is a rest block. Block H reverses the load block and returns the actuator to the zero load position. Block I reverses the load block and returns the actuator to the zero load position.

---

Chapter 2 - Experimental Apparatus and Methodology
Table: 2.1 The difference between the requested and actual loading rate.

<table>
<thead>
<tr>
<th>Requested time (s)</th>
<th>Actual duration (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>10</td>
<td>8.7</td>
</tr>
<tr>
<td>20</td>
<td>17.4</td>
</tr>
<tr>
<td>30</td>
<td>26.1</td>
</tr>
<tr>
<td>40</td>
<td>34.8</td>
</tr>
<tr>
<td>50</td>
<td>43.5</td>
</tr>
<tr>
<td>60</td>
<td>52.2</td>
</tr>
<tr>
<td>70</td>
<td>60.9</td>
</tr>
<tr>
<td>80</td>
<td>69.6</td>
</tr>
<tr>
<td>90</td>
<td>78.3</td>
</tr>
<tr>
<td>100</td>
<td>87</td>
</tr>
</tbody>
</table>
## Table: 2.2 Summary for specifications for the large scale deformation rig

<table>
<thead>
<tr>
<th>Specification</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>Standard Sample Diameter</td>
<td>100 mm</td>
</tr>
<tr>
<td>Sample Length (max.)</td>
<td>245 mm (from blueprint)</td>
</tr>
<tr>
<td>Sample Shape</td>
<td>right cylinder with ends lathed parallel</td>
</tr>
<tr>
<td>Axial Load (max.)</td>
<td>250 tonnes compression (50 tonnes in tension) 2000 kN</td>
</tr>
<tr>
<td>Axial stress (max.) on 100mm sample</td>
<td>254.65MPa (assuming constant cross sectional area)</td>
</tr>
<tr>
<td>Confining Pressure (max.)</td>
<td>10,000 psi 68.92 MPa</td>
</tr>
<tr>
<td>Pore Fluid Pressure (max.)</td>
<td>10,000 psi 68.92 MPa</td>
</tr>
<tr>
<td>Stroke</td>
<td>45 mm (from blueprint)</td>
</tr>
<tr>
<td>Axial Applied Strain (max.)</td>
<td>18.36%</td>
</tr>
<tr>
<td>Temperature</td>
<td>250 degrees Celsius</td>
</tr>
<tr>
<td>Typical test duration, maximum strain and calculated strain rate</td>
<td>36720 s for 18.36% strain at 5x10E-6 /s strain rate</td>
</tr>
</tbody>
</table>
Actuator Displacement

The movement of the main actuator is constantly monitored by transducers which convert a movement or pressure into an electrical signal. An R.D.P. D5/3000 long-stroke linear variable differential transformer (l.v.d.t.) is fixed to the main actuator and monitors the position of the piston with respect to a reference point. This transducer has a stroke of +/-76 mm and linearity of +/-0.5% of the total working range (linearity is best near the null position and worst near 100% displacement). Piston position, or stroke, can be monitored in real time during a test using the digital panel meter on the control panel by turning the selector to stroke. The position in mm is calculated from this readout (given in volts) using the amplification range which is selected on the stroke amplifier. Alternatively this digital readout can be output via a BNC cable to a chart recorder and plotted against time or load.

The measurement of the actuator displacement outside the pressure vessel should be corrected for apparatus distortion due to elastic stretching of the equipment during loading. Rig distortion is normally proportional to load applied. Here the distortion of the rig is unknown so this correction cannot be applied. To monitor distortion it would be necessary to load a dummy sample of e.g. steel and measure very accurately how the rig stretches. It is assumed however, that due to the design of the load frame comprising two cylindrical legs (0.4m diameter, 2.5m length) and a top plate (0.4m thick) of solid steel, the rig is inherently very stiff, and hence elastic stretching during loading of a sandstone rock is extremely small.

Axial Load

Two load cells are used to measure the total axial force applied to either end of the sample (see Figure: 2.5). Load cell 2, an external load cell, is located in the bottom section of the actuator. External load cells require the effects of the friction of the seals between the transducer and the sample to be considered. They are only suitable if differential load is much greater than the piston friction. The problem with corrections is that the piston friction may be variable and unpredictable as loading continues. Load cell 1, the specimen load cell, is an internal load cell located directly below the bottom endpiece of the sample in the base of the machine. Here the friction of seals does not need to be considered, as there are no seals between the sensor and the sample. The readings recorded should be precisely the conditions at the sample. The main problem with internal load cells is their calibration. I have assumed that the in situ calibration of both load cells carried out by E.S.H. Testing is accurate and absolute. The values measured at these two load cells are displayed on the digital panel meter as a Voltage. They show the same behaviour during a test although their
absolute values are different by a roughly constant amount. Similar to the stroke data, the
load data can be output to chart recorder and downloaded to disc. Most load data used in
chapter 3 will be from load cell 1, since it measures the loading conditions without the need
for corrections.

[2.6.4] Pressure and piston displacement in pressure intensifiers
Pressure and piston displacement are constantly monitored in the confining and pore
pressure intensifiers. Pressure in each is measured by an Intersonde high output pressure
transducer type HR17-R (see Figure: 2.7) placed in the upper region of the intensifier. This
transducer has a working range of 0-10000 psi and has a non-linearity and hysteresis of +/-
0.14% of full range sensitivity. Displacement of the piston is measured by a Sensonic A.C.
type E2 l.v.d.t. with a stroke of +/-50 mm and linearity of +/-0.5%. This l.v.d.t. is fixed to
the piston on the underside of the intensifier.

[2.7] Data Acquisition and Reduction
[2.7.1] Data Logging During Rock Mechanics Tests
During a test, under computer control, the outputs from all transducers are automatically
logged by the PC running the test. The sampling rate of acquisition of this data depends on
the time span of a loading block. In initial tests (run1d - run5d), a single loading block was
used for each major loading episode. It was noted, however, that a maximum of 300 data
samples were taken during each individual block. This was sufficient for tests comprising a
series of loading blocks of relatively short duration, but tests with loading blocks of longer
duration would have corresponding poorer sampling rates. From run6d-run16d, all tests
with loading time longer than approximately 50 minutes were split up into several sub
loading blocks to enable a much higher rate of data acquisition. The total number of blocks
was limited to a maximum of 12, which ultimately controlled the maximum attainable
sampling rate. Sampling rates for each block are calculated in the data processing stage
presented in the next section.

[2.7.2] Data reduction during rock mechanics tests
The transducer outputs are the values which are displayed on the digital peak meters as
scaled voltages during a test. E.S.H. Testing software reduces transducer output signals to
the (uncorrected) physical quantities which they represent using the hardware range settings,
input by the operator, as follows:
Load cell outputs to force in kN,
L.v.d.t outputs to displacement in mm,
Intensifier pressure to pressure in p.s.i.

This data is downloaded to disc at the end of each load block. The accuracy of the transducer measurements are assumed to be equal to the manufacturers specifications stated in the previous section.

[2.8] Data Processing

[2.8.1] Calculation of standard petrophysical parameters from logged data

The raw time, l.v.d.t., load and confining pressure piston displacement data was converted into axial stress, strain and confining pressure volume respectively by a purpose designed Excel spreadsheet created using Visual Basic for applications. Figure: 2.10 shows a printout of the Microsoft Excel 7.0 processing spreadsheet. Raw run data in the form of a single Excel data sheet is automatically accessed by the processing spreadsheet by “linking” the sheets together. Test parameters e.g. run i.d., strain applied, initial sample length, are added manually to the spreadsheet, and relevant quantities are automatically calculated. A new sub sheet is created for each timed loading block. The main calculations carried out are outlined below.

[2.8.2] Calculation of axial strain

Axial strain throughout the test is calculated by dividing the change in the measurement of actuator position by a measurement of initial sample length (i.e. change in length / initial length). The deformation of the testing rig itself is assumed to be negligible, and hence is not taken into account during this calculation. As discussed above, this is a reasonable assumption when deforming sandstone although it may not hold for tests on harder rocks like granites.

[2.8.3] Calculation of axial stress

Axial stress is calculated by dividing the values of load by the initial cross sectional area of the sample of e.g. 100 mm diameter. It is assumed that the cross sectional area of the sample remains constant throughout the test. This is a reasonable approximation, at least until the sample fails (Scott et al., 1994). After failure the load bearing area may be significantly different from the initial area due to the gradual disintegration of the sample.
**BIG RIG - ROCK DEFORMATION TEST ANALYSIS**

<table>
<thead>
<tr>
<th>Test Identifier</th>
<th>run15</th>
<th>Block Number</th>
<th>1</th>
</tr>
</thead>
<tbody>
<tr>
<td>Test Date (ddmmyy)</td>
<td>15/01/97</td>
<td>Block Function</td>
<td>(on, cpon, piston, pon, Jon,loff, poff, cpoft, off)</td>
</tr>
<tr>
<td>Rocktype</td>
<td>Loch</td>
<td>Points in Block</td>
<td>150</td>
</tr>
<tr>
<td>Sample Diameter</td>
<td>100 mm</td>
<td>Sampling Rate</td>
<td>5.749329</td>
</tr>
<tr>
<td>Sample Length</td>
<td>229 mm</td>
<td>Total Blocks</td>
<td>8</td>
</tr>
<tr>
<td>Axial Displacement</td>
<td>6.87 mm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Time (for Loading)</td>
<td>6000 s</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Axial Strain</td>
<td>3 %</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Axial Strain Rate</td>
<td>0.000005 /s</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Confining Pressure</td>
<td>5000 psi</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Pore Fluid Pressure</td>
<td>dry psi</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Test history is logged in a series of blocks, generally applying one procedure.

Load operations are split into a series of blocks to increase sampling rate.

<table>
<thead>
<tr>
<th>Time (for Loading)</th>
<th>Stroke (mm)</th>
<th>LVDT (um)</th>
<th>Load1 (kN)</th>
<th>Load2 (kN)</th>
<th>Press1 (psi)</th>
<th>Press2 (psi)</th>
<th>Disp1 (Eng) (mm)</th>
<th>Disp2 (Eng) (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.33</td>
<td>5.49E-02</td>
<td>-0.13428</td>
<td>8.300761</td>
<td>-11.8408</td>
<td>612.4878</td>
<td>0</td>
<td>4.747009</td>
<td>-33.7952</td>
</tr>
<tr>
<td>0.5</td>
<td>4.27E-02</td>
<td>-0.12207</td>
<td>8.300761</td>
<td>-11.8408</td>
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<td>0</td>
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<td>-33.7753</td>
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<td>0.72</td>
<td>4.27E-02</td>
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<td>8.361816</td>
<td>-11.6577</td>
<td>613.4033</td>
<td>0</td>
<td>4.734802</td>
<td>-33.7906</td>
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<td>0.88</td>
<td>2.14E-02</td>
<td>-0.12512</td>
<td>8.239746</td>
<td>-11.9019</td>
<td>613.4033</td>
<td>0</td>
<td>4.762258</td>
<td>-33.7616</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Time (for Loading)</th>
<th>ε (%)</th>
<th>σ1 (MPa)</th>
<th>σ1 (MPa)</th>
<th>σ3 (MPa)</th>
<th>σ1-σ3 (MPa)</th>
<th>σ1-σ3 (MPa)</th>
<th>σ1/ε1 (MPa)</th>
<th>Time (kyr)</th>
<th>Cnt3s</th>
<th>mm^3</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.33</td>
<td>-0.02399</td>
<td>-1.05689</td>
<td>1.50762</td>
<td>4.221143</td>
<td>44.05975</td>
<td>0.33</td>
<td>0</td>
<td>116.007</td>
<td>56.64824</td>
<td>0.5</td>
</tr>
<tr>
<td>0.5</td>
<td>-0.01866</td>
<td>-1.05689</td>
<td>1.50762</td>
<td>4.227452</td>
<td>57.06477</td>
<td>0.72</td>
<td>116.007</td>
<td>112.4635</td>
<td>0.88</td>
<td>-145.011</td>
</tr>
</tbody>
</table>
[2.8.4] Calculation of confining fluid volume

Confining pressure is output as a pressure in p.s.i., so no further calculations are necessary (with the exception of conversion to MPa). In a drained experiment, when the sample deforms during loading, the oil intensifier piston moves, allowing a certain volume of oil to enter or leave the pressure vessel in order to maintain the constant confining fluid pressure that has been requested on the command signal. By monitoring the displacement of the intensifier piston during a test at constant confining pressure, the volumetric strain of a sample can be directly measured without the use of strain gauges. Alternatively, in an undrained test the piston of the intensifier is kept in a set position e.g. the original position required to achieve a given pressure. When the sample deforms, the pressure increase or decrease for a fixed volume of fluid can be measured.

In the work described in this thesis, the confining fluid was monitored solely under drained conditions. To achieve this, the amount of confining fluid required to enter or leave the pressure vessel (in order to maintain constant confining pressure) is calculated by the procedure below. Firstly it is not necessary to know the absolute volume of fluid involved but simply the relative change in volume. In order to calculate this it is necessary to refer to Figure: 2.7 which shows the oil reservoir in the confining pressure intensifier (shaded region). This region is cylindrical in shape, so its volume can be calculated by using the equation \( V = \pi r^2 h \) where \( V \) = volume, \( r \) = the radius, \( h \) = height. The absolute value of \( h \) is not known, but it is known how \( h \) is changing during the tests i.e. from the piston displacement monitored by a calibrated l.v.d.t.. We must therefore consider the change in volume of a cylindrical slice of confining fluid (see Figure: 2.11), where the change in height of the slice is the difference in the present piston position with respect to a reference position, i.e. the value at the start of the block. We assume during these calculations that the confining fluid is completely incompressible, and is inside a closed system i.e. there are no leaks. A measurement of volume change can be converted into a volumetric strain if divided by the initial volume of the cylindrical sample. Pore fluid volumometry could be carried out using a similar method.

[2.9] Test Procedure

[2.9.1] Introduction

In this section the test procedure employed in rock deformation tests will be explained. The aims of these tests were first to create a fault in a previously intact sample, and then to observe its subsequent development under the following boundary conditions: (i)
Figure 2.11 Diagram to illustrate the volume change of confining fluid required to maintain constant confining pressure. This is calculated by assuming a the fluid occupies a cylindrical slice of radius $R$ and height $H$ inside the oil reservoir. The height $H$ is the change in the position of the piston with respect to a reference value e.g. the initial value on loading.
increasing post failure strain \textit{i.e.} simulating increasing amounts of shear; (ii) increasing confining pressure \textit{i.e.} simulating increasing burial depths. Three sets of tests were completed, the testing procedures for which are described in the following sections. The results and their analysis are presented in chapters 3 and 4.

[2.9.2] Series 1a: increasing axial strain (at constant confining pressure)

This test procedure involved the application of confining pressure (hence hydrostatic load) of 34 MPa (5000 p.s.i.), then a period of equilibration before application of differential load. The tests discussed here were all carried out at a constant nominal strain rate of $5 \times 10^{-6}$/s (with the exception of run 6 which was at $2.5 \times 10^{-6}$/s). Such strain rates are slower than standard tests on smaller core (25 mm) mainly because of the larger sample size. The movement of the main ram was controlled under stroke (displacement) control and was programmed to reach a pre-determined position in the time calculated to maintain the chosen constant strain rate. The samples in this first suite of tests were all deformed dry (with no pore liquid present) at room temperature and room humidity. Bedding was parallel to $\sigma_1$. This series of tests were carried out to increasing amounts of axial strain at the constant strain rate chosen. All sample presented here underwent failure then increasing amounts of subsequent sliding. In this way I investigated the effects of increasing post failure strain on deformation structures produced. At the end of the experiment, the axial load was removed, and then the confining pressure was reduced and bled to air. Confining fluid was required to drain for a few hours before the sample could be carefully liberated from the apparatus. A single experiment typically took 2-3 days to carry out: 4 hours to load a sample and prepare the rig; 4-8 hours to carry out the test; and 3 hours to shut down, drain the confining fluid and unload the deformed sample.

[2.9.3] Series 1b: increasing axial strain (at constant confining pressure)

Series 1b was carried out in a similar way to series 1a, the main difference being that the samples had bedding lamination in a different orientation. Here bedding was perpendicular to sample long axis, whereas in series 1a the samples had bedding lamination parallel to length. Also the Series 1b experiments were designed to investigate the deformation structures formed very close to peak stress and failure stress, and with the that result one sample did not actually undergo macroscopic failure.
Table: 2.3 Summary of the test conditions for experiments.
Series 1a: increasing axial strain (constant confining pressure)

<table>
<thead>
<tr>
<th>Test name</th>
<th>Run5d</th>
<th>Run1d</th>
<th>Run6d</th>
<th>Run3d</th>
</tr>
</thead>
<tbody>
<tr>
<td>axial strain</td>
<td>4.22</td>
<td>6.34</td>
<td>6.50</td>
<td>9.16</td>
</tr>
<tr>
<td>strain rate</td>
<td>5x10E-06 /s</td>
<td>5x10E-06 /s</td>
<td>2.5x10E-06 /s</td>
<td>5x10E-06 /s</td>
</tr>
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<td>34.5 MPa</td>
<td>34.5 MPa</td>
<td>34.5 MPa</td>
</tr>
<tr>
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</table>

Table: 2.4 Summary of test conditions for experiments.
Series 1b: increasing axial strain (constant confining pressure)

<table>
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<tr>
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<th>Run15d</th>
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<td>5x10E-06 /s</td>
<td>5x10E-06 /s</td>
<td>5x10E-06 /s</td>
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<td>34.5 MPa</td>
<td>34.5 MPa</td>
<td>34.5 MPa</td>
<td>34.5 MPa</td>
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<tr>
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<td>0</td>
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</tr>
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Table: Summary of test conditions for experiments.

Series 2: increasing confining pressure (constant axial strain)

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<th>Run10d</th>
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<tr>
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<td>Perpendicular axis</td>
<td>Perpendicular axis</td>
<td>Perpendicular axis</td>
<td>Perpendicular axis</td>
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</tbody>
</table>
[2.9.4] Series 2: increasing confining pressure (at constant axial strain)

This test procedure involved applying a confining pressure, and then increasing the differential load. Confining pressure was kept constant for the duration of each individual test but was increased in different tests to simulate the effects of increasing burial depth. Tests were all carried out at an axial strain rate of $5 \times 10^{-6} \text{ s}^{-1}$, and the main ram was again driven under stroke control. The samples were all subjected to the same amount of total axial piston displacement, leading to similar amounts of total axial strain. The samples all underwent some form of failure, although the high confining pressure tests showed no release of audible acoustic energy at failure. Following a test the same unloading procedure was carried out, as detailed above in section [2.9.2].

[2.10] Summary

The main characteristics of the experimental equipment used to deform rock samples and monitor deformation characteristics has been described above. A full operation manual for the safe use of the large-capacity rig, written and upgraded throughout this project, is included in Appendix C. The test procedure and sample preparation have also been outlined. The next two chapters present results in the form of the data collected during the tests (Chapter 3), and the data collected through subsequent geological analysis following the end of a test (Chapter 4).
CHAPTER 3: ROCK DEFORMATION TESTS - MECHANICAL DATA

[3.1] Introduction

This chapter deals with the data collected during rock deformation tests. The deformation of the samples during the pre-, syn- and post-failure phases of rock mechanics tests is monitored by sets of transducers measuring load, displacement and pressure. Results from two sets of experiments are presented below: Series 1 - increasing amounts of axial strain at constant confining pressure; and Series 2 - increasing confining pressure for constant axial strain. Series 1 and 2 tests were both carried out using the large capacity deformation rig described in chapter 2.

The physical quantities measured directly during a test (axial load, axial ram displacement, confining pressure, confining intensifier piston displacement) are converted into standard rock deformation parameters as described in chapter 2, section [2.8]. The resulting independent, experimentally determined quantities are as follows: Axial strain (applied shortening); Axial stress (resulting variation in stress); Confining pressure (applied confinement); Volumetric strain (from volume change in fluid required to maintain a specified confining pressure).

First, the stress resulting from applied axial strain is plotted up as a function of strain, and then these graphs are characterised in terms of their mechanical nature with reference to Figure: 1.8 in chapter 1. The characteristic values are then plotted to look for trends in the deformation data. Results are then discussed with reference to underlying physical processes. Following this, confining fluid volumometry data is presented, characterised, and interpreted with reference to the synoptic volumetric strain curve shown in Figure: 1.9. Finally the main conclusions of the chapter are presented.

[3.2] Results: Stress versus Strain

[3.2.1] Introduction

The standard way to present deformation data is to present pre-, syn- and post-failure data together as one plot of axial stress versus axial strain. According to Paterson (1978), this may be misleading after macroscopic failure has taken place, as the sample is disintegrating and has macroscopically heterogeneous stress and strain fields. The stresses in different regions of the sample can be very different so the values of stress and strain plotted are only meaningful as sample averages. Strictly, due to sample deformation, only load versus
displacement values should actually be used. The load supporting area will tend to decrease if sliding occurs along a single macrofracture hence the stress calculated, assuming initial cross sectional area, will underestimate the actual stress as deformation proceeds. However, one good reason to use stress and strain plots is for normalising load and displacement to compare samples of different sizes (e.g. those deformed using other apparatus) with those presented here. The complete loading curves for experimental tests are therefore presented below as axial stress versus axial strain plots, noting the reservations above. The test series 1a, 1b and 2 are treated separately in sections 3.2, with the general features being described first, then the departures from this general behaviour being documented for each individual test. NB: A correction equal to $\sigma_3$ should be added to all plotted values of $\sigma_1$ and $(\sigma_1 - \sigma_3)$.

3.2.2 Series 1a: Increasing axial strain tests (bedding parallel to length)
Figures: 3.1a - 3.1d show stress strain plots for test series 1a, where the effects of increasing axial strain are investigated for constant confining pressure on samples with bedding parallel to the sample axis. The graphs are ordered in increasing amounts of final axial strain and the main features of each graph are described below. The test conditions of each run along with the test identification (i.d.) are presented in Chapter 2 table: 2.3.

3.2.3 General features of Series 1a loading curves (bedding parallel to length)
Several key features are exhibited by all the tests. An initial period of apparent linear elastic behaviour is shown by the straight part of the curve corresponding to phase II in Figure: 1.8 in chapter 1. This is followed by phase a where the graph curves over, exhibiting strain hardening behaviour correlating to phase III in Figure: 1.8. This is permanent irrecoverable damage, and is thought to be due to inelastic processes. The stress-strain curve attains a maximum value known as peak stress (or ultimate strength) of approximately 120 MPa, which is followed by a short period of strain softening (where the graph curves back down), equivalent to phase IV of Figure: 1.8. A measurable (and often audible) sudden decrease in the value of stress then signifies dynamic failure. Behaviour in the post failure stage is usually characterised by a fairly constant value of axial stress (compare with phase V Figure:1.8), consistent with quasi-static frictional sliding.

3.2.4 Departures from generalised behaviour for individual tests
Run5d, 4.22% final axial strain (Figure: 3.1a). A sharp stress drop is not observed at dynamic failure in this case, and the graph curves down gently to a lower value of stress.
Chapter 3 - Rock Deformation Tests: Mechanical Data

After this failure, a small stress drop is observed after a short period of constant stress. The behaviour is then more or less constant for the extent of inferred frictional sliding with a slight decrease in values of stress towards the end.

Run1d, 6.04% final axial strain (Figure: 3.1b). In this test, dynamic sample failure occurs at approximately 2% axial strain. The sample undergoes a short phase of strain hardening during the post failure stage shown by the increase in the stress curve. This is followed by a phase of constant stress with increasing strain, again consistent with frictional sliding.

Run6d, 6.50% final axial strain (Figure: 3.1c). The first part of initial loading shows a short concave up phase, which is not observed in other tests, prior to the onset of linear behaviour. The stress drop due to dynamic failure occurs immediately after peak stress, without the strain softening phase commonly seen. Immediately after dynamic failure there is a period of strain hardening, and then small fluctuations are observed on the post failure sliding curve. The main fluctuations happen at approximately 50 s intervals between which behaviour is fairly steady. There is a gradual increase in stress from 82 MPa to 89 MPa during the post failure period.

Run3d, 9.16% final axial strain (Figure:3.1d). Only slight strain softening is observed after peak stress, and before stress drop at dynamic failure. A slight increase in the curve immediately after failure, implying strain hardening, is followed by more or less constant stress in the frictional sliding phase. The post failure part of the curve is punctuated by small stress drops. Overall there is a gradual decrease in post-failure frictional stress with increasing amounts of strain i.e. slight strain softening occurs.

[3.2.5] Series 1b: Increasing axial strain tests (bedding perpendicular to length)
Figures: 3.2a-f are stress-strain plots for the Series 1b tests. The Series 1b tests, similarly to those in Series 1a, are designed to investigate the effects of increasing axial strain at constant confining pressure. These tests differ only in the orientation of bedding in the samples. In Series 1b the bedding is perpendicular to sample axis whereas in Series 1a it was parallel to the sample axis. The test conditions are presented in Chapter 2, table: 2.4.
Figure: 3.1 Graphs of axial stress versus axial strain for series 1a tests deformed at increasing applied axial strain and constant confining pressure for: a) Run5d; b) Run1d; c) Run6d; d) Run3d.
Figure: 3.1 (continued)
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[3.2.6] General features of Series lb loading curves (bedding perpendicular to length)

The main features of Series lb loading curves are very similar to Series la. A very small amount of concave up behaviour (compare for phase I in Figure: 1.8) is observed prior to linear elastic behaviour (compare for phase II) which extends for the majority of the pre-failure phase. The graphs then curve over, marking the start of non-linear strain hardening behaviour (compare for phase III). Peak stress is reached shortly afterwards. The amount (if any) of strain softening between peak stress and dynamic failure is variable (phase IV). Dynamic failure itself is always denoted by a significant stress drop and the post failure stress remains roughly constant, with only small increases or decreases for increased axial strain.

[3.2.7] Departures from generalised behaviour

Run13d, 1.50% final axial strain (Figure: 3.2a). This test shows a pronounced concave upward rise at the initial period of loading. Some strain hardening is observed in the inelastic phase, following the phase of linear elastic loading. Final strain was attained before it is clear whether peak stress has been reached, but the final axial stress is comparable to the peak stress in the other tests. Dynamic failure is not encountered in this test.

Run14d 1.75% final axial strain (Figure: 3.2b). Inelastic behaviour at the start of the test is present but is not as obvious as in Run13d. Here, there is a short phase of strain softening following peak stress. The stress drop at dynamic failure is smaller than in the other tests in this series at approximately 10 MPa. Following failure, the stress continues to decrease, then starts to increase slightly in the short time before the termination of the test.

Run12d 2.00% final axial strain (Figure: 3.2c). In no other test is initial concave up inelastic phase so obvious or prolonged as in this test. There is a significant stress drop just at peak stress, with no phase of quasi-static strain softening observed between peak stress and dynamic failure. Strain hardening is observed immediately after the stress drop by a rise in the stress-strain curve which then levels off for a very short period of more or less constant stress before the test ends.
Figure 3.2 Graphs of axial stress versus axial strain for series 1b tests deformed at increasing applied axial strain and constant confining pressure for: a) Run13d; b) Run14d; c) Run12d; d) Run15d; e) Run16d; f) Run2d.
Run12d, 2% axial strain

Run15d, 3.0% axial strain

Figure: 3.2 (continued)
Figure: 3.2 (continued)
Chapter 3 - Rock Deformation Tests: Mechanical Data

Run15d 3.00% final axial strain (Figure: 3.2d). Strain hardening and softening surround peak stress, giving the graph a much more rounded appearance than in the earlier tests. After the main stress drop at dynamic failure, several smaller dynamic stress drops can be identified in the tail of the deformation curve as discrete jumps. These stress drops may be related to individual microstructural events. Apart from these jumps the stress remains essentially constant.

Run16d 8.00% final axial strain (Figure: 3.2e). In this case, no initial concave upward phase is identified. The onset of non-linear behaviour is difficult to distinguish from the linear phase, and there is only a small amount of strain hardening observed before macroscopic failure. After failure, the stress remains constant.

Run2d 11.0% final axial strain (Figure: 3.2f). No initial concave behaviour is observed. Linear initial loading is followed by a strain hardening phase then peak stress. After peak stress there is a very short amount of strain softening before the stress drop signifies dynamic failure. No strain hardening is observed immediately after failure. The post failure sliding curve decreases in value of stress at a fairly constant rate and shows fluctuations (mainly small stress drops) as the test continues. The main characteristic here is the proximity of peak stress and failure stress with no strain softening.

[3.2.8] Series 2: Increasing confining pressure tests (bedding perpendicular to length)
Figures: 3.3a - 3.3e are stress strain plots for Series 2 deformation tests. These tests were carried out in order to investigate the effects of increasing confining pressure on deformation characteristics. In Series 2, the samples have bedding perpendicular to the sample axis (as in Series 1b). The test conditions for each run are described in table: 2.5 of Chapter 2.

[3.2.9] General characteristics of loading curve
An upward concave initial phase lasting for only 0.1% axial strain is observed which compares to phase I of Figure: 1.8. This is rapidly followed by linear curve implying linearly elastic behaviour (phase II of Figure: 1.8). The graph subsequently curves over before reaching peak stress (phase III of Figure: 1.8). A short phase of strain softening occurs after peak stress (phase IV of Figure: 1.8). Dynamic failure is then indicated by a
Figure: 3.3 Graphs of axial stress versus axial strain for series 2 tests deformed at a constant axial strain and increasing confining pressure for: a) Run7d; b) Run8d; c) Run9d; d) Run10d; and e) Run11d.
Run9d, 6000 psi (41.4 MPa) confining pressure

Run10d, 8000 psi (55.2 MPa) confining pressure

Figure: 3.3 (continued)
Run11d, 8000psi (55.2MPa) confining pressure

Figure: 3.3 (continued)
drop in the magnitude of axial stress with hardly any associated increase in axial strain. The
behaviour immediately following dynamic failure varies from test to test and is treated
individually below.

[3.2.10] Departures from generalised behaviour

Run7d 2000 psi (13.8 MPa) confining pressure (Figure: 3.3a). In this test the stress drop at
failure is fairly large. Immediately after dynamic failure there is a period of strain
hardening, indicated by an increase in the magnitude of axial stress. Following failure, four
separate stages of sliding can be distinguished, each phase ending with a small dynamic
stress drop, and the next starting with a small amount of strain hardening (the final step has
the greatest amount of strain hardening). During these steps, the stress decreases slightly,
suggesting strain softening. The stress drop in each stage is larger than the stress recovered
through strain hardening in the subsequent step, so the overall trend is a decrease in stress.

Run8d 4000 psi (27.6 MPa) confining pressure (Figure: 3.3b). The dynamic stress drop
here is larger in this test than in any others. An increase in stress following dynamic failure,
begins as a logarithmic increase, and then continues following a linear trend. Post failure
peak stress is reached, and then stress decreases, before a small stress drop gives way to a
very gradual decrease in stress continuing until the end of the test.

Run9d 6000 psi (41.4 MPa) confining pressure (Figure: 3.3c). The dynamic stress drop at
failure is significantly smaller in magnitude than in both the lower confining pressure tests
presented above (run7d and run8d). After dynamic failure, fluctuations including several
subsequent small stress drops and subsequent stress increases are observed, but behaviour is
fairly steady until the end.

Run10d 8000 psi (55.2 MPa) confining pressure (Figure: 3.3d). The first departure from
linear behaviour, marked by the graph curving off, is less obvious here than in previous
tests. Peak stress is followed almost immediately by a very small stress drop. This leads to
post failure behaviour which is characterised by a gradual strain softening flattening off to
essentially constant stress.

Run11d 8000 psi (55.2 MPa) confining pressure (Figure: 3.3e). The initial curve here is
linear with no concave upward initial part (in contrast to the other Series 2 tests). The strain
hardening phase prior to peak stress, similarly to run 10d, is less obvious than in other tests. There is then an unexpected instantaneous jump in stress during the strain hardening phase, then strain hardening resumes until peak stress is reached. Following peak stress, stress reduces but no major stress drop is identified. A linear decrease in stress is followed by a small stress drop. This stress drop is followed by a phase of strain hardening, and then a series of small fluctuations about a fairly constant mean value of stress.

[3.2.11] Summary of stress-strain curves

The general features of the synoptic stress strain curve in Figure: 1.8 are exhibited in all the test data presented. The presence or absence of phase I appears to be independent of loading conditions although more tests with bedding perpendicular to length show this phase than samples with bedding parallel implying it may be caused by variations in sample heterogeneity. The main influence on the style of the deformation curves appears to be confining pressure. All tests could broadly be described as undergoing macroscopically brittle failure, with the exception of run 13 which did not reach failure, and run 10 and run 11 which may be interpreted to show transitional behaviour i.e. between strictly brittle and ductile regimes. An effort is made to quantify the variations in stress strain curves in the next section.

[3.3] Quantitative analysis and characterisation of loading curves

[3.3.1] Introduction

An attempt was made to quantify the variations in stress-strain curves for different tests. In the previous section [3.2], each stress strain curve was qualitatively interpreted, to identify the major characteristic deformation phases shown in Figure: 1.8. Here the actual stress and strain values for the beginning and end of these distinct phases are determined. The aim was to quantify the main differences in measured deformation in terms of the conditions applied (increasing amounts of post failure strain, and increased confining pressure) and the starting material anisotropy. Bernabe and Brace (1990) [and also Glover et al. (1996)] quantified peak, yield and frictional stress for the triaxial deformation of Berea [Darley Dale] sandstone in a similar manner. The definitions of the main deformation stages, already presented in Chapter 1, section [1.4.5] are given to avoid any ambiguity, then the data derived from the stress strain curves is presented graphically. Finally a short summary is given of the main trends and possible mechanisms are suggested to account for the characteristics observed.
Definitions used to analyse stress strain graphs

Definitions of the main points punctuating a stress-strain curve are annotated on Figure: 1.8 in Chapter 1. Below are noted the criteria established to measure these points. The points are defined in terms of the stress e.g. yield stress, and the corresponding values of strain at these points are given similar titles e.g. yield strain.

- The *yield stress* is defined as the point in the stress strain curve where departure from linear elastic behaviour can first be detected. A first approximation to the yield point is identified from stress strain graphs using a steel rule to highlight the departure from linearity. (These graphs are all plotted up on the same scale, with maximum x-axis axial strain of 3%, to enable a direct comparison between samples). The stress-strain data up to this first approximation of yield point is then plotted, and a linear regression fitted to this portion of the data. The next few data points are added / removed until the best fit linear regression (R squared value in excess of 0.98) is achieved. The point following the final one in this linear portion is then quantitatively defined as the yield stress, where departure from linear elastic behaviour begins.

- **Peak stress** is simply the highest value of stress reached during a test.

- **Strain hardening** is the inelastic behaviour between the first departure from linearity (yield point) and peak stress, in which stress continues to rise with increasing strain but at a decreasing rate. **Strain softening** similarly is the inelastic behaviour observed between peak stress and dynamic failure stress. In this phase the stress decreases quasi-statically but eventually accelerates to a dynamic stress drop. It was attempted to quantify the amount of strain hardening and work softening, by noting the change in axial stress and axial strain during what has been interpreted as the hardening or softening phase.

- **Dynamic failure stress** is the point beyond peak stress and after the strain softening phase where a sharp almost instantaneous decrease in stress is observed, often accompanied by an audible acoustic emission.

- The **dynamic stress drop** is the amount by which the value of stress decreases suddenly at this dynamic failure point.

- **Residual stress** is the value of stress at the bottom of the dynamic stress drop i.e. the stress immediately after dynamic failure.

- **Frictional stress** is the level that stress reverts to after a transient relaxation (e.g. strain hardening or softening) following dynamic failure where stress values remain essentially
constant for a significant time. The mean value of stress is calculated from the entire post failure curve.

- *Post failure stress drops* are identified where there is a clear step decrease in stress which can be identified in the stress strain curve tail.
- *Final stress* is the value of stress at the end of the test.

Several quantities are calculated from the quantities above:

- Anelastic stress change for strain hardening = peak stress - yield stress
- Anelastic strain change for strain hardening = peak strain - yield strain
- Anelastic stress change for strain softening = peak stress - dynamic failure stress
- Anelastic strain change for strain softening = peak strain - dynamic failure strain
- Total elastic stress = yield stress
- Total elastic strain = yield strain
- Young’s modulus = stress (elastic) / strain (elastic)
- Total inelastic stress = |dynamic failure stress - yield stress |
- Total inelastic strain = |dynamic failure strain - yield stress |
- Range of frictional stress = residual stress - final stress

[3.3.3] Stress strain analysis - the observations

In this section I examine the effects of the different control parameters (i.e. confining pressure, axial strain and sample anisotropy) on the characteristics of the deformation curves. Stress-strain characteristics are tabulated in tables: 3.1, 3.2 and 3.3 and plotted graphically in Figures: 3.4-3.12. Most graphs are plotted as a function of confining pressure as this has been qualitatively determined (section [3.2.11]) to have the most significant influence on the stress strain curves. In Figures: 3.4-3.12, the results from series la tests (bedding parallel to length) are plotted using a solid diamond marker. Series 1b and series 2 tests (bedding perpendicular length) are plotted together using an open box marker.

Yield Stress

The value of yield stress is plotted as a function of confining pressure in Figure: 3.4a. Yield stress is fairly constant for all samples with bedding parallel length (series 1a), at a value of approximately 90 MPa for the axial stress and 50 MPa for the differential stress. Tests with bedding parallel (series 1a) and bedding perpendicular (series
## Table: 3.1 Series 1a tests. Stress-strain analysis data.

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## Table: 3.2 Series 1b tests. Stress-strain analysis data.

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| YIELD S1-S3 | 63.9941  | 73.65192 | 79.34365 | 67.5472  | 80.42416 |
| YIELD E  | 0.786261 | 0.916866 | 1.092489 | 1.120536 | 1.121375 |
| YIELD T  | 1440.47  | 1675.45  | 2001.82  | 2090.63  | 2283.47  |
| M VALUE (Y=MX+C) | 96.711  | 106.09  | 107.39  | 105.63  | 106.8   |
| C VALUE (Y=MX+C) | 3.457   | 5.0382  | 5.4645  | 5.2399  | 9.4138   |
| RSQUARED | 0.9989  | 0.9993  | 0.9992  | 0.9992  | 0.9989   |

| WORK HARDEN S1 | 20.09642 | 38.64632 | 41.07871 | 36.9212 | 29.88039 |
| WORK HARDEN E  | 0.41312  | 0.662553 | 0.659767 | 0.536895 | 0.429112 |
| WORK HARDEN T  | 715.85   | 1151.01 | 1899.38  | 908.08  | 745.89   |

| PEAK S1 (MPa) | 97.56789 | 139.5558 | 161.4707 | 159.2949 | 165.131 |
| PEAK S1-S3 (MPa) | 84.09052 | 112.3003 | 120.4287 | 104.4684 | 110.3024 |
| PEAK E (%)     | 1.199381 | 1.579418 | 1.752256 | 1.657432 | 1.640486 |
| PEAK T (S)     | 2156.32  | 2826.46  | 102.44  | 2998.71  | 3029.36  |

| WORK SOFTEN S1 | -1.64751 | -2.27693 | -6.18591 | -4.2698  | -2.12937 |
| WORK SOFTEN E  | 0.085951 | 0.032124 | 0.044074 | 0.008014 | 0.081291 |
| WORK SOFTEN T  | 151.37   | 81.84    | 102.11   | 30.7     | -2906.71 |

| SUM ELASTIC S1 | 77.47147 | 100.9095 | 120.392  | 122.3737 | 135.2506 |
| SUM ELASTIC E  | 0.786261 | 0.916866 | 1.092489 | 1.120536 | 1.121375 |
| SUM INELASTIC S1 | 18.44891 | 36.36939 | 34.8928  | 32.6314  | 27.75102 |
| SUM INELASTIC E | 0.509071 | 0.694676 | 0.703841 | 0.544909 | 0.510403 |

| FAIL S1       | 95.92038 | 137.2789 | 155.2848 | 155.0015 | 163.0017 |
| FAIL S1-S3    | 82.4409  | 110.0213 | 114.2428 | 100.1849 | 106.1752 |
| FAIL E        | 1.295331 | 1.611542 | 1.79633  | 1.665445 | 1.721778 |
| FAIL T        | 2307.69  | 2908.3   | 204.55   | 3029.41  | 122.65   |
| FAIL AUDIBLE? | Y (POP)  | Y (LOUD) | Y (SOFT) | N (7SOFT) | N         |

| S1 DROP AT FAIL | 36.64916 | 56.05393 | 21.66621 | 13.77836 | 5.206658 |
| POST FAIL S1    | 59.27122 | 81.22498 | 133.6186 | 141.2267 | 157.759  |
| POST FAIL S1-S3 | 45.79386 | 53.96528 | 92.57666 | 86.40023 | 102.9664 |
| POST FAIL E     | 1.303327 | 1.624927 | 1.82037  | 1.681472 | 1.809732 |
| POST FAIL T     | 2328.07  | 2918.52  | 214.76   | 10.44    | 275.84   |

| FRICT S1       | 53.94   | 87.5     | 123.39   | 137      | 146.52   |
| FRICT S1-S3    | 40.46   | 60.25    | 82.35    | 82.18    | 91.69    |
| FRICT S1 RANGE | 11.32269 | 2.106002 | 14.97512 | 2.727661 | 11.0041  |
| E FOR FRICT SLIDE | 6.536625 | 6.225318 | 6.032733 | 6.015368 | 5.923608 |

| FINAL S1      | 47.94853 | 83.33098 | 118.6435 | 138.4991 | 146.7909 |
| FINAL S1-S3   | 34.7117  | 56.07126 | 77.59513 | 83.67257 | 91.96442 |
| FINAL E       | 7.839952 | 7.850246 | 7.853103 | 7.98684  | 7.733431 |
| FINAL T       | 4632.97  | 4711.12  | 4708.4   | 4897.17  | 4897.06  |

| FRICT S1 RANGE | 11.32269 | 2.106002 | 14.97512 | 2.727661 | 11.0041  |
| # S1 DROPS     | 4        | 2        | 3        | 1        | 2        |

| YIELD / PEAK S1 | 0.794026 | 0.723076 | 0.745597 | 0.768221 | 0.81905  |
| YIELD / PEAK E  | 0.655556 | 0.580508 | 0.623476 | 0.676068 | 0.738424 |
Figure: 3.4a Graph of axial stress at the yield point (where behaviour first becomes non-linear) versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression for bedding perpendicular samples is also shown.

Figure: 3.4b Graph showing the value of applied axial strain at the yield point versus confining pressure. The linear regression is shown for the samples with bedding perpendicular length.
1b) have yield stress values concentrating in distinct clusters, with yield stresses significantly higher for samples with bedding perpendicular (series 1b) at values of approximately 105 MPa axial stress (70 MPa differential stress). This is most likely due to a systematic increase in strength of the sample with bedding perpendicular to the sample axis consistent with fracture blunting due to the perpendicular structure as opposed to enhanced fracturing parallel to planes of pre-existing weakness in the sample with bedding parallel length. Series 2 displays an approximately linear increase in yield stress as confining pressure is increased from 77.5 MPa (64 MPa differential) at 13.8 MPa (2000 psi) confinement to 135 MPa (80.4 MPa differential) at 55.2 MPa (8000 psi) confining pressure.

The amount of axial strain at the yield point (see Figure: 3.4b) is similar for series 1a and 1b, which cluster together at approximately 1%, so the axial yield strain, unlike the axial yield stress is not sensitive to the structural ‘grains’ of the material. In contrast, the axial strain for yield point in series 2 increases approximately linearly with increased confinement similar to the same graph for axial yield stress.

**Peak Stress (Ultimate Stress)**

The value of peak stress as a function of confining pressure is also of interest, and is plotted in Figure: 3.5a for series 1a, 1b and 2. The peak axial stresses for tests with bedding perpendicular to length (series 1b) cluster at a value of 135 MPa (100 MPa differential stress). This is approximately 15 MPa higher stress than for tests with bedding parallel length (series 1a). The results of Series 2 tests (bedding perpendicular to length) initially show a systematic increase in peak stress as a function of increasing confining pressure. However, this increase flattens off at higher confining pressures, to a value of about 160 MPa axial (105 MPa differential) stress, at confining pressures of 55.2 MPa (8000 psi). The value of applied axial strain, where stress reaches a maximum, is on average 1.8% for tests with bedding parallel length (series 1a) compared to approximately 1.5% for tests with bedding perpendicular length (series 1b tests) (see Figure: 3.5b). Again the axial stress for this condition is more sensitive to material anisotropy than axial strain. Series 2 (bedding perpendicular) shows a systematic increase in the peak strain value with increasing confining pressure up to a value of 41.4 MPa (6000 psi), and thereafter decrease with further increase in confining pressure.
Figure: 3.5a Graph of the value of peak stress versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown.

Figure: 3.5b Graph showing the value of applied axial strain at the peak stress versus confining pressure. (Markers are as above).
Dynamic Failure

The values of stress and the strain at dynamic failure are plotted against confining pressure in Figures: 3.6a and 3.6b respectively. The dynamic failure stress shows the same general trends with respect to confining pressure as the yield point and peak stress: it increases systematically with increased confining pressure in series 2, and is significantly greater for bedding perpendicular than bedding parallel length tests. The axial strain at failure increases with increased confining pressure with the exception of the value at $\sigma_3 = 55.2$ MPa (8000 psi). Although the axial strain at failure is greater for bedding parallel to length tests (1.8%) than where bedding is perpendicular (1.59%), there is a spread of data and the two clusters overlap, so again the strain data is less sensitive to material anisotropy than the stress data.

Stress Drop

Figure: 3.7 shows the measured stress drop at dynamic failure. The data show a significant scatter but a general decrease with increased confining pressure. The amount of scatter in stress drop in different tests is best shown by the spread of data from series 1a and series 1b from 9 MPa in run5d to 33 MPa in run6d. There appear to be no systematic differences in the values of stress drop as a function of bedding orientation, because series 1a and 1b data overlap across their entire range. In series 2 (confining pressure) tests, the largest stress drop (of 55 MPa) occurs at the intermediate pressure of 27.6 MPa (4000 psi) and the smallest stress drop 15 MPa is observed at highest confining pressure. This overall trend of decreasing stress drop with respect to confining pressure can be fit by a linear regression with the equation $y = -0.82x + 53.29$. This trend is consistent, if extrapolated, with a stress drop of zero at $\sigma_3 = 65$ MPa, which defines the brittle ductile transition.

Young's Modulus

Young's modulus, equivalent to the slope of the linear-elastic part of the stress strain curve, is found to have a value of approximately 100 MPa for samples with bedding perpendicular to sample axis irrespective of the confining pressure conditions (Figure: 3.8). The value of Young's modulus is systematically lower for samples with bedding parallel to length, having a value of approximately 85 MPa. This is consistent with samples with bedding parallel to length having lower axial strength.
Figure: 3.6a Graph of the value of stress at dynamic failure versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression is shown for samples with bedding perpendicular to length.

Figure: 3.6b Graph showing the value of applied axial strain at dynamic failure versus confining pressure. (Markers are as above).
Figure: 3.7 Stress drop at dynamic failure for tests with bedding parallel (closed boxes), and perpendicular (open boxes) to length.

Figure: 3.8 Slope of the linear elastic stage of loading (i.e. Young's Modulus) versus confining pressure. Samples with bedding parallel length (closed diamonds) and bedding perpendicular to length (open boxes) are shown.
Residual and Final Stresses

The stress immediately after dynamic failure (residual stress) is shown in Figure: 3.9a as a function of confining pressure. The data show the same trend as the dynamic failure stress data (Figure: 3.6a), i.e. a systematic increase with respect to $\sigma_3$. Similarly the final stress at the end of the test systematically increases with respect to $\sigma_3$ (Figure: 3.9b).

Comparison of relative amounts of elastic and total anelastic strain

The relative amounts of elastic and inelastic behaviour are plotted against each other in Figure: 3.10a. The stress change during the elastic period (equivalent to the yield stress already presented in Figure: 3.4a) increases with increasing confining pressure and is slightly greater for samples with bedding perpendicular to length. The stress change for the inelastic phase is similar for samples with bedding parallel and perpendicular to length, although the amount of axial strain over which inelastic processes act (see Figure: 3.10a) is greater for bedding parallel length samples. The absolute axial strain over which elastic processes act increases systematically with confining pressure (see Figure: 3.10b). In contrast, the axial strain over which inelastic processes occur is constant as a function of confining pressure. Rutter and Mainprice (1978) suggested that microcrack density changes rapidly with increasing strain in the pre-failure stage whereas microcrack density is fairly constant across an increment of strain in the post-failure range. This implies that for a constant amount of anelastic strain in the pre-failure stage, as observed in my tests (see Figure: 3.10b), a roughly constant microcrack density would be produced. This implies that a constant damage (e.g. critical microcrack density) is required to induce failure, regardless of confining pressure.

Anelastic stress and strain during the strain hardening phase

The inelastic stress and strain change can be further decomposed into components of strain hardening and strain softening. The stress increase during the strain hardening phase is plotted in Figure: 3.11a as a function of confining pressure. The stress change during strain hardening shows no distinct trend with increasing confining pressure. Samples with bedding parallel (series 1a) and perpendicular (series 1b) to the sample axis both show a similar amount of stress increase, of approximately 25 MPa during the strain hardening phase (Figure: 3.11a). The change in strain for the strain hardening phase is plotted as a function of confining pressure in Figure: 3.10b. There is no distinct change in the amount of axial strain over which strain hardening occurs with respect to increased confining pressure,
Figure: 3.9a  Graph of the residual stress immediately after dynamic failure versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression is shown for samples with bedding perpendicular to length.

Figure: 3.9b  Graph showing the value of final stress at the end of a test versus confining pressure. (Markers are as above). Linear regression is shown for samples with bedding perpendicular to length.
TOTAL STRESS CHANGE FOR ELASTIC AND INELASTIC PHASES

Figure: 3.10a Graph of the change in stress during elastic and inelastic phases versus confining pressure. Stress changes for elastic phase is shown by closed diamonds and open box markers (defined as in preceding graphs) and stress changes for the inelastic phase are marked by open circles (bedding perpendicular length) and closed triangles (bedding perpendicular). Linear regression is shown for samples with bedding perpendicular to length.

% STRAIN DURING ELASTIC AND INELASTIC PHASES

Figure: 3.10b Graph showing the amounts of axial strain during elastic and inelastic phases as a function of confining pressure. (Markers are as above). Linear regression is shown for samples with bedding perpendicular to length.
STRESS INCREASE DURING STRAIN HARDENING PHASE

Figure: 3.11a Graph of the change in stress during strain hardening phases versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression is shown for samples with bedding perpendicular to length.

TOTAL AXIAL STRAIN DURING STRAIN HARDENING PHASE

Figure: 3.11b Graph showing the amount of axial strain over which the strain hardening phase extends as a function of confining pressure. (Markers are as above). Linear regression is shown for samples with bedding perpendicular to length.
the average value being 0.5%. Samples with bedding parallel to length tend to have strain hardening phases with slightly larger amounts of axial strain (0.7%) compared to samples with bedding perpendicular to length (0.45%).

Anelastic stress and strain during the strain softening phase
The quasi-static stress drop during strain softening, plotted against confining pressure in Figure: 3.12a, is variable and shows no distinct trend with respect either to confining pressure or with the orientation of bedding in samples. The change in stress is much smaller in magnitude than that for the strain hardening phase. The axial strain over which the strain softening extends, plotted in Figure: 3.12b as a function of confining pressure, is also variable and no trends were identified. The magnitude of the changes in strain here is much smaller than those for strain hardening.

Sample Length
Not all of the samples had exactly the same length. In order to check that this had no significant effect on the results, Figure: 3.13 shows the differential stresses at failure as a function of sample length. The length of samples appears to have no significant effect on peak stress. At first sight there is a hint of a slight increase with increasing length, but this trend is an artefact that the majority of the high confining pressure tests, which give a higher peak stress, happen to have longer samples.

Ratio of Yield Stress and Strain to Peak Stress and Strain
The ratio of yield stress / peak stress has a remarkably constant value of 0.75 and shows essentially no influence of confining pressure or bedding orientation (see Figure: 3.14a). Similarly, the ratio of yield strain / peak strain has a value of approximately 0.67 in all tests (see Figure: 3.14b). This implies that the yield point and ultimate strength are strongly related and have a relationship which is not dependant on confining pressure or sample anisotropy.

Frictional Sliding Stresses
The post failure frictional stress is presented as a function of confining pressure in Figure: 3.15. Frictional stress increases in a fairly linear way with increased confining pressure and at intermediate confining pressures, values are of the same order as yield stress values in tests at corresponding confining pressures. The increase in friction stress with confining
Figure: 3.12a Graph of the change in stress during strain softening phases versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression is shown for samples with bedding perpendicular to length (perpendicular).

Figure: 3.12b Graph showing the amount of axial strain over which the strain hardening phase extends as a function of confining pressure. (Markers are as above). Linear regression is shown for samples with bedding perpendicular to length.
Figure: 3.13 Graph of the peak differential stress as a function of sample length. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown.
Figure: 3.14a Graph of the yield stress / peak stress ratio versus confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression is shown for samples with bedding perpendicular to length.

Figure: 3.14b Graph showing ratio of yield strain / peak strain as a function of confining pressure. Markers are as described above. Linear regression is shown for samples with bedding perpendicular to length.
Figure: 3.15 Graph of the post failure frictional sliding stress as a function of confining pressure. Series 1a samples with bedding parallel length (closed diamonds), and series 1b and 2 samples with bedding perpendicular to length (open boxes) are shown. Linear regression is shown for samples with bedding perpendicular to length.
pressure levels off slightly at the highest confining pressure. Samples with bedding perpendicular to length have a systematic higher frictional stress than for those with bedding parallel of the order of 15 MPa.

Range of frictional stress
The stress strain analysis above concentrates mainly on stress-strain characteristics as a function of confining pressure rather than applied axial strain. This is because increasing amounts of post failure strain will not affect the sample failure and will in fact only influence the stress strain characteristics calculated from the post failure data e.g. frictional sliding and final stress. The range of frictional stress in the post failure phase (i.e. residual stress - final stress) is plotted as a function of applied axial strain in Figure: 3.16a for the three test series. This gives an indication of the total stress drop in the quasi-static frictional sliding phase due to the observation that there is little, if any, strain hardening associated with these small stress drops. The observation that residual - final stress is positive implies that there is also an overall strain softening. For samples with bedding perpendicular to length, the range of frictional stress increases with increasing axial strain and can be fit by a regression line with the equation $y=6.53\ln(x)-2.614$. This implies that softening prevails throughout the post failure phase in these samples, although the magnitude of softening decreases towards high strains. This trend is not displayed in the samples with bedding parallel to length. Another interesting observation is that confining pressure has no systematic influence on the post failure frictional stress range which contrasts markedly with the major effect confining pressure has on the magnitude of the stress drop at dynamic failure (see Figure: 3.7).

Final stress
Figure: 3.16b is a graph of final differential stress as a function of increasing axial stress. The final stress values for samples with bedding perpendicular to length, show a very slight decrease with increasing strain which reflects the gradual strain softening throughout the post failure stage. Samples with bedding parallel to length show essentially constant values of final stress. The major influence on final stress values is confining pressure, reflecting the same trends as identified already in the yield stress, peak stress and frictional stress descriptions i.e. stress increases as a function of confining pressure.
Figure: 3.16a Graph showing range of frictional stress as a function of axial strain. Series 1a samples with bedding parallel length are shown as closed diamonds, Series 1b samples with bedding perpendicular to length are shown as open boxes and series 2 samples are shown as open triangles. A logarithmic regression is fit to series 1b samples only.

Figure: 3.16b Graph of final differential stress as a function of applied axial strain. Markers are as described above.
Summary of main stress strain characteristics and underlying physical processes

In this section the main characteristics of the results are summarised, and interpretations are made in terms of physical processes which could be responsible for the specific features displayed on the stress-strain curves.

A small amount of strain hardening and strain softening is observed near dynamic failure but basically the deformation is still defined as macroscopically brittle. The exception are tests at high confining pressure (55.2 MPa) which start to depart from the recognised brittle mode, and are most likely the expression of a transitional regime between macroscopically brittle and macroscopically ductile end members. There is a marked systematic increase in strength shown by the yield stress, peak stress and dynamic failure stress with increasing confining pressure until 55.2 MPa (8000 psi) is reached. At 55.2 MPa this increase ceases and the graph levels off to display comparable values of stress for tests conducted at confining pressures of both 41.4 MPa (6000 psi) and 55.2 MPa (8000 psi). The observed increase in yield stress and the larger increase in peak stress with confining pressure over a finite range agrees with the qualitative observations of Handin and Hager (1957) for a variety of rock types, and the quantitative study of Berea sandstone carried out by Bernabé and Brace (1990) [also Glover et al. (1996)]. Both studies also observe the flattening of the stress strain curve at high confining pressure (seen here for the 8000 psi tests) which they interpret as the onset of macroscopically ductile behaviour.

The axial strain accommodated at yield, peak and failure stress systematically increases as a function of increasing confining pressure until at high confining pressure (55.2 MPa, 8000 psi) the values begin to decrease again. This reflects the greater amount of axial shortening required to achieve the higher stress necessary to reach yield, peak and failure at higher confining pressures (where the intrinsic rock properties and loading rate are the same).

The other major influence on strength seems to be bedding orientation. Samples with bedding perpendicular to length are observed to be approximately 20 MPa stronger than those with bedding parallel, as measured from values of yield, peak and failure stress. This is consistent with previous observations that sample anisotropy produces a strength anisotropy (e.g. Wong et al., 1997). The axial strain accommodated before yield point is reached is not influenced by bedding orientation, but the axial strain values at which peak stress and failure stress occur are noticeably larger for samples with bedding parallel to...
length. There is more strain change during the strain hardening phase for bedding parallel length samples than those with bedding perpendicular. This indicates that the amount of strain required to reach yield point is not affected by bedding orientation, but once inelastic processes start to take over, the structural properties of the rock have a measurable influence. No references were found to support this observation in the literature.

The stress/strain slope for the initial linear phase of the loading curve (which indicates Young’s modulus) is insensitive to confining pressure in this range, but is systematically steeper for samples with bedding perpendicular to sample axis than for samples with bedding parallel. This result agrees with tests on Rothbach sandstone, with bedding perpendicular and parallel to length, presented in Wong et al. (1997). This would be anticipated as this slope is an indication of the intrinsic anisotropic material properties of the rock (which depend amongst other things on heterogeneities) and does not depend on the loading conditions applied.

The magnitude of the stress drop at dynamic failure shows some scatter but decreases systematically with increased confining pressure. Sample bedding orientation appears to have little influence on the measured stress drop. A linear extrapolation of the results predicts the stress drop going to zero at confining pressure of approximately 65 MPa. Bernabe and Brace (1990) describe an absence of recognisable stress peak, no strain softening, and no dynamic stress drop to be evidence of macroscopically ductile deformation. Therefore a confining pressure of ~ 65 MPa at room temperature is interpreted as the brittle-ductile transition for Locharbriggs sandstone.

The changes in values of stress and strain during the periods of strain hardening and strain softening show no recognisable trends with increases in confining pressure. The strain hardening phase has larger magnitude changes in both stress and strain compared to the strain softening phase. The magnitudes of stress change are independent of bedding orientation although the samples with bedding parallel to length have larger amounts of strain accommodated during the strain hardening phase than samples with bedding perpendicular to length.

Sample properties appear to dominate the amount of strain accommodated during the inelastic phase. This may be due to the fact that the different bedding orientations will have different orientations of pre-existing flaws (assuming that pre-existing micro structures have a fixed orientation with respect to bedding). One series may have flaws in a similar orientation to maximum stress, whereas the other will have flaws in an opposing direction. This may result in different micromechanisms acting in the two sets of samples. The
samples with bedding parallel to $\sigma_1$ (commonly described as a direction of weakness) experience a longer period of inelastic processes than the "stronger" sample with bedding perpendicular. A possible reason is that the stress is lower in the "weaker" sample tests, and therefore the inelastic processes necessary before dynamic sample failure (i.e. the growth of a critical density of microcracks) will take longer to occur.

A logarithmic increase in frictional stress in the post failure phase is observed with increasing axial strain in tests with bedding perpendicular to length, implying that strain softening continues throughout this phase but decreases in rate as strain increases. This effect was not observed in tests with bedding parallel to length, and no trend was observed in the frictional stress associated with an increase in confining pressure.

The first departure from linear elastic behaviour occurs at a remarkably consistent value ~ 3/4 of the peak stress value for all of tests carried out in this study. This ratio compares very favourably to that of 0.72 (+/- 0.07) I calculated from tabulated data obtained from the deformation of Berea sandstone in Bernabé and Brace (1990). Brace, Paudling and Scholz (1966) describe rock dilating and causing a reduction in axial modulus relative to that expected for linear elastic behaviour at approximately 1/2 peak stress. Paterson (1978) quotes a value of 1/3 to 2/3 for the onset of dilatancy from a review of various studies. The discrepancy in these Figures may be due to the difficulty in actually identifying the exact point of the first departure from linearity. Also workers generally use a combination of volumetric strain and acoustic emission data to define dilatancy then plot this value onto the stress strain curves whereas I have interpreted yield point from the stress strain curves alone.

In a similar way, the ratio of yield strain to peak strain has a constant value of ~ 2/3 irrespective of the sample anisotropy or confining pressure applied. No references quoting this ratio have been found in previous studies. Both these results indicate a very strong correlation between yield and ultimate strength.

[3.4] Confining fluid volumetry

[3.4.1] Introduction

This section presents results of the changes in sample volume, interpreted from monitoring confining fluid volume changes, during the deformation tests. This data gives important information about how the sample is behaving in response to applied loading at different stages of the test e.g. by changing shape, size or deforming internally. Confining fluid volumetry is the technique of monitoring the amount of confining fluid (surrounding the
sample in the pressure vessel), required to maintain constant confining pressure. It is assumed that any changes in the fluid volume are due solely to changes in the volume of the sample, hence the changing volume of the sample throughout the loading test is effectively being monitored. In these tests, the volume change measured, automatically takes account of the sample shortening because of the balanced ram design. Oil is displaced out of the pressure vessel at exactly the right rate to compensate for movement of the platen into the pressure vessel (chapter 2, section [2.6]). Volumetric strain is calculated from change in volume divided by initial sample volume. In the following sections, graphs of volumetric strain are presented as a function of axial strain. The behaviour observed is characterised for the different deformation test series in terms of the synoptic volumetric strain graph in Figure: 1.9 of Chapter 1, and then possible mechanisms to explain the main features, and any differences observed in different test series, are discussed.

[3.4.2] Volumetric strain and stress versus axial strain

Figures: 3.17 to 3.19 are a set of graphs showing change in volumetric strain (in %) versus applied axial strain (in %) for different tests. Stress-strain curves (initially shown in Figures: 3.1 - 3.3) are superimposed onto the volumetry graphs, in order to compare the main features. The change in volume is calculated with respect to the value at the start of the loading procedure - it is assumed that no significant sample volume changes occur in the period of hydrostatic loading before the differential loading begins. The volume change during hydrostatic loading cannot actually be measured with the present set-up, however, for similar sandstones (Kayenta and St Peters), Zhang et al. (1990) observed a porosity decrease and hence a volume decrease of approximately 2% on applying a hydrostatic load of 35 MPa. This is the confining pressure used in the majority of my tests, therefore the sample volume change in my tests during hydrostatic loading may be of this order. Although this assumption may induce an error in the absolute values, the relative values of volumetric strain will not be significantly affected. A positive change implies a decrease in sample volume i.e. compaction, and a negative change implies an increase i.e. dilatancy. This is consistent with the convention used for axial strain where positive strain equates to sample shortening.
Figure: 3.17a Volumetric strain (open boxes) and axial stress (solid boxes) as a function of axial strain. A distinct change of slope near yield point marks the onset of dilatancy. A slight volume increase occurs near failure which is followed by a gradual decrease in volume until the end.

Figure: 3.17b Volumetric strain (open boxes), as a function of axial strain. Axial stress (solid boxes) is also plotted for comparison. Volume curve departs from linear trend near yield point, minimum volume is reached just prior to yield stress, then volume increases near dynamic failure. Post failure the curve levels off.
Figure: 3.17c Volumetric strain and axial stress as a function of axial strain. Initial behaviour is similar to previous graphs shown, however volume increases (i.e. dilates) more near failure. The post failure volumetric curve shows fairly linear trend of dilation, punctuated by a small departure possibly linked to a small post failure stress drop.

Figure: 3.17d This volumetric strain curve shows similar initial behaviour, corresponding to the stress curve. Dilation near yield point is followed by a turning point and an increase in volume at failure. Post failure the behaviour is essentially constant then begins to show an increase at 5% axial strain.
Figure: 3.17e  Volumetric strain versus axial strain. Graph shows a comparison between the series 1a Curves show similar trends although absolute values are slightly variable. The main feature to note is that following initial compaction, the samples all show dilation. Post failure the curve is relatively flat implying constant volume which would be consistent with frictional sliding.
[3.4.3] General characteristics of volumetric strain curves
The general characteristics of volumetric curves shown in Figures: 3.17-3.19 are presented in this section. The initial behaviour, which is observed in all tests, consists of a linear decrease in sample volume, equivalent to phase II compaction in Figure: 1.9. The volumetric strain curve then departs from this linear behaviour, tends to flatten off and then shows a relative increase in volume with respect to that linear trend. This departure from linear elastic behaviour always occurs before peak stress has been reached, and most commonly near (or just before) the yield point in the stress strain curve. It can be correlated to the yield dilatancy phase III in Figure: 1.9. The curve then often rounds off to a local maximum followed by a sharp (but often small in magnitude) increase in volume which tends to correspond fairly well to the dynamic stress drop on the stress-strain curve defined as macroscopic failure (phase IV in Figure: 1.9). The behaviour described so far mirrors the stress strain curve quite closely with the initial linear behaviour giving way to inelastic processes and a dynamic event correlating to macroscopic failure. Characteristics of the volume change for the post failure period (phase V in Figure: 1.9) are more specific to individual tests themselves. They show either a longer term increase, a decrease or constant volume, or, a combination of all three and are described in the following sections with reference to Figures: 3.17a to 3.17e.

[3.4.4] Summary: Increasing axial strain, constant confining pressure (series 1a)
Figures: 3.17a-d show volumetric strain data as a function of axial strain for samples with bedding parallel to length deformed at constant confining pressure. Figure: 3.17e is a plot showing data from all tests in this series for ease of comparison. All these tests show the same pre-failure volume change characteristics of initial compaction giving way to dilatancy. Runs 1d, 3d, and 5d show comparable values of volumetric strain where the curve departs from linearity. However, after dynamic failure, tests are more variable: run3d and run6d show dilatancy; run1d shows continuing bulk compaction, be it at a slower rate than the original phase; run5d shows a very small change but has essentially constant volume.

[3.4.5] Summary: Increasing axial strain, constant confining pressure (series 1b)
Volumetric strain data for samples with bedding perpendicular to length and deformed at constant confining pressure are presented in Figures: 3.18a-f. Individual tests are compared
**Figure: 3.18a Volume strain (open boxes) versus applied axial strain for Run13d. Only the pre failure section is seen and it decreases in volume linearly as expected until 1.5% axial strain where the test stops. (Axial stress is shown by solid markers in all graphs.)**

**Figure: 3.18b Volume change versus applied axial strain for Run14d. The initial phase is linear until 1.1% axial strain. At this point, the curve shallows indicating dilation. Volume decreases throughout the test although at 1.1% axial strain.**
Figure: 3.18c Volume change versus applied axial strain for Run12d. Following an initial linear decrease, the volume rounds off at 1.4% axial strain prior to macroscopic failure. Volume dilates at failure. Subsequently volume appears to decrease again.

Figure: 3.18d Volume change versus applied axial strain for Run15d. Volume decrease throughout the test is observed except where the graph flattens off displaying an almost constant volume immediately before failure at 1.5% axial strain. After failure, the volume reduction resumes again in a linear way but has a slightly shallower slope than the pre failure curve (hence exhibiting dilation).
Figure: 3.18e Volume change versus applied axial strain for Run16d
Initially, volume decreases in a linear way and levels off to at 1.5% axial strain near failure. After failure the volume decrease continues in a linear way.

Figure: 3.18f Volume change versus applied axial strain for Run2d
This test shows very similar behaviour to Run15d with a reduction in volume throughout the test. The slope of the initial part of the curve is greater than the later part (hence dilation is exhibited). A 'kink' at 1.5% axial strain, corresponding to sample failure separates the two portions.
Figure: 3.18g Graph showing comparison between volumetric strain for series 1b tests. Graphs highlight some variability but in general the same trends. All curves show volume decrease throughout but the slope of the curves does change at approximately 1.5% axial strain corresponding to the onset of dilation near the yield point.
to each other in Figure: 3.18g. One similarity in all these tests is that volumetric strain continues to increase throughout the entire test with the exception of a slight inflexion at or near macroscopic failure. The volumetric curve departs from a linear compaction trend at approximately 1.5% axial strain and a volume strain of approximately 0.12% in all tests (run16d has a slightly greater volume decrease than the other tests). Although the volume curve continues to display absolute compaction throughout the whole test, the slope of the initial linear portion of the curve is always steeper than the slope of the post yield phase. This indicates a relative dilatancy with respect to the volume strain expected for linear elastic compaction. The absolute volume strain at the end of run2d is much smaller than the corresponding value in run16d, but the trend (i.e. compaction throughout at different rates) is the same. It was considered that a small leak in the confining pressure system could have worsened in the later tests following the high pressure tests, but the fact that run2 (the second test carried out) has the same trend as the later tests would tend to disprove this theory.

[3.4.6] Summary: Constant axial strain, increasing confining pressure (series 2)
Figure: 3.19a-e present volumetric strain data for tests investigating effects of confining pressure. Figures : 3.19 f and g show comparisons of different tests. The low confining pressure test has a small volume reduction during the initial phase with respect to other tests. Also for this test, the volume dilation at dynamic failure produces an absolute increase in volume with respect to the original volume which continues for the rest of the test. The medium confining pressure tests follow the characteristics of the low confining pressure test, although the initial volume compaction is greater and the dilation at and beyond dynamic failure becomes smaller with increased confining pressure. The two tests at high confining pressure show substantially different behaviour from the other tests with much larger values of bulk volume reduction. Figure: 3.19f shows the systematic decrease in the amount of dilatancy produced by an increase in confining pressure.

[3.4.7] Summary
The overall conclusion from the volumetric strain data presented above is that confining pressure has the most significant influence on sample volume change. At low confining pressure, dilatancy dominates whereas at higher pressures, compaction dominates. All the tests show a recognisable "yield" point where the volume curve departs from linear elastic behaviour, which correlates well with the yield stress point in the macroscopic stress-strain
Figure: 3.19a Volume strain (grey open boxes) and axial stress (solid boxes) versus axial strain. Initial behaviour is linear compaction, changing to dilation near the stress strain yield point. Significant dilation is observed at failure, then the rate of dilation slows post failure. The bulk volume shows an absolute increase during this test.

Figure: 3.19b Volumetric strain (and axial stress) versus axial strain (markers as above). Graph is qualitatively similar to run7. The rapid increase in volume coincides with failure, then a short phase of almost constant volume gives way to a fairly linear increase in volume. This sample also shows an absolute increase in volume with respect to the initial value.
Figure: 3.19c Volumetric strain versus axial strain for run9b. Behaviour is similar to that observed in the preceding two graphs with the exception of the initial phase being slightly convex down. Following failure, the sample volume increases fairly linearly although this tests shows an overall decrease in sample volume.

Figure: 3.19d Volume strain versus applied axial strain for Run10d. This test has a dramatically different volumetric strain curve to that in previous tests. Initially it shows a constant rate of compaction until the yield point at 1.5% axial strain where the graph flattens near sample failure. After this slight departure, the curve continues to show compaction at a slightly slower rate until the end of the test.
Runhld, 8000psi (55.2MPa) confining pressure

Figure: 3.19e Volumetric strain versus axial strain for run 11d. Similar to run10d, this graph shows an increase in volumetric strain and hence a reduction in volume throughout. The total decrease in volume is less than in run10d. Also there is no major change of slope corresponding to failure point and instead the graph curves gently indicating a gradual decrease in the rate of compaction.

Figure: 3.19f Graph of volume versus axial strain for tests at confining pressures of 2000, 4000 and 6000psi highlighting the similar trends in behaviour and the systematic reduction in amount of sample dilatancy with increasing confining pressure.
Figure: 3.19g Graph showing a comparison of volumetric strain for series 2 tests. The most significant feature is the difference between the 8000psi tests and the lower pressure tests. These high pressure tests show sample compaction throughout and an absolute decrease in volume. The lower pressure tests show both compaction and dilatancy, and either a slight increase or essentially constant volume in the post failure phase.
curve. Dynamic failure is also recognisable in most volume curves irrespective of confining pressure or bedding orientation.

[3.5] Micro - Mechanisms responsible for volume changes

[3.5.1] Introduction

In this section, possible explanations for the bulk sample volume changes observed in the previous sections are presented, with reference to micromechanisms that are potentially responsible.

[3.5.2] General features of the volumetric strain curve

The initial phase of linear decrease in volume can be explained by sample compaction in the axial direction. The change in volume resulting from a simple linear shortening i.e. where the sample is shortened by a constant rate, is given by the equation: \[ \Delta V = \pi \times R^2 \times \Delta H \]

(see Figure: 2.9 and section [2.7.4]). Any change in position of the piston and hence a change in \( H \), results in a linear volume decrease due to shortening at a constant rate. If there is no lateral change in sample radius and hence \( R^2 = \text{constant} \), the resultant \( \Delta V \) will also be a linear decrease in volume (i.e. an increase in volumetric strain). The first part of the curve is therefore most likely due to axial sample shortening alone. A possible micro-mechanical cause is the elastic closure of pre-existing pore space in the direction perpendicular to maximum compressive stress (e.g. Paterson (1978), Scholz (1990) and references therein). The implication being that if the sample was unloaded at any point during this linear curve, pores should re-open and the sample regain its initial volume. This is consistent with elastic compaction in phases I and II of the synoptic volumetric strain curve in Figure: 1.9.

The point where the volumetric strain graph first curves over, departing from the initial linear trend, signifies that some non-linear processes have begun. This corresponds to either an absolute dilating phase, or at least a relative decrease in compaction rates with respect to strain i.e. relative dilatancy. In all cases the graphs of volumetric strain as a function of axial strain show a sharper deviation from linearity than that seen at the yield point in the corresponding stress-strain curves. An explanation is that a combination of axial shortening (linear decrease in \( H \)) and radial expansion (non-linear increase in \( R^2 \) term) are now both contributing to the \( \Delta V \) term and hence the overall trend of the volumetric strain curve. Bulk sample dilation may be evidence for new microcracks opening up with
opening direction perpendicular to maximum stress resulting in radial expansion of the sample. This would concur with Brace et al.'s (1966) observations of open microcracks oriented parallel to maximum stress in deformed crystalline materials and similarly Kranz' (1979) SEM observations of axial oriented microcracks in pre-failure samples. Alternatively, soil mechanics studies e.g. Lambe and Whitman (1969) attribute dilatancy to changes in the packing of grains due to relative motion between them. Perhaps the most likely explanation of dilatancy in a porous sandstone however, suggested by Bernabé and Brace (1990), is a hybrid of the two ideas above, where microfracturing is concentrated in the (weaker) cement at grain boundaries, hence freeing grains to move about and ride over each other causing lateral expansion. Individual microcracks are thought to grow then become stable and stop propagating. This results in "crack hardening" where an increase in stress is required to initiate further microcracking (e.g. Brace et al. 1966). This is consistent with the occurrence of dilatancy in the strain hardening phase of stress strain curves (phase III of Figure: 1.8).

The stage where the compaction curve either flattens off or shows an increase in volume results from a lateral volume increase which either balances or exceeds the volume decrease due to axial shortening. This turning point tends to precede peak stress on the stress-strain curve. The micromechanisms responsible for this are most likely an accelerated increase in microcrack damage and hence radial expansion at constant shortening rate. According to the acoustic emission studies of e.g. Scholz (1968), Lockner and Byerlee (1979), microcracking starts to localise in this stage in anticipation of a through going macroscopic fracture.

At macroscopic failure, a dynamic increase in dilatancy is observed in most tests. This can be explained by the development of a through-going fault plane whose surfaces lose cohesion instantaneously at the point of failure and hence induce a short lived lateral expansion (e.g. Dunn et al. (1973)).

Following dynamic failure, the sign and magnitude of the subsequent sample volume change varies for different tests and is difficult to generalise except for the observation that in all tests (barring run10d and run11d), the failure process has the effect of inducing a volume increase (i.e. absolute dilation) or at least reducing the compaction rate observed at the beginning of the tests. This is most likely because after failure the sample has disintegrated to some extent and is in more than one piece. As the constant rate of shortening continues, it is easier for the individual pieces to slide over each other, resulting in lateral expansion or maintenance of a constant volume, than to deform internally by pore
closure and micro fracture. This observation is comparable to the so-called 'shear enhanced dilatancy' described by Wong et al. (1997).

3.5.3 The effects of confining pressure.

It has been established in the previous section that tests carried out at high confining pressure do not conform to the general behaviour observed in the other tests. This implies that confining pressure significantly influences volumetric strain. Figures 3.19f and g present volumetric strain data for the confining pressure test series (series 2). The amount of volume decrease (i.e. increase in volumetric strain) observed in the initial compaction phase is directly related to the applied confining pressure; low confining pressure tests having a small volume decrease; high confining pressure tests showing a large volume change. In contrast, confining pressure is inversely related to the change in volume seen at and beyond dynamic failure: low confining pressure tests have large volume increases at failure; high confining pressure tests have small volume increases at failure. This is analogous to the general negative correlation between stress drop magnitude and confining pressure observed in stress strain curves (see Figure: 3.7). In summary, the first order effect of increasing confining pressure is to inhibit lateral expansion of the sample. Similar observations have been made by e.g. Wong et al. (1997) and studies referenced therein.

The second order effect is that the confining pressure controls the mechanisms of failure. This is observed in both the stress-strain curves (see Figure: 3.3f) and in the volumetric strain data (see Figure: 3.19g). Volumetric strain curves for low confining pressure show absolute dilatancy in the strain hardening and post failure sliding phases, the latter often accompanied by smaller dynamic events possibly due to fault roughness. In contrast, volumetric strain curves for high confining pressure tests show compaction throughout, implying that the sample is deforming internally in such a way as to show little lateral expansion. It certainly does not show enough lateral expansion to compensate for the volume reduction through axial compression and in run10d the rate of compaction actually increases following macroscopic failure. Again these observations agree with those of Wong et al. (1997) which are interpreted as a transition to deformation by distributed 'cataclastic flow' leading to 'shear enhanced compaction' as opposed to the localised failure seen at low confining pressures.
[3.5.4] The effects of bedding orientation

Tests with bedding perpendicular to length all show a similar trend of overall decreasing volume (increase in volumetric strain) throughout the test. In contrast, samples with bedding parallel to length exhibit more variation in behaviour between pre and post failure sections (see Figures: 3.17e and 3.18f), often displaying an increase in volume or constant volume post failure. This indicates that samples with bedding parallel to length expand more laterally (effectively increasing the volume again) in the post failure period compared to samples with perpendicular bedding. This follows logically as it is easier for a sample to part along a pre-existing plane of weakness (i.e. bedding parallel to length) than to create a new parting in virgin rock. Another interesting point is that the samples with bedding parallel to length show significantly more scatter than those with bedding perpendicular to length. This may be because deformation is much more sensitive to sample heterogeneity trending parallel to the maximum stress axis so small differences between different samples will be exaggerated. In contrast, samples with the same extent of heterogeneity but oriented perpendicular to length will appear more homogeneous so variability between samples will be less obvious.

Run6d shows slightly different behaviour from the other samples in series 1a (Figure: 3.17d). This may be a consequence of the test having a slower strain rate, which may in turn lead to differences in the volumetric response of the sample to load. It is difficult, however, to draw a clear conclusion when only one test was run at this slow strain rate. An alternative explanation is that sample heterogeneities caused varying strength characteristics.

[3.5.5] A note of caution

It is noted that there was a small oil leak from the pressure vessel through the Connax connector associated with all tests. It is therefore possible that this oil leak affects the volumetric strain curves observed and that the effect of this leak may get worse both in the later tests and the tests run at high confining pressure. However, if the leak was worsening, I would expect the initial linear part of the curve to have a different (steeper) slope in later tests. This is not the case, because similar values of volume change (approximately -2000 to -3000 mm$^3$) are achieved at similar strain (1.5%) throughout the three series of tests. The continuing decrease in volume seen in the late series 1b tests could in principle also be symptomatic of an oil system leak. However, the same trends are seen in run2 (the second test run on the equipment), which imply this characteristic is a real feature depending on
bedding orientation. As in all deformation tests it is important to be aware of such potential problems, even if, as in this case, there appears to be no significant systematic effect on the data.

[3.6] Errors

In this section I discuss the sources of measuring uncertainty and the quantitative effect of any systematic errors on the major conclusions discussed above. First, errors in plotting up axial strain (i.e. the x-shift of the strain data along the axis due to the zero pot. having a slight offset) reach a maximum error of only $+/- 0.06524\%$, in general, errors are approximately $+/- 0.03\%$. The raw data is collected to 5 significant Figures, so there should be no significant numerical rounding error during the calculations carried out in the Excel Visual Basic program. The only significant error brought in will be in the assumptions made e.g. assumption of constant cross sectional area. This assumption is reasonable at least until macroscopic failure (Scott et al. 1994), and is unavoidable since sample diameter can not be directly monitored during a test at present.

Measurements of sample length are accurate to $+/- 0.25\text{ mm (0.1\%)}$, sample diameter measurements are accurate to $+/- 0.25\text{ mm (0.25\%)}$, so these are also effectively negligible. The stress-strain curve sometimes starts above the zero point. This systematic error is due to the actuator being brought down onto the main ram manually prior to the start of the loading block. A small $\sigma_1$ is applied to ensure that the actuator is in direct contact with the cap piece of the main ram. This discrepancy is therefore not an ‘error’, but the real value of the applied axial stress. Uncertainty in the characterisation of stress-strain curves e.g. interpretation of yield stress will induce further systematic errors, but these should be of the order of 10% or less, because of the objective methods derived to characterise the deformation curves. I have also shown that any leaks in the confining pressure system could result in an overestimation the sample volume change. A small leak has been noticed from a connax connector which is used to get the wires out of the pressure vessel. However, this leak is small and does not significantly affect the observed behaviour in a way that is internally consistent with this mechanism.

In summary, the results presented here are likely to be correct to an accuracy of $+/- 12\%$, the largest error coming from the estimation of yield points and the scatter in the volumometry plots (due to the servo-control on confining pressure).
[3.7] Conclusions
The experimental data recorded during deformation tests has been presented and interpreted in this chapter. The behaviour displayed in all tests, with the exception of tests carried out at the highest confining pressure, can be described as macroscopically brittle deformation with linear elastic deformation giving way to small amounts of inelastic strain hardening and softening prior to dynamic sample failure. The high confining pressure tests display behaviour commonly observed in the transitional regime between the brittle and ductile end members. Samples with bedding parallel to length are weaker than those with bedding perpendicular to length and have a smaller elastic modulus. Strength increases as a function of increased confining pressure but this trend flattens at high confining pressures. The magnitude of stress drop at failure is inversely linearly related to confining pressure and a brittle ductile transition value is predicted at 65 ± 15 MPa. Yield and ultimate stress are strongly correlated and are independent of applied confining pressure or sample anisotropy. Samples exhibit distinct phases of volume compaction and dilatancy which can be closely correlated to the main deformation phases interpreted from the stress strain data. Dilatancy decreases with increased confining pressure until at high pressures volume compaction dominates throughout loading. In summary the strongest control on deformation is confining pressure. The basic dynamic observations made during the deformation of my large scale samples correlate very well with results from previous rock deformation tests carried out on smaller sized samples. This is a very encouraging result as it implies that there are no anomalous effects due to an appreciable increase in sample size. My deformation tests can now therefore be used to investigate deformation structures at a larger scale compared to previous laboratory studies, with a view to extending the size range of laboratory studies and bringing them a little closer to the realm of real faults. The following chapter will introduce new structural features observed in these large scale samples which have not previously been observed in small scale laboratory samples.
CHAPTER 4: ROCK DEFORMATION TESTS - STRUCTURAL DATA

[4.1] Introduction
In this chapter I describe in detail the structural characteristics which make up the laboratory-induced fault zones produced in this work, and how these characteristics change systematically for increasing amounts of deformation. The primary structural data are collected following the end of deformation tests, and where possible are quantified in order to draw comparisons between different test samples. In all cases, a thorough but more qualitative geological description is given. The techniques used for this 'post-mortem' structural analysis include: hand specimen analysis; laser particle size analysis; thin section microstructural analysis (qualitative and quantitative); and Scanning Electron Microscopy. These methods are introduced in turn below. This structural data collected from laboratory test samples forms the basis for a comparison with those obtained from field outcrop studies presented and compared in chapter 5.

[4.2] Hand specimen analysis
The visual examination of deformed hand specimens is an elementary technique, requiring only a hand lens, yet yields a great deal of valuable information about the style and extent of damage. It can determine whether failure is localised or distributed, the spatial organisation of deformation structures, a qualitative measure of the density of damage, the orientation of structures with respect to the principal stresses, the spatial relation of structures to each other, and is the first stage in formulating more detailed or quantitative observations. The information gleaned from such an analysis is presented below (section [4.2.2] and [4.2.3]). First, the fault zones are exposed by horizontal sawcut (Figure: 4.1) with the minimum disturbance to the sample, then observations are made on these exposed sections. Finally the samples are disassembled into two halves split along the fault zone, to fully investigate the fault zone structure in three-dimensions.

[4.2.1] Exposure of fault zones
Following the end of a deformation test, samples deformed to increasing amounts of axial strain (Series 1a and 1b) and confining pressure (Series 2) were carefully removed from the loading apparatus for structural analysis. Initially, samples were kept in their rubber jackets to minimise disturbance of the fault zone structures. The major characteristics of the fault zone could be detected through the opaque rubber jackets e.g. a visible offset and stretching
Figure 4.1 Schematic diagram of cylindrical sample showing the orientation of cutting used to expose radial cross sections for visual analysis. The orientation of the fault zones and induced microcracks are shown as well as the sample axis which all orientations are measured from.
Chapter 4 - Rock Deformation Tests: Structural Data

of the rubber jacket implying an offset between the hanging wall and the footwall. All samples in Series 1a and 1b showed macroscopic shear failure whereas samples in Series 2 showed a much more complex deformation, characterised by either a shear fracture, sample bulging or a combination of both.

In order to expose the fault zone, samples still in their rubber jackets, were cut in half radially at room temperature and ambient humidity using a hand saw (see Figure: 4.1). The cutting procedure was carried out dry to prevent fine gouge material being washed away. Care was taken during this procedure to avoid applying excessive torque which could snap the sample along an incipient fault. The samples were cut at approximately the middle point of the fault zone, then the exposed radial sections were photographed and measured before one half of each sample was disassembled for laser particle sizing and the other half kept intact for epoxy impregnation and thin sectioning. It is possible that the unloading and cutting procedure damages the sample or alters the orientation of deformation structures, however, I assumed that this disturbance caused very little damage in comparison to that induced during the deformation test. I estimate that only a few degrees of rotation would be expected in the microstructures and in any case, rotations would presumably be random and should therefore not dominate trends.

[4.2.2] Description of fault zones: Series 1a and Series 1b

Figure: 4.2a-j is a set of photographs showing radial cross sections of samples deformed by increasing amounts of axial strain. The plane of the photographs are perpendicular to maximum compressive stress and parallel to confining pressure. Figure: 4.2a, b, c, d, e, f, g, h, i, j. show comparable radial cross sections for samples which have undergone final axial strains of 1.5%, 1.75%, 2%, 3%, 4.2%, 6%, 6.5%, 8%, 9%, 11%, respectively. The samples shown all underwent macroscopic failure, with the exception of test run13 shown in Figure: 4.2a, which was halted immediately prior to failure in order to investigate the failure process. All tests were carried out at axial strain rate of 5x10E-6 /s with the exception of run6 (Figure: 4.2g) which was carried out at 2.5x10E-2.5/s in order to investigate the effects of varying strain rate. All samples which failed, did so along one macroscopic shear, as opposed to a conjugate set. Conjugate sets of faults are commonly observed in experiments approaching the brittle-ductile transition, or experiments when localisation along one plane is inhibited by laterally constrained end-pieces forcing the development of two or more shears (e.g. Paterson, 1978; Scott et al., 1994). The deformation in my tests is expressed as a series of distinct fault strands which form a zone of deformation. The individual fault
Figure 4.2 Photographs showing radial cross sections of Locharbriggs sandstone samples deformed to different amounts of final axial strain in series 1a and 1b tests. Samples are 100mm in diameter, the maximum compressive stress is perpendicular to the photograph, and confining pressure is applied round the circumference of the sample, in a plane parallel to the photograph. Samples are presented in order of increasing values of applied axial strain: a) 1.5% (run13); b) 1.75% (run14); c) 2.0% (run12); d) 3.0% (run15); e) 4.2% (run5); f) 6.0% (run1); g) 6.5% (run6); h) 8.0% (run16); i) 9.0% (run3); j) 11.2% (run2).
Figure: 4.2 (continued)
Figure: 4.2 (continued)
Figure: 4.2 (continued)
Figure: 4.2 (continued)
strands are pale in colour, less than 1 mm wide and interweave with each other. Between these strands, are lenses of apparently undamaged host rock caught up inside the fault zone. The fault zones are studied on more detail in the following sections.

**Number of Gouge Strands**

In an attempt to quantify the analysis of fault zones, the number of individual, discrete gouge strands making up a fault zone were measured from the radial cross sections. A transect across the middle of the fault zone and perpendicular to it was taken, and every gouge strand which could be distinguished (using a x10 magnification hand lens and light source) was counted. Figure: 4.3 is a graph of number of distinct gouge strands as a function of final applied axial strain (%). Data is presented in table: 4.1. Error bounds show the systematic estimated error involved with the measurement as +2 strands. The systematic error is positive because the observation will, if anything, under-estimate the number of strands by interpreting two adjacent strands as a single one and will likely be strongly resolution dependent. The graph shows a good positive correlation between increasing numbers of strands and increasing amounts of applied axial strain. A linear trend with equation: \( y = 1.417x - 1.3307 \) fits the data with a correlation of \( R^2 = 0.9644 \).

**Width of Fault Zone**

The width of each fault zone was measured perpendicular to the trace of the fault and midway between the sample edges. The width was defined as the extent of significant visible damage i.e. pale coloured fault strands. In some samples the fault zone had a subsidiary branch extending out at a high angle to the main fault. This type of feature was not included in the “width of fault” measurement. Figure: 4.4 presents quantitative data on the scaling of fault “width” defined in this way with increasing deformation, collected from radial cross sections using a x10 magnification hand lens and steel rule (see table: 4.1). The graph of apparent fault width versus applied axial strain (see Figure: 4.4) shows a weak positive correlation of increasing width as a function of increasing applied axial strain. This trend is best displayed in samples deformed to greater axial strain. The linear trend with equation: \( y = 2.3522x + 3.0332 \) fits the data with the correlation factor \( R^2 = 0.6469 \).

**Description of fault traces and surfaces**

The samples were disassembled, by removing rubber jackets and splitting the samples along the fault zones, in order to analyse the nature of the fault zone surfaces and gouge material.
Figure: 4.3 Graph of number of distinct gouge strands versus final applied axial strain measured from the exposed radial hand specimen sections shown in figure: 4.2a-j. A hand lens and light source were used to improve accuracy. Estimated error bounds for the primary data are shown on the graph, together with the best fit straight line ($R^2=0.96$).

Figure: 4.4 Graph of apparent width of fault zone versus applied axial strain measured from the radial cross sections shown in figure: 4.2, using a x10 magnification hand lens and a steel rule. There is a linear correlation of increasing width with increasing applied axial strain best fitted by the solid straight line ($R^2=0.65$). Note the poorer fit compared to figure: 4.3.
Table: 4.1 Apparent number of gouge strands and width of fault zone, measured from radial sections in Figure: 4.2. Data is also graphed in Figure: 4.3 and 4.3.

<table>
<thead>
<tr>
<th>Strain (%)</th>
<th>Number of strands</th>
<th>Width of fault zone</th>
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</thead>
<tbody>
<tr>
<td>1.5</td>
<td>0</td>
<td>0 mm</td>
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<tr>
<td>1.75</td>
<td>1</td>
<td>2 mm</td>
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<td>10 mm</td>
</tr>
<tr>
<td>3</td>
<td>3</td>
<td>16 mm</td>
</tr>
<tr>
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<td>25 mm</td>
</tr>
<tr>
<td>6.3</td>
<td>6</td>
<td>10 mm</td>
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<td>12</td>
<td>24 mm</td>
</tr>
<tr>
<td>11.2</td>
<td>15</td>
<td>30 mm</td>
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</tbody>
</table>

Table: 4.2 Statistics from laser particle size analyser for: a) size data by volume and b) size data by number.

A) STATISTICS (ARITHMETIC) OF LASER SIZE DATA BY VOLUME

<table>
<thead>
<tr>
<th></th>
<th>PEAK</th>
<th>MEAN</th>
<th>MEDIAN</th>
<th>MODE</th>
<th>CONF LIMITS 95%</th>
<th>STDEV</th>
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</thead>
<tbody>
<tr>
<td>Undeformed</td>
<td>212</td>
<td>161.2</td>
<td>169.4</td>
<td>211.7</td>
<td>136.1</td>
<td>186.3</td>
</tr>
<tr>
<td>Run5d - strand</td>
<td>9.97</td>
<td>41.51</td>
<td>11.44</td>
<td>10.52</td>
<td>23.26</td>
<td>59.77</td>
</tr>
<tr>
<td>Run6d - strand</td>
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<td>28.67</td>
<td>9.7</td>
<td>10.52</td>
<td>19.32</td>
<td>38.01</td>
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<tr>
<td>Run2d - strand</td>
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<td>22.34</td>
<td>8.699</td>
<td>10.52</td>
<td>15.34</td>
<td>29.35</td>
</tr>
</tbody>
</table>

NB: All measurements shown in above table are in microns

B) STATISTICS OF LASER SIZE DATA BY NUMBER (CUMULATIVE)

<table>
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<tr>
<th></th>
<th>PEAK</th>
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<th>(D(10-200))</th>
<th>(D (&lt;200)^*)</th>
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<tr>
<td>Undeformed</td>
<td>200 microns</td>
<td>0 %</td>
<td>2.474</td>
<td>3.088</td>
<td>2.777</td>
</tr>
<tr>
<td>Run5d - pod</td>
<td>145 microns</td>
<td>4.2 %</td>
<td>2.54</td>
<td>3.484</td>
<td>3.021</td>
</tr>
<tr>
<td>Run6d - pod</td>
<td>9.97 microns</td>
<td>6.5 %</td>
<td>2.66</td>
<td>3.59</td>
<td>2.974</td>
</tr>
<tr>
<td>Run2d - pod</td>
<td>29.13 microns</td>
<td>11.2 %</td>
<td>2.685</td>
<td>3.467</td>
<td>3.045</td>
</tr>
<tr>
<td>Run5d - strand</td>
<td>9.97 microns</td>
<td>6.5 %</td>
<td>2.646</td>
<td>3.541</td>
<td>3.09</td>
</tr>
<tr>
<td>Run6d - strand</td>
<td>9.97 microns</td>
<td>11.2 %</td>
<td>2.726</td>
<td>3.834</td>
<td>3.193</td>
</tr>
<tr>
<td>Run2d - strand</td>
<td>9.97 microns</td>
<td>11.2 %</td>
<td>2.726</td>
<td>3.834</td>
<td>3.193</td>
</tr>
</tbody>
</table>

\(\)\(^*\) = size range over which power law exponent is calculated (in microns)
coating them as well as to investigate structural evidence for slip. Exposed fault surfaces are generally covered with large amounts of fine grained pale gouge material. Linear striation features are observed on several of the hand specimen gouge covered fault planes and are often easily identified due to their dark brown colour (see Figure: 4.5). The striation features trend down the direction of maximum dip of the fault implying a pure dip slip sense of movement. These features strongly resemble the lineations commonly observed on the “slickensides” (smooth, shiny, parting surfaces) which often occur on natural faults in the field, although the lineations in the laboratory have no polished shiny surfaces. Several surfaces have abundant slip features and sometimes more than one such surface exists per sample. It is unclear whether the striations occur on each individual strand making up a fault zone, or are concentrated on just one or two isolated strands within a fault zone. Test samples 1 and 2 display clear striations on two strands located at the outside edges of the fault zone and no visible striations on other strands inside the fault zone. This may indicate that these outer strands are undergoing a different slip mechanism to the inner strands and that they are the final strands to form. Paterson (1978) observed that slickensides are most common where large amounts of gouge are formed. This implies that a variability in the amount of gouge in individual strands may be a factor controlling the presence or absence of striations in my experiments. The amount of gouge in different strands cannot be determined uniquely through this type of analysis.

Undulations are observed in most fault strands in cross sections perpendicular to the shearing direction. The wavy nature of the fault trace, i.e. expression of the fault plane on the cut surface, is clearly seen in the radial cross sections shown in Figure: 4.2a-j. Wavelengths of undulations in the fault strand traces vary across a single strand and between different samples. An attempt was made to quantify this by measuring half wavelengths on the radial exposed surface using a steel rule. This yielded an estimated range of half wavelengths between 10 mm (56 host mean grain diameters) and 20 mm (112 mean grain diameters) and amplitudes of approximately 5 mm (28 mean grain diameters).

**Location of damage structures**

Figure: 4.6 is a schematic diagram showing the location of the macroscopic fault as observed in hand specimen for the tests carried out at constant confining pressure and increasing axial strain. It indicates that the majority of tests have single shear faults traversing the sample from the base corner to the side edge. The location of these structures may be a test artefact, due to the boundary conditions imposed by sample shape e.g. the
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Figure 4.5 Photograph showing a fault zone surface formed in the laboratory, with linear striation features trending from the top to the bottom of the figure, parallel to the maximum dip direction. These lineations are easily identified against the white coloured, fine grained gouge due to their dark brown coloration.
adjacent to the relevant sketch.
Loading orientations for series of samples. The applied axial strain and sample number are shown.

**Figure 4.6: Schematic diagram showing the location of fault zones with respect to sample edges and**

<table>
<thead>
<tr>
<th>Axial Strain (%)</th>
<th>Damage Location</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.5</td>
<td>Run 1d</td>
</tr>
<tr>
<td>1.75</td>
<td>Run 14d</td>
</tr>
<tr>
<td>2</td>
<td>Run 12a</td>
</tr>
<tr>
<td>3</td>
<td>Run 15</td>
</tr>
<tr>
<td>4.2</td>
<td>Run 6</td>
</tr>
<tr>
<td>6</td>
<td>Run 1</td>
</tr>
<tr>
<td>6.5</td>
<td>Run 16</td>
</tr>
<tr>
<td>8</td>
<td>Run 2</td>
</tr>
<tr>
<td>9</td>
<td>Run 3</td>
</tr>
<tr>
<td>11</td>
<td>Run 2</td>
</tr>
</tbody>
</table>
corners of a cylindrical sample are known to be sites of high stresses (see Figure: 2.4 this study, after Paterson, 1978). The bottom corner, where most faults terminate, is the region furthest away from the loading piston which implies that the direction of movement of the ram may influence fault localisation. It is possible that the top part of the sample is effectively fixed to the upper loading platen, due to friction, and they act as one. The only place where lateral movement, necessary for the initiation of a macroscopic fracture, would therefore be permitted, is this lower corner, because of the lubricated end piece arrangement (see Figure: 2.3).

Run6 and run12 do not conform to this pattern and instead have single shear faults which traverse from the top corner to side, and side to side, across the sample respectively. Sample 6 was deformed at a reduced strain rate and was observed, before loading, to have a small defect in its upper face. It is possible that the macroscopic fracture localised at this small defect or that the strain rate had some influence. However, sample 12a underwent deformation at the same strain rate as the majority of tests and had a different bedding orientation to sample 6. There appears to be no clear reason why the deformation in sample 12 should contrast with the rest of the tests and instead be similar to sample 6. It is of course possible, that both samples had material heterogeneities which were preferential sites of localisation regardless of the loading technique.

[4.2.3] Description of the fault zones: Series 2

Figure: 4.7a-e is a set of photographs of radial cross sections of samples deformed under differing confining pressure conditions. Figure: 4.7a, b, c, d, e, f show samples deformed at confining pressures of 13.5 MPa (2000 psi), 27.3 MPa (4000 psi), 41.1 MPa (6000 psi), 54.8 MPa (8000 psi) and 54.8 MPa (8000 psi) respectively. All tests were carried out at a strain rate of 5x10E-6 /s with a final applied axial strain of approximately 7.87%. Deformation is again expressed as a set of pale white fault strands, which anastamose with each other. Host rock lenses are again caught up in the fault zone. There is however, a striking systematic change in the spatial organisation of these fault strands as confining pressure is increased. The visible damage structures become significantly more radially distributed with increasing confining pressure. At low confining pressures the deformation appears to be localised along a single fracture plane, but at high confining pressure the deformation in the bulk sample is expressed no longer as a discrete plane, but as macroscopic bulging of a large proportion of the sample due to more distributed
Figure 4.7 Photographs of radial cross sections of samples deformed at different confining pressures, according to Series 2 test procedure detailed in chapter 2. The photographs are presented in order of increasing confining pressure: a) run 7 at 2000 psi (13.8 MPa); b) run 8 at 4000 psi (27.6 MPa); c) run 9 at 6000 psi (41.4 MPa); d) run 10 at 8000 psi (55.2 MPa); and e) run 11 at 8000 psi (55.2 MPa). Note the interweaving strands forming the fault zone extend more widely, and more symmetrically, across the sample than in figures 4.2. The damage becomes systematically more radially distributed as confining pressure is increased.
Figure: 4.7 (continued)
E

Figure: 4.7 (continued)
deformation. This observation is consistent with previous laboratory work observations on small sized samples e.g. Bernabé and Brace (1990), Wong et al. (1997).

Description of fault traces and surfaces

The fault surfaces here are more difficult to expose than in Series 1, due to the spatial complexity of the damage structures that resulted (Figure: 4.7a-f), hence not every sample can be studied. Sample Run8 is identified to have linear striation features highlighted by brown material, which have a fairly uniform orientation in the direction of maximum dip. Run 9 has similar features observed in a small section of the fault zone which is preserved, again brown material (thought to be hematite grain coatings smeared out) draws attention to them. No linear surface features are apparent on sample 11. Measurements of the waviness of the fault traces gave half wavelengths of 5 mm to 25 mm for sections of the fault observed in different regions of the same sample.

Other interesting features noticed in sample 8 are small steps in the fault surface, oriented perpendicular to the linear striations, and hence normal to the direction of shear. Figure: 4.8 is a photograph of these features (also sketched in Figure: 4.9) which are referred to as incongruous steps, after Norris and Barron (1969), Doblas et al. (1997), and consist of risers which face in the opposite direction to the movement of the hanging-wall. Such features, observed in the field, are used to determine the sense of slip and are commonly interpreted to be down-stepped (i.e. face) in the direction of movement of the hanging-wall (e.g. Hobbs et al. (1976), Scholz (1990)). If this criteria is used here, the sense of shear interpreted from the step features in my tests, is opposite to that actually observed from the macroscopic stress field (see Figure: 4.9). However, step structures created in the laboratory by Paterson (1958), Means (1976) are commonly stress risers which appear to oppose the direction of movement of the blocks as those found in my tests. They have been interpreted to be unloading release features, formed when the blocks move in the opposite sense (Paterson 1978, Norris and Barron 1969). More recently, Hancock (1985) observed, in the field, step features trending in the same orientation as my samples (see figure: 4.9) which he interpreted to be truncated Riedel shears and from which he would infer the same macro-stress orientation as observed in my tests.

Location of damage structures

The location of damage, in tests carried out at increasing confining pressure, are sketched in Figure: 4.10. In general, fault planes appear to initiate on the sides of the samples, and in one case at the base corner most distant from the loading piston. Otherwise there is no trend
Figure: 4.8 Photograph showing a laboratory formed fault zone with fine grained gouge material occurring on a fault plane. Also apparent are the step riser features oriented perpendicular to the direction of slip and facing upwards against the direction of movement.

Figure: 4.9 Schematic diagram showing: a) incongruous step features on a footwall fault surface. The arrows show the sense of shear due to axial compression. b) section view of an incongruous step feature highlighting the contradiction between the shear direction which would be interpreted from the feature itself (small arrows) and the actual sense of slip due to the applied stress conditions (large arrows).
The relevant confining pressure is noted adjacent to each individual sketch.

![Figure 4.10 Schematic diagram showing the location of damage in tests carried out at varying confining pressures](image)

<table>
<thead>
<tr>
<th>Confining Pressure (psi)</th>
<th>Damage Location</th>
<th>Test</th>
</tr>
</thead>
<tbody>
<tr>
<td>2000</td>
<td><img src="image" alt="Damage Location Sketch" /></td>
<td>Run 7</td>
</tr>
<tr>
<td>4000</td>
<td><img src="image" alt="Damage Location Sketch" /></td>
<td>Run 8</td>
</tr>
<tr>
<td>6000</td>
<td><img src="image" alt="Damage Location Sketch" /></td>
<td>Run 9</td>
</tr>
<tr>
<td>8000</td>
<td><img src="image" alt="Damage Location Sketch" /></td>
<td>Run 10</td>
</tr>
<tr>
<td>8000</td>
<td><img src="image" alt="Damage Location Sketch" /></td>
<td>Run 11</td>
</tr>
</tbody>
</table>
to the behaviour, except that the higher confining pressure tests i.e. at 6000 psi (41.4 MPa) and 8000 psi (55.2 MPa), deformation is expressed by a discrete section of the sample bulging and a section either at the top or the bottom of the sample which appears undamaged.

Sample 9 deformed at 41.4 MPa (6000 psi) shows deformation concentrated at the bottom 2/3 of the sample with the top end, near the loading ram, being apparently undamaged. In contrast, sample 11 deformed at 54.8 MPa (8000 psi) has damage concentrated at the top end of the sample near the loading actuator, with the bottom 1/3 of the sample seeming undamaged. The behaviour is so variable that it is difficult to interpret any systematic trend. The only clear trend is the way in which deformation changes from being expressed as either a discrete fracture at low confining pressure, to a gradual and more continuous bulging of the sample at high confining pressure, or at intermediate pressures, as some combination of both.

The spatial organisation of the visible damage changes systematically with confining pressure. The extent of deformation along the length of the sample, and the amount of radial expansion i.e. fault zone width or width of definable damage area, are both seen to vary. Deformation at low confining pressure occurs along a plane which is much narrower, but also may be longer than the deformation at high confining pressure. This, in contrast, is radially extensive but only appears to affect a certain portion of the sample and leave the rest intact.

[4.2.4] Summary

The most striking deformation feature, observed during the visual analysis of hand specimens, are pale interweaving strands of granulated material, which increase in number as a function of increasing strain. Similar features known as “deformation bands” have been observed in the field e.g. Aydin (1978), this study (Chapter 5), but their hierarchical development has never before been reproduced in the laboratory. These characteristic strands are observed to distribute the deformation more radially with increasing confining pressure, eventually producing web-like structures. This gradual switch from localisation on a plane to distributed damage as a function of confining pressure is well known for laboratory experiments on crystalline rock (Paterson 1958), and sandstone (Bernabé and Brace 1990), but the expression of the damage as the distinct pale granulated strands observed in this study is less common. Linear striation features, are observed on many gouge strand surfaces, in the direction of maximum dip. These features are common in field
faults and are observed in laboratory faults. Their presence here is evidence for shear movement along the bands implying that the strands are not purely compaction features and implies significant slip has taken place on these particular surfaces. This may be at the expense of slip happening elsewhere in the fault zone which leads to a comparison with the slip surfaces observed in the field studies of Antonellini et al. (1994).

[4.3] Particle Size Analysis
The particle size distribution of fault gouge is sensitive to the granulation processes producing it, and therefore may give an insight into the specific micro mechanisms which have been active during these deformation tests. For example Engelder, (1974) observed a systematic decrease in mean grain size and degree of sorting with increased sliding along a sawcut surface in Tennessee sandstone under laboratory compression. Therefore the relative amount of sliding along different faults in my samples may, in principle, be discernible from the particle size analysis. Specimens from the three distinct parts of the samples already recognised in hand specimen - i.e. fault gouge strands, host rock pods caught in the fault zone, and intact host rock - were analysed, with a view to comparing the relative types of deformation which could be responsible for any systematic change in the particle size distribution. This section describes the laser sizing technique used to determine the particle size distributions, the equipment used, and the resulting data (with supplementary material presented for completeness in Appendix D).

[4.3.1] Laser Particle Sizing Method
A Coulter LS100 Laser Particle Size Analyser was utilised to investigate grain size. This apparatus exploits the principle that the angle of diffraction of laser light is influenced by the size of particles in its path. A disaggregated sample, suspended in a distilled water solution, is passed between the laser source and detector (see Figure: 4.11a). The collimated laser beam is diffracted by each individual particle, by an amount related to the curvature and refractive index (R.I.) of that grain. If the grains are spherical (and have the same R.I.), the amount of diffraction from the axis, will therefore be a function of their radius (assuming Fraunhofer, or small angle, diffraction). Small particles cause higher angle diffractions from the axis than larger particles and hence the diffraction data may be used to calculate size distributions. The composite flux pattern produced due to many diffractions from different sized particles consist of a series of concentric rings which are decomposed using a Fourier lens shown in Figure: 4.11b. This lens focuses all the light arriving at a
Figure: 4.11 Schematic diagram of a laser particle size analyser: a) showing the diffraction sample cell where the sample in suspension passes in front of the laser. The location of the Fourier lenses and the set of detectors are also shown. b) The Fourier lens of a laser particle size analyser. This lens focuses all light arriving at a similar angle onto a single point on the detector screen, irrespective of the velocity or position of the particle causing the light to diffract.
certain angle (and hence from similar sized grains) into one position (or channel) on the
detector screen. The position and motion of the individual particles do not significantly
affect the diffractions as the Fourier lens focusing depends only on angle. The density of
light arriving in each channel i.e. the number of particles of a given radius, is measured as a
voltage at the detector and output as ‘binheight’. Appendix D presents details of the Coulter
LS100 Sizer specifications, sample preparation, measurement technique, and calculation of
grain size data.

[4.3.2] Presentation of results
First, results are presented as plots of volume (%) as a function of particle diameter (μm), in
order to highlight the grain diameter of the majority of the grains and the sorting. Data is
plotted on logarithmic-linear space to check for a log-normal relation, characterised by bell
shaped curves, which indicate a log-normal grain size distribution by volume. The peak of
the bell is the peak grain size (or mode) and the width of the shoulder indicate the degree of
sorting (poor sorting being implied by a wide spread). This data is subsequently plotted as
cumulative number (%) versus particle diameter (μm) on a log-log plot, to check for a power
law relation which would indicate a fractal size distribution. Fractal size distributions by
number have been observed in 2-d and 3-d studies of field gouge by Sammis et al. (1987),
and An and Sammis (1994), respectively, and also in simulated gouge generated in the
laboratory by Marone and Scholz (1989). As we shall see, such log-log plots can mask
some of the characteristic features of the gouge distribution that may be important in
determining the mechanical and hydraulic properties of such comminuted material.

Size distribution by volume
Figure: 4.12 is a set of graphs showing volume (%) as a function of grain diameter (μm), for
three samples deformed to different axial strains (4.2%, 6.5%, 11.2%), which compare the
grain size distributions in different regions of the deformed sample. The graphs show data
for specimens of (i) fault gouge (strands), (ii) host rock caught up in the fault zone (pods)
and (iii) undeformed Locharbriggs sandstone. The data for the undeformed host rock and the
gouge strand specimens are both characterised by approximate bell-shaped curves with main
peaks at 200 μm and 9.97 μm respectively, interpreted as the most frequent grain diameters.
These peaks roughly correlate to changes of slopes on all three curves, implying that these
characteristic grain sizes already existed before the experimental deformation. The gouge
strand curve is much wider than the undeformed host rock, indicating reduced sorting. The
Run 5d, 4.2% axial strain

Figure: 4.12 Laser particle size analyses graphs showing incremental total volume (%) as a function of particle diameter: a) shows data from run 5, deformed to 4.2% axial strain; b) presents data from run 6, which underwent 6.5% axial strain; and c) shows data from run 2 which underwent 11.2% axial strain. Specimens of (i) gouge strand material and (ii) the host rock pods caught up in the fault zone are plotted separately (see key), along with a control specimen (iii) from virgin Locharbriggs host rock which has undergone no laboratory deformation. The difference in grain size distributions for specimens taken from different areas of a sample are clear.
Run 2d, 11.2% axial strain

Figure: 4.12 (continued)
fault pods have variable peak grain sizes (145 μm, 9.97 μm, 9.97 μm) in the three samples (Figures: 4.12a, b, c, respectively) but a common feature to all, is the broad range of grain sizes and a skewed distribution implying a higher proportion of large particles. This is most likely due to the fact that this sample is a composite of the other two fractions.

Figure: 4.13 shows a comparison of the grain size distribution in a) gouge strands and b) fault zone pods, of samples 5, 6, and 2 subjected to 4.2%, 6.5% and 11.2% axial strain respectively. Grain size distribution data for undeformed Locharbriggs host rock is also plotted for comparison. Each gouge strand selected for study was randomly chosen from a separate deformed sample. The fine details of the curves (see Figure: 4.13a) may differ but the main characteristics are strikingly similar. The peak grain size of 9.97 μm measured in one gouge strand is very similar to that in a gouge strand from a different sample irrespective of the total axial strain applied to the bulk samples. The widths of the peaks in all three samples shown are very similar implying a similar degree of sorting. The fault zone pods (Figure: 4.13b) show variable grain size distributions, but all show fairly poor sorting due to the wide shoulders of the curve, and several curves are skewed towards larger grain sizes.

Figure: 4.14 is a cumulative frequency plot of the data presented above for gouge strands and host rock. It highlights the poor sorting in the gouge strands (indicated by a shallower slope) and the reduction in grain size (indicated by a shift left along x-axis) with respect to the host rock. This graph is included mainly for comparison with the thin section size analysis presented in section [4.4.10] and other studies which present size data in this manner.

The effects and implications of the sonicating process, used to disaggregate the specimens in the laser sizer sample reservoir immediately prior to grain size analysis, on the grain size distribution measured are considered in Appendix D.

Size distribution by number

The particle size distribution by number can be recovered from the size distribution by the volume divided by the mean volume in the sample bin i.e. \((\text{diameter}/2)^2\). Figure: 4.15 is a set of log-log plots of cumulative number or frequency (%) versus grain diameter (μm) for gouge strand material from three samples deformed to 4.2%, 6.5% and 11.2% (Figures: 4.15a, b, c respectively). The size distribution by number for all three gouge samples approximates a power law and is best fitted by two exponents. The apparent fractal
Figure: 4.13 Graphs of incremental total volume (%) versus grain diameter (μm) from the laser particle size analysis. These plots compare the grain size distributions in similar regions of tests carried out at increasing strain rather than comparing the different regions as in figure: 4.12. In both graphs (a and b), the grain size distribution for undeformed Locharbriggs sandstone is plotted for comparison. a) presents a comparison of the size distributions of gouge strands from deformation tests run2, run5 and run6. The three samples have very similar characteristics with curves which exhibit both the same mean value and a similar shape. b) shows a corresponding plot comparing the fault zone pod material generated in different deformation tests runs5, 1, 6, and 2. Here the grain size distributions are much more variable for different tests.
Figure: 4.14 Graph showing the cumulative frequency of grain size by volume for gouge strands from three tests deformed to the strains shown. The curve for undeformed host rock is also shown.
Figure 4.15 Graphs showing the cumulative particle size distribution by number in the form of number (%) versus grain diameter (microns), from laser size analysis on, log-log plot for: a) gouge from run5 (4.2% strain); b) gouge strands data from run6 (6.5% strain); and c) gouge strand data from run2 (11.2% strain). The data are best fit by 2 separate power law curves for the size ranges < and > 100mm. The exponents (equivalent to the apparent fractal dimension D) are displayed on the individual plots and in table: 4.1b.
C

Figure: 4.15 (continued)
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dimensions (negative power law exponents) for the size range <10 μm are a) 2.69, b) 2.65, c) 2.72, and for the size range >10 μm are a) 3.47, b) 3.45, c) 3.83.

The size distribution by number for the fault zone pods is presented in Figures: 4.16a, b, c for the same three test samples as shown in Figure: 4.15a, b, c. Again two power law slopes best fit the data and their exponents are presented as fractal dimensions in table: 4.2. Also in table 4.2, are exponents for the smallest size fraction (diameter <10 μm) of the host rock (see Figure: 4.17). The bump in the tail of the curve at grain sizes greater than approximately 40 μm indicates this size range cannot be fitted by a power law and hence is not a fractal distribution.

[4.3.3] Uncertainties in laser sizing technique

The main assumptions inherent in the algorithm used here to calculate size distributions from the scattering patterns (see Appendix D) were that all particles were spherical and mono-mineralic (quartz). In a granulated sample, where initially spherical grains have been broken into several angular fragments, the assumption of sphericity is not strictly correct, and would most likely result in an overestimation of the particle size due to i) surviving curved surfaces of the parent grains (which scatter the laser light by the same angle as for the parent, leading to similar grain size predictions for grains that are actually much smaller) and ii) the fairly straight, angular, broken edges which would scatter laser light like a very large sphere similarly overestimating grain size. In an intensely crushed zone, the assumption of sphericity would be more accurate as smaller particles became more equant due to several stages of fracture. Very small particles tend to flocculate (group) together, which would also lead to over-sizing, but this effect is avoidable by extra sonicating immediately prior to measurement (see Appendix D for details). I would expect few grains to be undersized, except perhaps, grains consisting of a mineral other than quartz. It is possible that the dissagregation process itself (detailed in Appendix D) may have an imprint on the observed size distribution, but this is difficult to estimate on “real” samples where the 3-dimensional size distribution prior to dissagregation is unknown.

[4.3.4] Summary of laser particle size analysis

Deformed samples have been laser size analysed to reveal the variation in grain size distribution in the different regions of the samples and for different amounts of applied axial strain. Gouge strands and host rock have particle size distributions by volume which can be approximated by log-normal size distributions. This agrees with previous
Figure: 4.16 a), b) and c) are the cumulative particle size by number plots for fault zone pods from run5 (4.2% axial strain); run6 (6.5%) axial strain; and run 2 (11.2%) axial strain respectively. The data are best fit by 2 separate power law curves for the size ranges < and > 10mm. The exponents (equivalent to the apparent fractal dimension D) are displayed on the individual plots and in table: 4.1b.
Figure: 4.16 (continued)

$y = 11.136x^{2.0613}$
$R^2 = 0.9991$

$y = 162.93x^{0.3609}$
$R^2 = 0.9899$

C

Figure: 4.17 Plot for the cumulative particle size by number for host rock. The data is fit by a power law for a limited size range only.
studies on natural fault gouge e.g. Sammis et al. (1986). This is not true however, of the fault zone pods which have a different size distribution, which is most likely because the samples are a combination of both gouge and host rock end-members. Gouge strands are observed to have both a reduced grain size, and reduced sorting, with respect to the undeformed host rock. A reduction in size implies reduced permeability (via Poisseuille’s flow mechanism presented in chapter 1, section [1.6.3]), but not necessarily a change in porosity if the shape and packing of grains remains constant. Reduced sorting however implies both a reduced porosity and reduced permeability as grains pack together more efficiently eliminating a great deal of the pore space and reducing pore throats (e.g. Guéguen and Palciauskas, 1994). Also, the peak grain size by volume, and degree of sorting, of the gouge strand samples is very similar irrespective of the total bulk strain applied to the individual samples. The gouge strand material particle size distribution by number is well approximated, for two discrete size ranges (<10 μm and >10 μm), by apparent fractal dimensions of D=2.7+/-0.05 and 3.6+/-0.25 respectively. An and Sammis (1994) calculated D=2.7 +/-0.2 for natural fault gouge over a wide range of grain sizes. Fault zone pod material and host rock can also be approximated by a fractal distribution (at least for the smallest size range), although this relation breaks down in the host material at grain sizes larger than 40 μm. The fractal dimension D, interpreted in the gouge strands is systematically higher than that in the pods and the host rock for the size range 0-10 μm. Figure: 4.18a is a graph of fractal dimension D as a function of the peak grain size of the samples (measured from the size by volume plots in Figure: 4.15, 4.16, 4.17) for gouge strands, fault zone pods and host rock. The two data sets presented indicate the exponent of the power law (i.e. D) fit over the size ranges 0-10 μm and 10-200 μm respectively. Both size ranges suggests a weak inverse correlation between D and peak grain size, such that the largest D values are obtained for the small grain sized samples i.e. the gouge strands. Figure: 4.18b compares the power law exponents describing the grain size distributions of strands, pods and host for the two size ranges considered and indicates that D(strands) > D(pods) > D(host). Figure: 4.18a, b, both reflect the fact that there are proportionally more small particles in the gouge strands than the host rock due to cataclastic grain size reduction and are consistent with the observations of Marone and Scholz (1989).
Figure 4.18 Graphs of apparent fractal dimension as a function of a) Peak grain diameter and b) size range over which exponent was measured. D (strands) > D (pods) > D (host) and D is greater in the larger size range.
[4.4] Thin Section Microstructural Analysis

[4.4.1] Introduction

Optical microscopy is a straightforward technique which can yield a great deal of information regarding deformation processes. Direct observations can be made of the fractures and flaws which may help explain the characteristic form of the macroscopic stress-strain curves observed during tests (presented in Chapter 3). Impregnation of thin sections by blue coloured epoxy resin, can significantly improve the visibility of fractures and pores, making quantitative studies possible. A microstructural analysis was therefore carried out on thin sections of deformed test samples using transmission optical microscopy. The aim of this analysis is to describe in detail, the deformation features present in order to ascertain the contrasts with the features formed under different test conditions. It was also hoped to offer possible explanations for processes involved in the formation and evolution of the larger scaled structures seen in hand specimens (see section 4.1), and to estimate the effects these structures may have on fluid flow and further deformation. First, the general structures observed in Series 1 (tests at increasing axial strain and constant confining pressure) are described, then the microstructures observed in Series 2 (tests at constant axial strain and increasing confining pressure) samples are presented. There then follows a detailed quantitative microstructural analysis of deformation features, pore space and grain scale microfractures.

[4.4.2] Method

Large-scale thin sections (75 mm x 50 mm) were cut from deformed samples in orientations parallel to the maximum compressive stress direction and in a plane perpendicular to the fault zone (see Figure: 4.19). This allowed analysis of damage inside the fault zone itself and as a function of distance from the fault zone in the host rock. Some thin sections were impregnated with blue epoxy resin to aid the identification and quantification of pore space. The thin sectioning technique and materials used are described for reference in Appendix: E. Initial reconnaissance surveys of thin sections were carried out using a Wild macroscope. The macroscope allowed lower magnification than a standard petrological microscope and hence a wider field of view. This was extremely useful in analysing fabrics and microstructural spatial relations. A technique found to be useful in detecting linear fabrics was to slide the sensitive tint plate under crossed polars, hence changing the order of the polarisation colours. This appears to optically highlight small variations in either pore space or fine intergranular material hence drawing ones attention to potential areas of interest.
Figure: 4.19 Diagram shows the orientation of the plane of the thin sections on a half cylinder sample (already exposed as in figure: 4.1). Maximum stress is vertical and parallel to the cutting direction. The plane of the thin section is chosen to be perpendicular to the plane of the fault, in order to expose the fault structures.
[4.4.3] Qualitative microstructural data: Series 1 test samples

Initially, a qualitative microstructural analysis was carried out to describe the main features and highlight areas of interest for more detailed investigation. Figure: 4.20a is a photomicrograph showing part of a laboratory induced fault zone. Distinct individual buff coloured strands are clear, and occur sub parallel to each other at angles of less than 0.45 degrees to $\sigma_1$ (trending top right to bottom left in the frame). These sub parallel fine grained gouge strands are equivalent to the pale interweaving strands observed in the radial hand specimens in Figures: 4.2a-j. There is a significant reduction in grain size inside these strands which is clearly highlighted in Figure: 4.20b, which is a photomicrograph taken under cross polars, at higher magnification, showing two gouge strands oblique to the frame. The strands are therefore referred to as "gouge strands". This observation is consistent with laser size analysis results (e.g. Figure: 4.13a). Figure: 4.20c is a photomicrograph, taken under plane polarised light, which shows that structures can also be identified within the individual strands. The structure manifests itself either as zones of intense grain size reduction (ultrafine gouge) lying parallel to the sides of the gouge strand or linear "stringers" of brown very fine grained material (opaque under cross polarisers) lying at an angle to the strand edges. This opaque material is interpreted to be hematite which originates as grain coatings on the quartz grains, and is also concentrated along bedding laminations in the undeformed. Longitudinally continuous lenses of host rock, referred to here as "pods", occur between gouge strands and show significant grain fracture which is measured quantitatively in section [4.4.12]. No significant grain size reduction is apparent in these pods with respect to the host rock. In contrast to the intensive microcracking in the pods and wall rock, there is a distinct lack of microfractures in the original grains caught up in the gouge strands. The gouge strands are commonly approximately 300 $\mu$m in width, compared to the pods which are approximately 800 $\mu$m wide. Although respective widths do vary, the pods of host rock are always wider than the strands. The texture of the gouge strands i.e. shape and angularity of grains, is different from the host rock, the former being dominated by more angular fragments. The strands consist of grains which are mainly surrounded by a sea of fine particles resulting in a matrix-supported texture.

[4.4.4] Qualitative Microstructural Analysis: Series 2 Test Samples

Thin sections of Series 2 test samples were cut parallel to $\sigma_1$ but the deformation consisted of many planes so the planes are not necessarily cut perpendicular to their dip as was the case in series 1 samples. All the dips are therefore apparent dips. Figure: 4.21a shows a
The sample was shortened by 11.22% axial strain. Maximum stress (σ1) is vertical and confining stresses (σ2-σ3) are in the horizontal plane. The field of view along their horizontal edge is 8.7mm.

Figure: 4.20 b) Photomicrograph of the same thin section as in a), at higher magnification highlighting the structure occurring within gouge strands. The field of view is 4.3mm along the horizontal axis and σ1 is vertical.
Figure: 4.20 c) Photograph of sample 2 shortened to 11.22% (as shown in preceding two figures) under cross polarisers. The horizontal field of view is 4.3mm.
Figure: 4.21 Photomicrographs of confining pressure tests cut parallel to $\sigma_1$ (horizontal field of view = 2.7 mm in figures a and b; and 4.3 mm in figure c. $\sigma_1$ is vertical in all photographs.). a) An isolated strand (run11) shows little adjacent microcracking. b) the area between two neighbouring strands in the same thin section, is significantly damaged with microcracking. c) a wide pod (run9) shows little microcracking.
Figure: 4.21 C
photomicrograph of sample 11a near an isolated single strand. There are very few microcracks here, whereas the density of microcracks near closely spaced strands is much higher (Figure: 4.21b). Microcracks appear to have an axial orientation and are abundant near gouge strands which are composed of several distinct strands.

Figure: 4.21c is a photomicrograph of sample 9a showing a wider than normal distance between neighbouring strands. When the pod between two strands is wide then there are fewer microcracks than observed in narrower pods. Although microcracks are absent in this pod, in general, this thin section is fairly well cracked.

The main difference in the style of deformation as a function of confining pressure is the systematic change from localisation on a plane to radially distributed deformation (see Figure: 4.7). This is observed best on radial cross sections perpendicular to maximum compressive stress, i.e., parallel to the plane cut in Figure: 4.1, and that displayed in Figure: 4.7. Therefore a second set of thin sections (e.g., from Run 11) were cut parallel to this plane and Figure: 4.22 shows such a thin section. The blue epoxy impregnation in this particular sample is not a good indicator of porosity as clear epoxy was initially impregnated to aid the cutting procedure and the blue epoxy used later only reaches some of the pores. Hence, the thin section is viewed here under crossed polars to avoid misleading the eye. Cataclastic gouge strands and adjacent microcracks are the main expression of damage and they occur as roughly radial spokes which traverse the sample. The intersection points between more than one strand lead to complex deformation. Approximately 8 major spokes are identified in the section studied, which consists of an area of approximately 3870 mm$^2$. Some spokes consist of single strands, whereas others have multiple distinct gouge strands. Microcracking appears closely associated with the strands and in Figure: 4.22a the strike of the microcracks is sub-parallel to the strike of the wavy gouge strands. The density of microcracks associated with the gouge strands appears to be higher in the areas near gouge strands which are closely spaced. When strands are more isolated (Figure: 4.22b), the number of microcracks associated with them are much less, and tend to only extend one or two grain diameters out into the wall rock (if indeed there are any at all). The abundance of microcracks is also observed to increase towards the centre of the test sample, whereas the outer edges are fairly fracture free.
Figure: 4.22a Photomicrograph of a confining pressure (series 2) test, cut perpendicular to the sample long axis, under crossed polarised light. Horizontal field of view is 4.3mm. Frames A and B show different regions of the same thin section. a) Two closely spaced strands trend mid/lower left to upper right. Note the abundance of microcracking, especially where the strands meet and microcracks following the strike of the strand.

Figure: 4.22b Photomicrograph of the same thin section showing an isolated strand (top right to bottom left). Note the relative absence of microcracking except immediately adjacent to this strand.
[4.4.5] Point Counting Technique

In order to obtain an unbiased quantitative analysis of the fault zone and host rock, a point counting technique was utilised. A mechanical device, fixed to the thin section, advances the thin section slide a known distance across the field of view along the x or y axis. The mineral grain appearing directly under the cross hairs, at every position, is then identified and tallied.

I carried out a series of linear point count transects in orientations parallel or perpendicular to the direction of maximum compressive stress and also parallel to the trend of the fault zone, with the number of counts per transect varying from 30 to 150. Here a standard petrological microscope, with magnification of x4, x10, x40, was used to gain detailed data from individual grains. In these transects I noted information concerning the grain diameter, shape and fracturing, as well as the pores and other grains surrounding the ‘cross hair’ grain.

Maximum and minimum grain (or pore) diameters were measured using an optical micrometer inserted in the eyepiece of the microscope. The degree of angularity of each grain was estimated by comparison with the classification of Pettijohn et al., (1987), where numbers 0 to 5 define increasing roundness (see Figure: 4.23). The grain was then investigated for fractures. The number, orientation and type of fractures (minor fractures or major through going fractures) was recorded. Figure: 4.24 is a sketch showing the definition of a) minor and b) major through going intra-granular fractures and the number of surrounding grains and pores greater than, equal to and less than the size of crosshair grain were recorded. These particular characteristics were chosen as they describe the textures and fabric of the sample, they reveal how grains fit together and relate to each other, they are visible displays of damage and reveal information regarding the porosity of different parts of the sample. They are good features to point count as they are clearly identifiable and can be measured without ambiguity.

All data acquired from point counting analysis was entered into a data file then subsequently processed using a purpose built ‘Visual Basic for Applications’ programme sheet (see Appendix: F). The programme calculated e.g. grain diameter in microns (as opposed to micrometer ticks at a specified magnification), aspect ratio of grains, number of minor fractures per grain, means and standard deviations of quantities, the sum of minor, major and total fractures in a grain. Point count data is tabulated in table: 4.3.
Figure: 4.23 Categories of roundness for sediment grains. For each category a grain of high and low sphericity is given. After Pettijohn et al. (1987).

Figure: 4.24 Sketch showing the definition of a) minor and b) major through going fractures subsequently used in quantitative microstructural analysis.
Table: 4.3 Visual basic template for processing raw point count data.

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| Reference (x,y) | (0, 0) | x is perpendicular; y is parallel to max compressive stress |

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<th>Space microns^2</th>
<th>Orientation of Fractures in degrees 000-max stress</th>
<th>Total #</th>
</tr>
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<tbody>
<tr>
<td><strong>P&lt;</strong></td>
<td><strong>P&gt;</strong></td>
<td><strong>Orientation of Fractures in degrees 000-max stress</strong></td>
<td><strong>Total #</strong></td>
</tr>
<tr>
<td><strong>Area G</strong></td>
<td><strong>Area P</strong></td>
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<td><strong>Minor F</strong></td>
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Table: 4.3 (continued)

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<td>323</td>
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<td>10</td>
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<table>
<thead>
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<th>Fractures in degrees from 000 = maxini and Total</th>
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</thead>
<tbody>
<tr>
<td>MTGF</td>
</tr>
<tr>
<td>29</td>
</tr>
</tbody>
</table>

Subtotal 172
Mean 0.827273
Stdev 1.563636

1.516667 #/counts
1.516667 #/(counts-pores)
[4.4.6] Point Count Transects

The specific orientations and locations of each point count transect are presented in Figure: 4.25 and Table: 4.4. The aim of these transects was to quantify the texture and deformation mechanisms in different areas of the sample. Transects were carried out (i) within the fault zone in the gouge strands and host rock pods, (ii) adjacent to the fault zone in the wall rock and (iii) at perpendicular distance of 10-12 mm from the fault zone in the host rock. “Counts” are defined as the number of grains appearing under the cross hairs in the transect. The type of information collected in each transect was influenced in some sense by the region under consideration. High magnification was required for measurement of gouge strand grains, whereas low power and a large field of view was required to examine the widths and orientations of the gouge strands themselves. Comparison of features at different magnifications e.g. microfractures is difficult as some features which do not show up at low power magnification may dominate at higher magnification. As a result, some transects were point counted for only specific variables and microfractures were only counted under a constant magnification of x10. Table: 4.5 is a list of the variables counted on the different transects at different magnifications.

[4.4.7] Quantification of discrete gouge strands

In the qualitative description of the fault zone in section [4.2.2], it was observed that fault zones in the series 1 tests consist of larger numbers of strands as applied axial strain is increased (see Figure: 4.3). In order to quantify these observations independently, the number of gouge strands encountered on a series of horizontal traverses (see Figure: 4.25: traverses labelled T6 and T7) across the thin section fault zones was counted under an optical microscope. Several transects were carried out in this orientation for each thin section. Figure: 4.26 shows the number of gouge strands encountered as a function of the final applied axial strain reached by the different samples. For reference, the failure strain of the samples measured from the loading curves, is also plotted here (see tables: 3.1, 3.2, 3.3), assuming that the number of gouge strands at dynamic failure is zero. This is a reasonable assumption as run13 which was stopped just prior to failure shows no evidence of strands (Figure: 4.2a). The graph shows a significant positive correlation of increasing number of strands with increasing axial strain ($R^2=0.96$). The regression curve intersects the x-axis at a value of strain of approximately 2%, which is close to the value of strain at failure for the samples considered. Therefore, the number of strands depends linearly on the amount of strain.
Figure: 4.25 Line diagram showing locations and orientations of thin section point count transects, with reference to the fault zone structures and maximum compressive stress orientation. Individual transects are labelled T1-T10 for later reference. i) fault zone gouge strands / host pods; ii) wall rock; iii) host rock.

<table>
<thead>
<tr>
<th>Number of Gouge Strands</th>
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<tr>
<td></td>
</tr>
<tr>
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</tbody>
</table>

\[ y = 125.01x - 3.2367 \]
\[ R^2 = 0.9588 \]

Failure Strain

Figure: 4.26 Graph showing the number of gouge strands versus applied axial strain (%), as measured from horizontal point count transects (T6 and T7 marked on figure: 4.25) of thin sections. The data shows a linear correlation of increasing numbers of strands for increasing applied axial strain (solid line). The value of applied axial strain at macroscopic failure for the samples considered is plotted on this graph for comparison.
Table: 4.4 Denotes the orientation and position of point count transects in thin sections with respect to the fault zone structures and the macroscopic stress imposed.

<table>
<thead>
<tr>
<th>Transect</th>
<th>Counts</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>T2</td>
<td>60</td>
<td>Transect parallel to the fault direction, in the fault zone, in the pod nearest to the edge</td>
</tr>
<tr>
<td>T3</td>
<td>60</td>
<td>Transect parallel to fault, in a fault pod near the middle of the fault zone</td>
</tr>
<tr>
<td>T4</td>
<td>60</td>
<td>Transect parallel to the orientation of the fault zone, in the wall rock immediately adjacent to the fault</td>
</tr>
<tr>
<td>T5</td>
<td>60</td>
<td>Transect parallel to the fault, but in the host rock at distances of 10-12mm from the edge of the fault.</td>
</tr>
<tr>
<td>T6</td>
<td>-</td>
<td>Transect perpendicular to $\sigma_1$, across fault zone</td>
</tr>
<tr>
<td>T7</td>
<td>-</td>
<td>Transect perpendicular to $\sigma_1$, across the fault zone</td>
</tr>
<tr>
<td>T8</td>
<td>60</td>
<td>Transect parallel to fault in a gouge strand (located near to the host)</td>
</tr>
<tr>
<td>T9</td>
<td>60</td>
<td>Transect parallel to the fault, in a gouge strand (located away from the host)</td>
</tr>
<tr>
<td>T10</td>
<td>60</td>
<td>Transect parallel to fault zone, in a gouge strand</td>
</tr>
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</table>
Table: 4.5 List of the variables counted on each transect

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<thead>
<tr>
<th>T2, T3, T4, T5 (at Magnification x10):</th>
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</thead>
<tbody>
<tr>
<td>Diameter of grain (maximum).</td>
</tr>
<tr>
<td>Diameter of grain (minimum).</td>
</tr>
<tr>
<td>Angularity of grain estimated using Pettijohn’s classification see fig: 4.16.</td>
</tr>
<tr>
<td>Orientation of long axis of grains measured with respect to $\sigma_i$.</td>
</tr>
<tr>
<td>Number of neighbouring grains: less than; equal to; and greater than the size of the crosshair grain.</td>
</tr>
<tr>
<td>Number of neighbouring pores: less than; equal to; and greater than the size of the crosshair grain.</td>
</tr>
<tr>
<td>Whether the crosshair grain was Matrix (too small measure) or a Pore.</td>
</tr>
<tr>
<td>Orientations of up to 6 minor fractures within the grain measured with respect to $\sigma_i$.</td>
</tr>
<tr>
<td>Orientations of up to 15 major through going fractures (again with respect to $\sigma_i$).</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>T6, T7 (at Magnification x4):</th>
</tr>
</thead>
<tbody>
<tr>
<td>Width of linear features.</td>
</tr>
<tr>
<td>Defined linear features as fault strands or fault pods.</td>
</tr>
<tr>
<td>Orientations of any fabrics within the fault strands.</td>
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</table>

<table>
<thead>
<tr>
<th>T8, T9, T10 (at Magnification x40):</th>
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<tbody>
<tr>
<td>Grain diameter (maximum) measured.</td>
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</table>
[4.4.8] Measurement of gouge strand orientation

The orientation of gouge strands, with respect to the maximum stress axis, is presented in Figure: 4.27. Orientation was measured for horizontal transects across the fault zones (transects T6 and T7 on Figure: 4.25) from thin sections (taken parallel to $\sigma_1$) of three samples subjected to increasing amounts of axial strain. Figure: 4.27a, b and c show rose diagram plots of the orientation histogram data determined from the data from samples subjected to 4.22%, 6.34% and 11.22% axial strain respectively. The data presented here is equivalent to the angle of maximum dip of the fault measured with respect to the maximum compressive stress orientation. Data is plotted clockwise with respect to the maximum stress orientation marked at 000 and the length of ‘petals’ is proportional to bin density. The gouge strands trend at between 020 and 040 degrees to $\sigma_1$ with mean orientations at 027, 025, 036 respectively for samples strained to 4.22%, 6.34% and 11.22%. The strands are sub-parallel, and there is little scatter in the data. The orientation of strands show most spread for the highest strained sample (although this would be expected due to the larger number of strands). There is no obvious systematic rotation of gouge strand direction with respect to changes in axial strain.

The orientation of the dominant structure within the individual gouge strands is presented in Figure: 4.28 a, b, c and shows a mean trend of 060 degrees but show a greater spread than gouge strand orientations. This structural trend in the gouge strands, described qualitatively in section [4.4.3], therefore lies at approximately 30 degrees to the strands themselves. Again a larger spread in the data is observed at higher strains. Host rock pods caught up in the fault zone are bounded by the gouge strands and hence adopt the same trends.

[4.4.9] Measurement of gouge strand width

Engelder (1974) created a simulated gouge in the laboratory, on single saw cut surfaces of sandstone samples, and found a linear correlation between the thickness of gouge produced and the displacement along the sliding surface, confirmed by Jackson and Dunn (1974). It may seem intuitively obvious that the amount of gouge generated should depend on the displacement along a fault, but Engelder (1974) also showed that in field studies “a random measurement of gouge thickness along shear fractures does not indicate a clear relationship between thickness and displacement”. This paradox is ascribed to the bifurcation of shear fractures, where individual strands frequently cross each other, resulting in an extremely variable total fault width which can not be described accurately by a single measurement in
Figure: 4.27 Rose diagram plots showing the frequency distribution of the angle of dip of gouge strands (left-hand column) and trends within the strands (right-hand column) with respect to sigma 1. The angle is plotted clockwise from sigma 1 at 000, in 10 degree bins, and the length of the “petals” are proportional to the frequency. The mean angle, and number of data points used to determine the mean, are displayed beneath each diagram. a) shows run5 data; b) data from run1; and c) data from run2. The mean orientation of the strands in all cases is approximately 030 degrees, and of the trends within the strands is approximately 060 degrees.

Figure: 4.28 Rose diagrams show the orientations of structures within the gouge strands with respect to sigma 1. a) shows run5; b) data from run1; and c) shows run2 data. The general trend of these structures is at 60 degrees to sigma 1.
the field. The structures produced in my experiments (Figure: 4.2) do show this kind of bifurcation in plan view, but in the plane containing the fault (Figure: 4.20a) there is a fairly constant width of fault zone at least for the extent of the thin sections analysed here. In this sub-section, I present width measurements for the total fault zone, individual gouge strands, and the sum of gouge strand width, as well as measurements of the gouge zone spacing (i.e. pod width) due to fault zone pods occurring between the strands themselves.

The total fault zone width (measured on thin section from the sum of the fault strand and fault pod widths) is plotted as a function of axial strain in Figure: 4.29. The width of fault zones is positively correlated with axial strain showing a linear relationship ($R^2 = 0.96$) which corresponds to a similar relationship for hand specimen data presented in Figure: 4.4. Again, if no fault exists before failure (a reasonable assumption in view of Figure: 4.2a), then the intersection of this linear relationship with the x-axis should coincide with amount of axial strain at dynamic failure. The graph shows this linear regression plotted together with the measured axial strains at failure for the three tests considered from the stress strain curves (see chapter 3, table:3.1, 3.2, 3.3). The agreement between the two is good.

The width of individual gouge strands and the host rock pods occurring between them is presented in Figure: 4.30 for samples deformed to strains of 4.22%, 6.34% and 11.22% respectively. The gouge strands are commonly 300-500 μm in width, compared to the pods of host rock which are approximately 800 μm wide. The mean values and standard deviations for the gouge strands are 408 μm +/- 142 μm, 616 μm +/- 304 μm and 210 μm +/- 130 μm. The equivalent values for fault zone pods are 2008 μm +/- 554 μm, 1275 μm +/- 613 μm and 1072 μm +/- 1298 μm. Although the respective widths are variable, the pods of host rock are always wider than the gouge strands in any particular sample.

The mean width of strands and pods as a function of axial strain are plotted in Figure: 4.31. There is no strong correlation with axial strain although the mean pod width decreases non-linearly with increased strain. Therefore in summary, the main correlation observed is between the total fault zone width and axial strain.

[4.4.10] Grain size distributions

Grain size analysis is carried out on thin section samples in order to obtain an independent measure of distributions where spatial relations are preserved and no disaggregation of the sample (as is necessary for laser size analysis - see Appendix D) has taken place. Figure: 4.32a, b, c show the cumulative frequency distribution of grain size for samples deformed at 34.5 MPa confining pressure by different amounts of axial strain of 4.22%, 6.34%, and
Figure: 4.29 Graph of the total width of fault zone (i.e. the sum of strands plus the sum of pods) versus axial strain measured from longitudinal thin sections. The failure strains of the samples are also plotted for comparison.

Figure: 4.30 Graphs show the relative widths of pods and gouge strands. Plotted as width of areas of pods or strands in microns on log scale versus position across the fault zone for: a) run5 subjected to 4.2%; b) run1 subjected to 6.3%; and c) run2 subjected to 11.2%. The width of the fault zone pods or strands is measured perpendicular to the dip of the fault zone.
Run 1d, 6.3% axial strain

Figure: 4.30 B

Run 2d, 11.2% axial strain

Figure: 4.30 C

Mean Pod Width and Mean Strand Width

Figure: 4.31 Graph of mean pod width and mean width of strands versus applied axial strain.
Figure: 4.32 Graphs showing the grain size - frequency distributions for the gouge strands, fault zone pods and host data measured from point count analysis of thin sections. Cumulative frequency (smaller than) is plotted as a function of grain diameter which is plotted on a log scale. Data is presented for: a) run5; b) run1; and c) run2.
Figure 4.32 (continued)
11.22% respectively. The cumulative frequency smaller than \((N(D \leq d))\) is plotted as a percentage of total grains counted as a function of grain diameter \((d)\) in microns plotted on a log scale. Data is presented for (i) fault gouge strands, (ii) host rock pods caught up in the fault zone, (iii) wall rock adjacent to the fault zone, (iv) host rock away from the fault zone. There are basically two characteristic shapes of curves displayed on each graph: for gouge strands and for all other transects. In each graph (see Figure: 4.32a, b, c), the data collected from different gouge strands (transects T8, T9, T10) occurring in the same fault zone show a good match with each other, irrespective of position in the fault zone, both in the style of the curve and also position of the curve along the x-axis.

Figure: 4.33a shows a comparison between gouge strands in samples deformed to increasing strain. The style and position of the curves are again very similar, hence any single gouge strand has an equivalent grain size distribution to any other strand, whatever the strain applied to the bulk sample. The curves showing data collected in the host rock, the wall rock and the fault zone pods show broadly the same behaviour as each other - see Figure: 4.33b. The steepest gradient is at intermediate diameters and the curves flatten off both at high and low sizes. There is no consistent trend of behaviour recognisable in the different regions and the grain size distribution is therefore interpreted to be essentially the same in the pods, host rock and wall rock. Again the host, pod, wall rock curves show the same characteristics in samples deformed by different amounts of axial strain. The contrast between the shallow slope and broad scale range of the gouge strand curves compared to the steeper slope and narrower scale range for other areas of the sample imply a poorer sorting in the gouge strands. The shift left in the position of the curves along the x-axis also indicates a significantly smaller mean grain size in the gouge strands as highlighted in Figure: 4.34 showing the mean grain size as a function of region. The mean grain size in the strands is 10 \(\mu m\), whereas mean grain size in the pods is of the order of 160 \(\mu m\), which is equivalent to grain size in the host rock and the wall rock. The reduction in grain size in the strands is therefore an order of magnitude corresponding to a change from a medium-sand grade to a silt grade.

Figure: 4.35 are density plots showing number (\%) as a function of grain diameter for the same data already presented in Figure: 4.32. These graphs are included for ease of comparison to laser sizing data (section [4.3.2]) and other studies. They highlight the fact that grain size distributions, for finite ranges, can be approximated by a gaussian i.e. normal distribution which is eluded to in the description of the cumulative frequency plots. I note that the smaller grain size, combined with poorer sorting, in the gouge strands with respect
Figure 4.33 Cumulative frequency versus grain diameter graphs plotted for specific regions i.e. a) strands and b) pods respectively to allow a comparison of characteristics for increased applied axial strain.
Figure: 4.34 Graph of mean grain diameter as a function of different regions within three samples subjected to 4.2%, 6.34% and 11.22% axial strain. Mean grain size is similar in pods, host and wall rock but is significantly reduced in gouge strands.
Figure: 4.35 Graphs showing particle size by number for: a) Run5d (4.22% axial strain); b) Run6d (6.5% axial strain); c) Run2d (11.2% axial strain). Curves are shown for different regions (denoted by T#) as defined in Figure: 4.25.
Point Count Analysis
11.2% Axial strain

Figure: 4.35 (continued)
to the host, implies a significant reduction in permeability across the gouge zones (see sections [4.3.4] and [1.6.3] this study). I now consider the effects of these observations on the porosity and permeability in more detail by quantifying the pore space directly through point count analysis.

[4.4.11] Pore space analysis

I have shown that the grain size in gouge strands is significantly reduced with respect to the host rock (sections [4.2.7] and [4.4.10]). A reduction in grain size alone will not necessarily cause a reduction in porosity if the grain shape, sorting and packing remain constant but as shown in section [4.4.10], sorting is also significantly different in the gouge strands compared to the host and this will certainly affect the poro-perm nature of the strand. In an attempt to quantify the pore space by point counting, thin section samples were impregnated with blue epoxy resin (see Appendix E for details on impregnation techniques). The photomicrograph in Figure: 4.36a shows part of a fault zone containing both fault zone pods of host rock and a gouge strand. Figure: 4.36b is a photomicrograph of host rock at a perpendicular distance of from 10 mm away from the fault. The thin section, shown in both Figures, is cut parallel to the maximum stress and perpendicular to the strike of the fault zone (i.e. the plane exposed in Figure: 4.19). Significant presence of blue epoxy resin is observed in the fault zone pods (Figure: 4.36a) and also in the host rock shown (Figure: 4.36b), implying significant porosity in both regions. In contrast, no impregnation of blue epoxy resin is observed in the gouge strands (see Figure: 4.36a). This implies that the porosity, and hence the permeability of the gouge strands is significantly smaller than both the pods and the host rock. The absence of resin impregnation means that pore space could not be measured within the gouge strands using the point counting technique but implies that porosity is extremely small here.

The skeleton (or primary) porosity in the fault zone pods and the host rock was measured by point counting the pore space neighbouring each cross hair grain in a series of traverses. Here, secondary porosity due to microcracks was not included (this is considered separately later) and therefore the voids filled with blue epoxy resin were, in general, sub-rounded. The pore space was divided into sizes smaller than, equal to and larger in diameter than the grain under the cross hairs. The graphs in Figure: 4.37 show the mean number of neighbouring pores per grain in different areas of the sample for pore sizes greater than, equal to, and smaller than the cross hair grain. The transects here were all carried out in directions parallel to the trend of the fault zones (see Figure: 4.25) and were taken in: (i) the
Figure: 4.36a Photomicrograph showing thin section which has been impregnation with blue coloured epoxy resin in order to highlight the pore space. The frame shows part of a fault zone containing one gouge strand (trending top right to bottom left) and two adjacent fault zone pods. \(\sigma_1\) is vertical and the horizontal edge is 2.7mm. The gouge strand is obvious due to its lack of epoxy impregnation whereas the fault zone pods show good impregnation. Sample was deformed by 11.2\% strain.

Figure: 4.36b shows thin section of host rock approximately 10mm away from the fault zone. The section is again impregnated with blue epoxy impregnation which clearly highlights the pore space. The section has a horizontal edge of 4.3mm and \(\sigma_1\) is vertical.
Figure: 4.37 a, b, c. Graphs of point count data showing the neighbouring pore space per grain versus the position or region of the sample. Data is split up into three subsets of pores which are larger (>), equal to (=) and smaller (<) in size than the cross hair grain. Most pores are smaller than the grains. There is a definite reduction in pore space in the fault zone with respect to the host rock.
host rock (T5) 10-12 mm away from the fault zone edge; (ii) the wallrock (T4) defined as the region adjacent to the fault zone, 5-10 grain diameters wide where density of microcracks is notably higher than in the host; (iii) a pod near the edge of the fault zone (T2); and (iv) a pod in the middle of the fault zone (T3) as for the particle size analysis. The x-axes indicate the type location of transects. Although this data can not be directly converted into a porosity (without a series of assumptions concerning grain shape and packing), it does give a quantitative indication of the amount and nature of pore space surrounding the grains in different parts of the sample.

The consistent finding from the analysis presented here confirmed that the amount of pore space in pods located at both the centre and edge of the fault zone is seriously diminished compared to the pore space measured in the host rock. (This observation can also be made qualitatively in Figure: 4.36a, b). In general most of the pore space surrounding a grain is smaller than the size of the grain. In two cases Figures: 4.37 a, c, the pore space in the wall rock adjacent to the fault zone is reduced but in the third Figure: 4.37b it is actually enhanced relative to the host rock. This may imply some dilatant effect and will be considered in more detail with reference to mechanical observations in the discussion (chapter 6). There appears to be no correlation between the amount of pore space and axial strain in these samples. In summary, the permeability in the gouge strands is qualitatively much lower than that in the fault zone pods which surround it. Similarly, the pods show a lower amount of pore space than the host rock, hence the gouge strands are interpreted to have significantly reduced porosity and permeability with respect to the host rock.

[4.4.12] Microfractures
On initial inspection of hand specimens (section [4.2.2]) it appeared that the fault zone pods occurring between the gouge strands were lenses of undamaged host rock caught up in the fault zone. Under microscopic investigation however (see section [4.4.3]), these pods are observed to have significant microcracking within grains. The photomicrograph in Figure: 4.38 shows significant microfracturing in the pods within the fault zone near to a gouge. The microfractures commonly extend across the diameter of one or more grains. Most grains are still intact but show significant damage.

In order to obtain a measure of the density of microcracking and the orientations of microcracks, linear point counting transects were carried out along fault zone pods in a direction parallel to the fault zone trend (see T2 and T3 in Figure: 4.25). To investigate the
Figure: 4.38 Photomicrograph of sample deformed to 11.2% strain showing pervasive microcracking in fault zone pods adjacent to a gouge strand. Also shown clearly is the structure within the strand and non-fractured host grains surrounded by finer grained material. The thin section has a horizontal edge of 2.7mm and σ1 is vertical.
density of cracking with distance away from the fault zone and with respect to the host rock, microcracks were also analysed in the wall rock (T4) and the host rock (T5). During this analysis microcrack damage within grains was differentiated into minor fractures and major through going fractures defined in Figure: 4.24. The number of minor fractures generally exceeds the number of major through going fractures in any given transect (see Figure: 4.39a, b, c). The largest proportion of microcracks per grain are found in the central fault zone pods. In the wall rock there are still a significant number of microfractures and in Figure: 4.39b, the number of fractures in the wall rock actually exceeds the number of fractures in the fault zone pods. Outside this wall rock zone the density of cracks diminishes markedly to what is defined as the background level. Microcracks in the wall rock are pervasive, up to approximately 5-10 grain diameters out into the host rock irrespective of the axial strain applied to the host rock.

Figures: 4.40a, b, c show rose diagrams of the microcrack orientation data for samples deformed to strains of 4.22%, 6.34% and 11.22% respectively. The mean directions of cracks are quoted in Figure: 4.40 along with density factor which is equal to number of fractures per point count grain. The orientation of the minor and major microcracks was found to be almost identical and hence they are plotted together. A pronounced microcrack anisotropy is observed due to microcracks in the fault zone pods predominantly oriented sub-parallel to the maximum stress axis (mean orientation between 347 and 002). These cracks are subsequently defined as axial microcracks. Microcracks in the wall rock zone show slightly more spread, and lower densities, but the mean direction is still axial. It is possible that the larger spread in orientation is due to later rotations of initially axial microcracks. Microfractures occurring in the host rock show further spread of fracture orientations and very low densities.

[4.4.13] Neighbouring grains
In order to characterise the environment which surrounds each grain and how this varies in different regions of the sample, the neighbouring grains to each cross hair grain were noted. Point count transects were carried out in the fault zone pods (T2, T3), wall rock (T4) and host rock (T5) (see Figure: 4.25). Data from these transects is presented in Figure: 4.41 where number of neighbouring grains per grain i.e. grain density factor, is plotted for the different regions of the samples specified along the x-axis. Figure: 4.41 a, b, c show data from samples deformed by 4.22%, 6.34%, 11.22% respectively. In all tests, the largest proportion of neighbours were smaller than the cross hair grains. Grains were surrounded
Chapter 4 - Rock Deformation Tests: Structural Data

Run5d, 4.2% axial strain

![Graph A]

Run1d, 6.3% axial strain

![Graph B]

Run2d, 11.2% axial strain

![Graph C]

NB: mf = minor fractures, MTGF = major throughgoing fractures

Figure: 4.39a, b, c, Graphs of point count data for samples from runs 5, 1, and 2 respectively. The graphs show the density of minor and major microcracks in different parts of the sample. Microfracture density is plotted as a number of microcracks per grain.
Figure: 4.40 a, b, c. Rose diagrams (plotted as described in figure: 4.27) show the orientation and abundance of microcracks in the fault zone, wall rock and host rock of three samples. a) shows data for sample run5; b) shows data from run1; and c) gives data for run2. These rose diagrams have the equivalent bin sizes of 10 degrees and frequency is proportional to the length of the petals.
Figure: 4.41 a, b, c Graphs of point count data showing the neighbouring grains surrounding the cross hair grain versus location in the sample. The number of counts represented by the data is displayed on each graph by $n = \#$. Data is split up into grains larger than (>), equal to (=) and smaller than (<) the cross hair grain. Approximately 3 grains surrounding the cross hair grain are smaller, 2 are roughly equal in size and 1-2 are larger in size.
by approximately 2 grains of similar size, 3 grains smaller and 0.5 to 1 grains larger than the one being counted. There was no obvious trend to the measured number of neighbours with respect to different regions in the sample or increasing amounts of axial strain except that there tended to be more smaller grains and less large grains in the fault zone compared to the host rock.

[4.4.14] Limitations of 2-dimensional thin section analyses

One important limitation of the analysis of 2-D thin section slices of a 3-D granular matrix, which must be mentioned, is that grains are not necessarily cut at their maximum diameters. Underwood (1970) highlighted this factor by pointing out that a series of different sized spheres can produce one particular sized circular section if they are all cut at different places (see Figure: 4.42a) and conversely a series of similar sized spheres can produce different sized circular cross sections depending on where they are sliced (see Figure: 4.42b).

The mechanical grinding action necessary to produce thin sections may itself induce microfracturing. One section studied here shows features which have been interpreted as grinding induced fractures. They occur only in one area of the section and are characterised by a set of sub parallel curved scores which cross several grains. They do not appear to damage the grains they cross, and show no impregnation by blue epoxy resin in contrast with most other major fractures in that sample. Therefore I believe they are distinguishable from laboratory deformation features.

Point counting is a rewarding but extremely time consuming technique so a balance was sought between the aim of reducing errors by counting large numbers of grains and the necessity of obtaining a significant amount of data in a reasonable time.

These uncertainties are not expected to significantly affect my main conclusions, although their potential effects should be noted.

[4.4.15] Summary of data obtained thin section analysis

The data obtained from qualitative and quantitative thin section analysis of deformed samples has been presented above. The fault zones are observed to consist of distinct gouge strands of reduced grain size, sorting and porosity, with respect to the host rock, lying sub-parallel to the direction of shear. These strands are separated by pods of relatively undeformed host rock which have the same grain size as the host, but have significantly reduced pore space between grains and show pervasive axial micro-cracking. The number of gouge strands increases as a function of applied axial strain. Increased confining pressure
Figure: 4.42 Sketch from Underwood - Introduction to Stereology highlighting some of the problems of 2-d analysis of a 3-d material. a) similar sized circular section can be achieved by cutting spheres of different diameter in different places. b) variable circular sections can be achieved from uniform sized spheres.
produces more radially-symmetric deformation structures, whose density appears to be a function of the depth simulated. These microstructural features are compared to observations of field microstructures in Chapter 5 and are discussed with regard to mechanical results in Chapter 6.

**[4.5] SEM Analysis**

**[4.5.1] Introduction**

Scanning Electron Microscopy (SEM) has become a standard examination technique used in sedimentary petrology. Very high magnifications are possible which allow the viewer to distinguish and analyse features down to approximately 0.1 μm in size. A wide range of magnification from approximately x15 to x50000 allows the possibility of observing whole objects as well as individual grains or several grains together. The whole sample can be surveyed to gain an understanding of the surrounding environment before a grain itself is studied in detail. The great depth of field makes this technique invaluable for studying the surface textures of grains. The technique has been presented previously in many texts (e.g. Terwin 1988).

SEM was used here to analyse the same deformation test samples as described in previous chapters. The samples chosen for study are presented in the next section, and following this, the key features of several Scanning Electron Photomicrographs are presented and described.

**[4.5.2] Samples analysed**

Specimens from two different tests (run2 and run6) were analysed using SEM. Three specimens representing different areas from each deformed sample were chosen in order to study the textures, grain sizes and fabrics of the deformation zones. The specimens are identified by a number (e.g. 2, 6) denoting the run from which they were obtained, and a letter (e.g. g, f, t) denoting the type of specimen. Fresh gouge material (g) was carefully scraped from a single gouge strand from each sample. The gouge was stuck to double sided sticky tape mounted on a stub then gold coated under a vacuum. A small section of fault zone surface (f), with the gouge material still adhering, was glued directly to a specimen stub and gold coated. The topography of a fault surface (t) was exposed by removing all the gouge material using a high pressure air line. This specimen was again glued to a stub and gold coated.
Figure: 4.43 SEM photomicrographs of gouge material scraped off a fault zone surface a) taken at x80 magnification showing a surviving host grain from sample 6g; b) gouge fragments of sample 6g at x1500 magnification; c) striations on an individual gouge fragment of sample 6g under x3000 magnification.
Figure: 4.44 SEM photomicrographs of host rock grains appearing in gouge material a) host rock grain of sample 6g under a magnification of x400; b) host rock grain showing en-echelon fractures from sample 6g under x4000 magnification.
Figure: 4.45 SEM photomicrograph of wall rock adjacent to the fault zone and in the plane of the fault showing: a) a series of roughly equal sized, equant shaped grains on sample 6t, magnified by x40; b) more detailed view of the same surface at x800 magnification showing the fractures in the intact surface; c) x1600 magnification of a single grain from this surface highlighting an en-echelon series of fractures.
**4.5.3 Description of SEM photomicrographs**

The resulting Scanning Electron Photomicrographs are described in this section, in terms of the size, shape, surface textures of grains, their fabrics, and relationships between grains. Analysis of these generic features of the gouge material, host grains caught up in the gouge, and wall rock at a fault zone surface are presented separately below.

**Gouge material**

Figure: 4.43a is a Scanning Electron Photomicrograph of gouge material, showing surviving host grains which are the large grains seen. The fabric of the gouge, and the contrast in grain sizes between the gouge and the surviving host grains, are clear. The smaller grains which compose the majority of the gouge material are generally equant in shape though are often very angular. Figure: 4.43b is a close up photomicrograph of gouge material. Gouge fragments are very angular and roughly equant in shape (i.e. have high sphericity according to Pettijohn’s classification in Figure: 4.23) and have a size of approximately 15 μm in diameter. The surface textures of gouge particles can be clearly seen in Figure: 4.43c which shows indented striations appearing on the face of an angular grain of 20 μm diameter. These scallop like features are most likely indicators of wear.

**Host grains within gouge material**

The fabric of the gouge material is shown in Figure: 4.43a and has several host sized grains caught up in it as described above. Figure: 4.44a shows a single host grain of 160 μm diameter surrounded by angular fragments of <20 μm diameter and very fine material concentrated together in clumps. The single grain is sub-rounded to sub-angular and slightly elongate with a major through going fracture extending across the centre of the grain, parallel to the longest axis. Host grains observed in the gouge material are generally similar in size and shape but do not always show microfractures. Figure: 4.44b shows a close up of another host sized grain caught up in the gouge. This grain is more equant in shape and shows a set of en-echelon curved fractures whose individual slices are preserved in place. Damage in the same orientation as the en echelon fractures extends out into rest of grain, as dark linear features which show no opening.

**Wall rock**

Figure: 4.45a is a photomicrograph showing the wall rock surface at the edge of a fault zone. The plane of the photograph is equivalent to the plane of the fault. This frame highlights the
uniformity of grain size with an average grain diameter of 150 µm. Grains are all equant and most are fairly well rounded. A fine covering of gouge powder adheres to the surface and fills some of the pore space. Pore space which remains between the grains is generally approximately one third the size of the grains themselves and is roughly triangular in shape. Figures: 4.45 b, c show the surface of the fault zone wall in more detail. Pervasive damage through microcracking is observed to occur in the majority of grains. Grains are split up into angular fragments which still lie intact. Further deformation of this area would likely cause these fractured grains disaggregate and become incorporated within the gouge material. The propagation of fracture paths is displayed in Figure: 4.45c as a series of en echelon fractures. There is also a fracture surface parallel to the plane of the photograph where the grain appears to have been sliced in half. The outside rim of the grain coated with fine grained material (possibly hematite). This indicates that the grain coatings are a feature which pre-dates fracturing and that the fracturing and granulation process will most likely scrape off grain coatings and incorporate them into the gouge. This could be a source mechanism for the concentrations of dark brown (hematite) referred to as 'stringers' observed in thin section analysis (see section [4.4.3], Figure: 4.20a).

[4.5.4] Summary of S.E.M. analysis

The gouge material has a typical grain size of between 5-20 µm (on the samples studied) and is commonly composed of very angular fragments which tend to be equant as opposed to elongate in shape. Some surviving original grains are still present in the samples of gouge. These grains are commonly sub-rounded to sub-angular. The fact that large grains are rounded and smaller ones are angular implies that the small ones are large ones which have broken down. (This is consistent with the laser particle size analysis results (section [4.3.4]) where the gouge strands show an absence of large grains implying they have been broken down). Large grains have also been observed with major fractures running right through them.

Scallop marks have been observed on grains in the form of grooves which occur in sub-parallel sets. These may indicate relative intergranular movement between grains. The different sets of grooves on a single grain in different directions may indicate either that many grains were in contact at the time of movement, or that many phases of movement took place after one another. Scallop marks cover the surfaces of small grains in the gouge samples, implying that they have undergone extended phases of movement, or that they were surrounded by many angular grains. In the sample from the edge of the fault zone, large
grains often retain a coating, whereas there is evidence of these grain coatings being scraped off in other grains.

Intact samples of the deformed sandstone show that intact grains tend to be well rounded and equant in shape. The grain size is approximately 140 μm in diameter which agrees well with both laser size data and also data collected in 2-dimensional thin section slices.

Fractures in individual grains are composed of en-echelon arrays which have been exploited or broken through by the fracture plane. Some of these process zones occur without the through going fracture implying that they happened first.

[4.6] Summary of Post Test Data

The data obtained by post mortem geological analysis of samples which have undergone different amounts of deformation have been presented in this chapter. Deformation is expressed as a series of pale white strands that together form a fault zone. The individual strands anastamose in plan and are sub-parallel in longitudinal cross section. Material within strands is reduced in grain size and sorting, and shows evidence of granulation / wear processes. Grain sizes determined independently using three different techniques, show good agreement: laser sizing yields a peak size = 9.97 μm (section [4.3.2] Figure: 4.13), thin section analysis gives a mean size = 10 μm (section [4.4.10], Figure: 4.32), and SEM offers a typical range = 5-10 μm (section [4.5.3], Figure: 4.43). Host rock material is often caught up in the fault zone in discrete “pods” which act like spacers, separating the neighbouring strands. These pods show no grain size reduction, but do show a reduction in pore space and significant, pervasive microcracking damage. Microcracking in the pods has a strong anisotropy in a direction parallel to maximum compressive stress. In contrast, microcracking is very rare in the grains within the gouge strands. Increases in axial strain cause the formation of a greater number of fault zone strands (measured in both hand specimen - Figure: 4.3 and thin section - Figure: 4.26) and a corresponding wider fault zone (again observed in hand specimen - Figure: 4.4 and thin section - Figure: 4.29), rather than greater dislocation on a single fault strand. Increases in confining pressure cause deformation, still in the form of pale granulated strands, to be more distributed throughout the sample, and at high confining pressure the sample deforms by barrelling rather than developing a discrete macroscopic shear zone.

The observations presented above highlight two important points. Firstly, that the individual unit of the deformation structures produced in my samples (i.e. single gouge
strands and microcracking in adjacent wall regions) are identical to those produced in previous laboratory tests on small samples. This implies that the same micromechanisms are active in both cases. The second and most exciting point is that by deforming large samples, I have created fault zones consisting of compound sets of these individual units and have observed the hierarchical development of the complex multiple strands as a function of increasing strain. The next chapter will compare in detail, the complex fault zones I have produced under laboratory conditions, with field examples of deformation band structures.
CHAPTER 5: FIELD OBSERVATIONS OF FAULTS IN SANDSTONE
LOSSIEMOUTH, SCOTLAND

[5.1] Introduction
The motivation for carrying out fieldwork was to find and analyse naturally occurring fault zones in porous sandstone to allow a qualitative and quantitative comparison with laboratory induced deformation features, in order to determine whether or not the laboratory deformation bands are geologically realistic. In Chapter 4, the structural and microstructural characteristics of laboratory induced deformation features were presented in detail. This chapter aims to present comparable structural data from field examples of deformation bands, in order to investigate the features common to both and also identify the key differences between laboratory and field structures. Possible reasons for the similarities and differences observed will be considered in Chapter 6 (Discussion). The characteristics of deformation bands observed in the field have been described previously in the literature (e.g. Antonellini and Aydin 1994 see section [1.5.3] Chapter 1), but the definitions of damage and textures are somewhat subjective making comparisons between those studies and my laboratory work difficult. I have therefore analysed field and laboratory samples in precisely the same manner to allow direct comparison.

The field area selected for study was the Hopeman sandstone exposed near Lossiemouth in North East Scotland. The aims of this fieldwork were not to repeat a comprehensive large-scale mapping study (already carried out by Edwards et al., 1993) but instead, to build on their work by observing in detail the fault structures exposed here and collecting specific structural data in order to allow qualitative and comparison to laboratory data. First, a brief geological history of the region is reviewed, and then the Cummingstown area studied here is described in more detail, including its main structural features. In the next section, data obtained through qualitative and quantitative analysis of thin sections obtained form outcrop samples are presented, and then finally a summary is presented with reference to the analysis of laboratory samples.

[5.2] Brief Geological History of the Field Area: A Review
The field area is situated on the South coast of the Moray Firth near Lossiemouth, North East Scotland (see Figure: 5.1a). The area was recently described by Edwards et al. (1993),
Chapter 5 - Field Observations of Faults in Sandstone

Figure: 5.1a Map of Scotland showing Lossiemouth area studied. Box shows the region shown in figure: 5.1b.

Figure: 5.1b Map of the geology of the Burghead to Lossiemouth region after Edwards et al. (1993).

Figure: 5.1c Map of Lossiemouth fault and Splay fault located near Cummingstown after Edwards et al. (1993).
who investigated the compartmentalisation of the Hopeman sandstone by the faults of the region.

The Hopeman sandstone is a quartz-rich, aeolian sand, dated as Permo-Triassic in age (Peacock et al. 1968), with large-scale cross bedding dipping to the South West. The sand is well sorted and forms a porous quartz cemented fabric. The Burghhead sandstone, thought to be Triassic in age, is a poorly sorted fluviatile yellow sand, containing pebble horizons and commonly carbonate cement (Frostick et al. 1988). The Hopeman and the Burghhead sandstones are in contact across the Lossiemouth Fault (the major fault in the region), which dips steeply to the North (Figure: 5.1b, after Edwards et al. 1993). A splay fault, and the smaller scale structures associated with this fault, are well exposed in the Hopeman Sandstone to the North of the Lossiemouth Fault (Figure: 5.1c). This splay fault strikes East-West, and is the main structural feature of the Cummingstown area. The medium and small scale faulting in the region predominantly shows an East-West striking conjugate fault set dipping to the North and South at 50-90 degrees (Figure: 5.2a). These features strike sub parallel to the splay fault. According to the Andersonian model of faulting, this would be consistent with a vertical maximum compressive stress, an East-West trending intermediate compressive stress and minimum compressive stress (i.e. relative extension) in the North-South direction (Figure: 5.2b). A secondary set of faults strike North-South, and are sub vertical or dip steeply to the East. The relative ages of the faults have been determined (Edwards et al. 1993) by cross cutting relationships, the eldest set being the extensional East-West conjugate set, with a pure dip slip sense of slip. Younger faults of various orientations display wrench displacements rather than conjugate or Riedel sets. The Deformation Bands making up the conjugate sets occur both as compound zones and solitary bands. Solitary bands are thought to be young faults with little offset, whereas compound zones are interpreted as being more mature fault zones accommodating much more strain (Edwards et al. 1993). This observation agrees very well with my laboratory observation that the number of individual strands making up a fault zone is linearly proportional to applied strain (Figure: 4.3, chapter 4). An obvious colour change is observed in the strands with respect to the host rock. Fresh surfaces of strands are pale white as opposed to the buff colour of the host. This may be due to the angular quartz fragments which make up the strands having different optical properties to rounded host grains or grain coatings, which tend to affect the colouring, being sheared off.

According to Frostick et al. (1988) the major and minor faulting in the region occurred predominantly in the Jurassic. In order to estimate the overburden at the time of
Chapter 5 - Field Observations of Faults in Sandstone

Figure: 5.2a Photograph showing North and South dipping, East-West striking fault structures in Hopeman sandstone near Cumingstown [NJ128693]. North (the Moray Firth) is to the right of the photograph which is taken looking due West.

conjugate fault set pattern

Figure: 5.2b Schematic diagram of Andersonian model of faulting in the same orientation as figure: 5.2a.
### Chapter 5 - Field Observations of Faults in Sandstone

#### Southern Moray Firth

<table>
<thead>
<tr>
<th>Age</th>
<th>Southern Moray Firth</th>
</tr>
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<tbody>
<tr>
<td>Jurassic</td>
<td></td>
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<tr>
<td>Pleinsbachian</td>
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<tr>
<td>Sinemurian</td>
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<td>Hettangian</td>
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<td>Triassic</td>
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<td></td>
<td>cherty rock</td>
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<td></td>
<td>Lossiemouth Sandstone</td>
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<td></td>
<td>Burghead Beds</td>
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<tr>
<td>Permian</td>
<td>Hopeman Sandstone</td>
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Figure: 5.3 Schematic diagram showing stratigraphical units (after Frostick et al. 1988) and thickness of beds after i) ii) iii) and v) Stephenson and Gould (1995), and iv) Glennie and Buller (1983).
faulting, I gathered published depth ranges for different stratigraphical units from borehole
and outcrop data. The depths (with references) are presented in Figure: 5.3 and the
maximum overburden due to Triassic and Jurassic sediments is calculated as approximately
300 m. To establish a comparison with laboratory data, the overburden was calculated using
the method below (Wong et al., 1997):

\[(\rho - \rho_w)gz\]  \hspace{1cm} \text{equation: 5.1}

where \(\rho\) is the saturated rock density (\(\sim 2250 \text{ kg.m}^{-3}\)), \(\rho_w\) is the density of water (\(\sim 1000 \text{ kg.m}^{-3}\)), \(g\) is the acceleration due to gravity, \(z\) is the depth in m. In a compressional regime
the effective overburden pressure is analogous to \(\sigma_3 - P\) in a triaxial compression experiment. 300 m sediment gives an effective overburden (i.e. confining pressure) of 3.75 MPa. This is
markedly lower than the confining pressure of 34.5 MPa applied in the deformation tests
presented in the preceding chapters of this study.

[5.3] The Cummingstown locality

[5.3.1] Introduction

The Cummingstown location was chosen for my detailed study because it has very good
three-dimensional exposure due to a horizontal wave cut platform (exposing a plan view
directly comparable to my laboratory plan view) and weathered vertical North-South
striking faults exposing cliff faces and forming caves (exposing cross sectional views
directly comparable with my laboratory section view). Access to the beach was easy from
the village of Cummingstown. The area studied here has Ordnance Survey co-ordinates
[NJ 128693] and is located near the tip of the splay fault described by Edwards et al. (1993)
which runs from co-ordinates [NJ123682] to [NJ126693] along the coast (Figure: 5.1c). In
this tip area there is no single slip plane, but instead the fault splits into a set of compound
zones which together take up the displacement. A sketch of the dip and strike of the
compound zones studied is presented in Figure: 5.4.

[5.3.2] Description of field relations

Deformation is mainly expressed in this area as a conjugate set of deformation bands shown
in Figure: 5.5a with the acute angle bisected by the vertical and the obtuse angle bisected by
the North-South horizontal, as shown in Figure: 5.2b. Deformation bands often occur as
compound fault zones of approximately 50 cm - 100 cm in width. These are particular well
exposed in the “Altar rock” looking West (see Figure: 5.2a). These compound fault zones
Later reference:
zones studied are presented and the locations of photographs and thin section samples are marked for

Figure 5.4 Sketch map of Cunninghamston area studied. The dip and strike of compound deformation

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Figure: 5.5 Photograph showing the exposure of deformation structures in 3-dimensions. a) differential weathering of South and North dipping compound fault zones taken looking East with North to the left; b) showing the anastamosing wavy nature of the fault trace and the parallelism in the North South vertical plane are clearly shown.
are easily identified due to their differential resistance to weathering which contribute to the
topography of features like Altar rock (Figure: 5.2a). Figure: 5.5b shows deformation bands
exposed in 3-dimensions to highlight both anastamosing in plan view (i.e. plane
perpendicular to maximum compressive stress) and sub parallel strands in longitudinal cross
section (i.e. parallel to maximum compressive stress).

Various deformation band structures are clearly seen in the Cumingstown area. Figure: 5.6a shows breached relays on the wave cut platform. Figure: 5.6b shows a single strand occurring between two compound zones which diverges and splits into several separate anastamosing strands before reforming as a single strand. In places, Figure: 5.7, groups of South dipping strands occur as fairly regular spaced linear clusters with apparently undamaged host rock between them in pod/ strand sets. This is very reminiscent of the characteristics seen at a smaller scale in longitudinal thin sections of experimentally deformed samples (see Figure: 4.20a). Figure: 5.8 is a graph of the width distributions of the fault zone and the host rock across the compound fault zone shown in Figure: 5.7. Data was collected as a horizontal transect in a South to North direction. From this graph we can see that the average width of a fault zone in this exposure is approximately 25 mm whereas the width of the intervening 'undamaged' host rock pod is approximately 40 mm, and is always wider than the width of the fault zone. Fault zone pods are always wider than the width of the fault zone compound bands in this example and interestingly tend to follow the same pattern as the fault zone width but shifted off to the North. This may be a consequence of the direction of dip to the South.

[5.4] Thin Section Analysis

[5.4.1] Introduction

Oriented rock samples were collected from various sites along the Cumingstown coastline. These samples are representative of the various types of deformation features exposed in this particular area. Large-area thin sections (75 mm x 50 mm) were made from these samples, and impregnated with blue epoxy resin in exactly the same way as for laboratory samples (see Appendix E). All of these sections were cut in a vertical plane, with a North-South strike, in order to expose the plane perpendicular to the dip of the faults (this is exactly the same plane exposed in the laboratory samples see Figure: 4.19). The thin sections selected for detailed analysis were labelled as follows: (i) L4, collected from the middle of a South-dipping major compound zone approximately 25 cm in width and striking at 266 degrees; (ii) L5, taken from an intermediate region approximately 15 cm to the North
Figure 5.6 Photograph of wave cut platform in Hopeman sandstone showing: a) a breached relay structures; b) a single strand which splits into several distinct strands which anastamose then recombine to give the appearance of a single strand again. A 15cm long pen is shown for scale.
Figure: 5.7 Photograph of a vertical face showing sets of strands clustering together in fairly regular fault zone - host rock patterns.

Figure: 5.8 Graph showing distribution of fault zones and host rock across a vertical plane striking North. The width of Fault Zones and host rock (absent of deformation bands) is plotted in mm as a function of position across the exposure.
of the compound zone sampled in L4, and containing a series of less well developed South
dipping strands; (iii) L6, consisting of an isolated vertical strand striking at 276 degrees,
surrounded by host rock and located between two compound zones approximately 3m away,
and dipping to the South and North respectively. The general locations of these three
samples are marked on the sketch map in Figure: 5.4. Photographs of the sample locations
are shown in Figures: 5.9 a and b. These particular samples were chosen because they
allowed a comparison of the characteristics of individual deformation bands caught up in
different larger scale deformation structures i.e. isolated bands (L6), bands near compound
zones (L5), bands actually inside compound zones (L4).

[5.4.2] Qualitative Description of Thin Sections

This section describes, qualitatively, the deformation features observed in thin sections of
samples L4, L5, and L6. As with the laboratory samples in chapter 4, the fieldwork thin
sections are initially viewed using a Wild Macroscope with low magnification and a wide
field of view. The first striking feature in plane polarised light is the amount of blue epoxy,
i.e. impregnated pore space, apparent in the three samples. Sample L6 is almost completely
dominated by a blue shade, L5 has a fairly even amount of blue and non-coloured areas and
L4 has significantly less blue areas than the other samples. This implies a decreasing
permeability adjacent to the strands studied in the three field samples as a function of
proximity to a major compound fault zone.

Host rock

The areas of blue epoxy impregnation in sample L6 are considered to be host rock due to
their significant distance from major compound fault zones and their even epoxy
impregnation, implying a fairly well-sorted, undamaged fabric. Figure: 5.10a shows a
photomicrograph of the host rock in L6. Most quartz grains are observed to have quartz
overgrowths making many grains fairly angular in shape. Figure: 5.10b is a
photomicrograph taken under cross polarised light which highlights the grain contacts,
which are often straight boundaries, showing that a large proportion of the neighbouring
grains are in contact. There are abundant concavo-convex contacts (i.e. where one grain
encroaches into its neighbour) and also some sutured contacts implying the initial stages of
pressure solution have occurred. These features have no preferred orientation related to a
maximum compressive stress in the vertical direction. In one particular sample, Figure:
5.10c, fluorite cement is present between the quartz grains in a limited region near a gouge
Figure: 5.9 Photograph showing the location of thin section samples: a) compound zone sample L4 and intermediate deformation area sample L5; b) solitary deformation strand exposed in sample L6.
Figure 5.10 Photomicrograph of host rock in sample L6 with magnification of 32x and field of view is 2.7mm x 1.8mm. North is to the right of the frame and vertical is upwards. Photomicrographs are: a) taken under plane polarised light, the pore space (highlighted by the blue colour) is abundant, sorting of grains is good i.e. most grains are of roughly a similar dimension and quartz overgrowths (responsible for the angular shape) can be identified on many grains; b) under crossed polarised light; c) host near a gouge strand highlighting fluorite cement between quartz grains.
Figure: 5.10 c
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strand. This observation is consistent with those of Edwards et al. (1993) who also noted a dramatic reduction in fluorite cement with distance away from the fault zone. They suggest that fluorite solutions gained access during the reactivation of faults which became a network of open conduits and then precipitated adjacent to the faults.

*Isolated gouge strand*

In the hand specimen of sample L6, the deformation zone appeared to be an isolated strand but under the optical macroscope, it clearly consists of two distinct strands, separated by a fault zone pod. The gouge strands contain no blue epoxy resin, however, the central pod and surrounding host rock do show some pore space impregnation with blue epoxy resin (see Figure: 5.11a). Under crossed polarised light the individual grains making up the strands can be identified (see Figure: 5.11b). A high proportion of host-sized grains are preserved, but between them are much smaller, fairly angular quartz fragments filling up the intervening space. The material in the strand appears to be grain-supported *i.e.* surrounded by a network of large grains as opposed to being suspended in a sea of small particles. Large grains in the host rock, pods and gouge strands themselves show little (if any) evidence of microfracturing within grains under optical microscopy. It is possible that the lack of microcracking may be influenced by the relatively low confining pressure estimated at the time of deformation.

*Gouge strands in the intermediate zone*

Sample L5, collected from an intermediate zone adjacent to the major compound zone sampled in L6, is dominated by strands and pods, with no host rock apparent. Fault zone pods show significant pore space, whereas the strands show no evidence of epoxy impregnation (see Figure: 5.12a). The strands, similarly to sample L6 described above, have reduced sorting caused by the addition of smaller grains alongside the host sized grains, but individual strands show variable characteristics. One example is that larger grains in some strands are grain-supported (see Figure: 5.12b) whereas other strands are dominated by matrix-supported grains (see Figure: 5.12c).

*Gouge strands within compound zone*

The first impression of thin section L4 is that it contains a much larger proportion of strand material, and less fault zone pod material, than samples L5 and L6. Small blue epoxy impregnated areas are however still found. The strands are much wider than those in L5 and
Figure: 5.11 Photomicrograph of L6 showing one of the separate strands. North is to the right, vertical is up. Photomicrographs are taken: a) under plane polarised light showing and the absence and presence of blue epoxy in the strands and pod respectively with a field of view 8.7mm x 5.7mm; b) under crossed polarised light with a field of view 2.7mm x 1.8mm. This highlights the reduced sorting in the strand with respect to that in the host due to the addition of smaller sized grains amongst the host sized ones.
Figure: 5.12 Photomicrograph of intermediate sample L5 showing a) the different impregnation of blue epoxy in fault zone pods and strands where field of view is 8.7mm x 5.7mm; b) under crossed polarised light, a strand in sample L5 showing reduced sorting but grain supported fault strands; c) a deformation strand in which larger grain are matrix supported. (b, c, horizontal field of view = 2.7mm)
Figure: 5.12 C

Figure: 5.13 Photomicrograph under crossed polarised light of sample L4 taken from a major compound zone. The frame is completely filled with the image of a single strand showing very poor sorting and matrix supported host sized grains. Field of view is 2.7mm x 1.8mm.
L6, and it is more difficult to distinguish between individual strands. In most strands (see Figure: 5.13), large host-sized grains are matrix-supported \textit{i.e.} completely surrounded by a sea of smaller particles. There is almost no intra-granular microcracking in either the pods or strands. Grains inside the fault strands tend to be less angular than those in the host rock of sample L6 possible due to a relative absence of quartz overgrowths which have probably been sheared off and granulated to make the fine-grained material.

\textbf{[5.4.3] Point Counting - Quantitative analysis}

This section describes the quantitative analysis of thin section samples. The point counting transect technique used here to obtain a quantitative analysis of the fault zone characteristics is identical to that described in sections [4.4.5] and [4.4.6] of Chapter 4. Point count transects of 60 counts were carried out in orientations parallel to fault zone trends in longitudinal thin sections. The locations of transects, with respect to structures within the samples, are shown in Figure: 4.19 and table: 4.4. Transects were carried out in Lossiemouth field samples L4, L5 and L6 described above in section [5.4.1]. The transects studied in each sample depended to some extent on the composition of the individual samples. In sample L6, point count transects were carried out in the host rock (T5), fault zone pod (T2), wall rock (T4) and gouge strands (T8, T9) whereas samples L5 and L4 didn't have any host rock so transects were carried out in the pods (T2) and strands (T8, T9) only. All orientations were measured with respect to the vertical. The diameter of grains, pore space, microfractures, grain angularity and neighbouring grains were all considered. Data was also collected on the relative width of strands and pods.

\textit{Width of pods/strands}

Figure: 5.14 a, b, c show the variation in widths of individual fault zone pods and strands measured from thin sections of samples L6, L5 and L4 respectively. Sample L6 (a) shows that the widths of pods and strands are similar, both ranging between 1000 \(\mu m\) and 3500 \(\mu m\). Sample L5 (b) has strands which are approximately the same width as those in sample L6 but pods which are significantly wider, in the range 3000 \(\mu m\) to 5500 \(\mu m\). Sample L4 has strands and pods which have the largest width range of 1000 \(\mu m\) to 9000 \(\mu m\). Strands tend to be either narrower (1000 to 2000 \(\mu m\)) or wider (7000 to 9000 \(\mu m\)) than fault zone pods which have the intermediate widths of 4000 \(\mu m\) to 7000 \(\mu m\). All three samples examined have fault zone strands of approximately 1-2 mm (or 5-10 grain diameters). Sample L6, which is distal from the compound fault zone, has one pod of comparable width to these
Figure: 5.14a, b, c, Graphs showing the width in microns of pod areas and distinct strands across the sample. Data was collected by measuring separate pods and strands at different positions along their lateral extent. a) data from sample L6 shows a similar width for pods and strands of approximately 2000µm. b) shows that pods in sample L5 are generally wider than strands. c) shows pods which have an intermediate width lying between narrow and wide strands.
Figure: 5.15 Graph of mean pod and strand width for samples L6, L5 and L4. Each point plotted is the mean width derived from the data presented in figure: 5.9 a, b, c. Dotted lines show that widths can be separated into three fields: standard strands (lower section); pods (middle section); wide strands (top section).
strands. Although the widths of fault zone pods in samples L5 and L6 are similar, sample L4 (from the compound fault zone) has some very wide fault strands which are wider than the pods. This minority of observations is therefore different from those on the laboratory tests, all of whom had strands narrower than the pods (section 4.4.9, Figure: 4.30).

Figure: 5.15 shows the mean pod and strand widths across samples L6, L5 and L4. This graph highlights: the differentiation in size between pods and strands in sample L5; the similarity in strand and pod width in sample L6; and the existence of very wide strands in sample L4. On closer examination, it is found that sample L4 has alternate wide and narrow strands as a traverse is taken across the sample. Separating these strands are pods of roughly constant width.

Grain diameter
The grain size distributions within different regions of samples L6, L5 and L4 are displayed on cumulative frequency plots in Figures: 5.16 a, b, and c respectively. In these log-linear plots, distributions with near vertical slopes (resembling a step function) imply very good sorting i.e. essentially all grains fall into a single size range, whereas a shallower slope is interpreted as much poorer sorting with a wide range of grain sizes. Key characteristics displayed by all three samples analysed are as follows: Firstly, strands have a much larger range of grain sizes than those of pod / wall rock / host rock regions, highlighted by the greater extent of data along the x-axis. Secondly, strands have a shallow sloping curve, whereas the pod / wall / host rock regions have much steeper curves, indicating poorer sorting in the strands and good sorting in the pod / wall / host rock. Transects in the host, wall and pod areas display a small but progressive shift of the frequency curve to the left (i.e. towards a smaller grain size), with proximity to the fault zone. Figure: 5.17a shows a comparison of grain size distributions from samples L6, L5 and L4. All the strands follow a similar trend including the range of sizes. On closer inspection of the smaller size fractions (see Figure: 5.17b) it is observed that strands in sample L4 have the highest proportion (approximately 6.5%) of fine grained matrix material (< 10 μm diameter). Sample L5 strands have a lower proportion (at 4%) but in turn have more fine material than sample L6 (at 1%). This correlates well with the observation, described qualitatively in section [5.4.2], of a transition from grain supported strands to matrix supported strands with proximity to the compound fault zone of the area.

Figure: 5.18 shows mean grain size in different regions of the samples, averaged from the data presented in Figure: 5.16. The mean grain size of the host rock observed in sample
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Figure: 5.16 a, b, c, Cumulative frequency - diameter graphs plotted as percentage of sample smaller than a given diameter (μm) on log linear axes for samples L6, L5 and L4 respectively.
Figure: 5.16 C
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Figure 5.17a Cumulative frequency comparing the grain size distributions within gouge strands of samples L6, L5 and L4. Figure 5.17b Close up of small grain fractions already presented in figure 5.17a.
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Figure: 5.18 Graph of mean grain diameter (μm) versus the position or region within the sample. Data (obtained from point count transect analysis) is shown separately for samples L4, L5 and L6. Each transect i.e. point on the graph represents the average of 60 grains so the whole graph represents 660 counts.
L6 is 242 µm. The wall rock adjacent to the fault zone shows a slight reduction in mean grain size to 208 µm and the fault zone pods of sample L6 show a similar magnitude decrease to a value of 196 µm. The differences between the same area in different samples is up to 60 µm, with the intermediate sample L5 having largest pods. The grain size in the strands is significantly reduced, by approximately 1/2 with respect to the fault zone pods for all three samples having values of approximately 100 µm. The largest mean grain size in the strands of 123 µm is observed in sample L4.

Pore space

Figure: 5.19 shows the pore space neighbouring each cross hair grain in different regions of the samples. The majority of pores are smaller than the cross hair grain under consideration. The trends of all sizes of pores are similar, showing highest values i.e. largest amount of pore space, in the host region. The amount of pore space is reduced slightly in the wall rock but the most significant reduction, with respect to the host rock, is observed in the fault zone pods. Data is mainly acquired from sample L6, as this is the only sample which has host rock and wall rock (see section [5.4.3]), but values of pore space in fault zone pods of samples L5 and L4 are also superimposed. Samples L4 and L5, similarly to L6, show the largest proportion of pore space is smaller than the cross hair grain and the smallest proportion of pore space is larger in size than the cross hair grain. However in contrast, the actual numbers of pores per grain in the fault zone pods are much higher for sample L4 and L5 than L6, implying a higher porosity. This could be due to differences in the initial porosity in different areas or alternatively may be due to the late dissolution of whole grains by a brine or aggressive fluid inducing a secondary porosity of crystal “holes”. This dissolution is observed (Edwards et al. 1993) to be greater adjacent to fault strands as opposed to elsewhere. It is most likely caused where the reactivation of faults causes them to open up and produce a temporary plumbing network. Brines or other aggressive fluids then permeate a small distance from the faults into the host rock and dissolve grains there. This effect will obviously be more significant where faults are more dense. No epoxy impregnation is observed in the gouge strands themselves, implying very low permeability and much lower porosity than in the fault zone pods. Also this suggests that the stage when faults were open conduits was a transient phase and not a permanent feature.
Figure: 5.19 Graph showing the neighbouring pore space per grain versus region of sample. Data is split into pores which are larger than (>), equal to (=), or smaller than (<) the cross hair grain. A definite reduction in pore space is identified in the fault zone pods with respect to the host rock of sample L6.

Figure: 5.20 Graph of the neighbours surrounding grains in different regions of the samples.
Microfractures

Very few microfractures are observed (at x10 magnification) in any of the Hopeman sandstone samples studied here. A total of six minor fractures and five major through going fractures (see Figure: 4.24 for definitions) have been identified from all the point count transects carried out. They occur in only six grains out of 300 grains counted so are clearly not a pervasive feature under optical analysis at this magnification. It is possible that microfractures were present at the time of deformation but they have subsequently healed due to chemical precipitation and are no longer visible at the resolution used here.

Neighbours

The environment surrounding each grain was quantified by noting the number and relative size of neighbouring grains. Figure: 5.20 shows the point count data, collected mainly from sample L6, in the form of pores greater than, equal to, or smaller than the size of the cross hair grain for different regions of the sample. Grains in the host region were surrounded on average by 2.2 larger neighbours, 2.1 equal sized neighbours and 1.7 smaller grains. A higher proportion of small grains (3.4) and fewer large grains (1.2), surrounded the grains in the wall zone. In contrast, grains in the fault zone tend to be neighboured by equal numbers of smaller and equal sized grains (2.5) with the number of larger grains (1.2) being similar to that in the wall rock. The neighbouring grains in fault zone pods for samples L4 and L5 show very similar values for all the size groups considered.

[5.5] Summary of Field Observations

This section summarises the information collected from field examples of faulting in porous sandstones. The most striking feature of deformation structures in Cummingstown area is the anastamosing of strands observed in plan view juxtaposed against their parallelism in vertical section. Many pods, of relatively undamaged host rock occur inside the fault zone. They are lens-shaped in plan, but elongate, with sub-parallel sides (sub-parallel to strands) in vertical section.

Strands of similar widths can be found in all samples studied, although sample L4, obtained from within a major compound zone also has very wide, well developed strands. Pods are equal to or wider than strands, except in the case of the unusually wide strands observed in sample L4. A reduction in the mean grain size and the degree of sorting is observed in the strands relative to the fault zone pods and host rock. A transition from
grain-supported strands to matrix-supported strands is observed with proximity to a major compound zone. There is a reduction in pore space measured in fault zone pods with respect to host rock and essentially no pore space is found in the fault zone strands. Very few microcracks exist in any region of the samples.

[5.6] Comparison between field observations with laboratory observations
This section presents the main similarities and contrasts observed between the field examples of faults studied in Cummingstown, Scotland, and those produced in the laboratory under triaxial compression.

Deformation structures
Field exposures and laboratory faults show striking similarities in the anastamosing style of deformation in plan view (see Figures: 5.21 a and b and Figure: 4.2) and the parallelism in longitudinal section (see Figures: 5.22 a and b, and Figure: 4.20a). Features observed at a field scale, for example the relative spacing of bands and host pods in the vertical sections, are mimicked both at a smaller scale in thin sections of these field samples, and in the thin sections of experimental fault zones, implying a degree of scale invariance in the structures.

Individual strands
Grain size reduction and reduction of sorting are observed in both field and laboratory strands with respect to host rock (see Figures: 5.23 a, b and Figure: 5.24 as well as Figures: 4.20a and 4.34). The fabric of the strands is variable but field strands become more similar to laboratory faults as they move from isolated strand towards strands located within a compound fault zone. In field examples of isolated strands distal from major faulting, the gouge strands are poorly developed (see Figure: 5.11b). In the intermediate zone some strands appear more well developed than others (see Figures: 5.12 b and c). Within a major compound zone, strands are wide zones which are very well developed (see Figure: 5.13) and resemble the fabric of laboratory strands most closely (see Figure: 5.23b, and Figure: 4.20b).

The mean grain diameter of strands measured in the field and laboratory are shown together in Figure: 5.24 (see also Figure: 4.34). The absolute grain size of field strands is a factor of 10 greater than that of laboratory strands although both show a significant reduction compared to the host rock. The relative grain size reduction is also greater in the laboratory samples when normalised by the host rock grain size, indicating a significant
Figure 5.21 Photograph of a) anastamosing strands on wave cut platform in field, b) anastamosing gouge strands in lab sample.
Figure: 5.22 a) sub parallel strands in field, b) sub parallel strands in laboratory under cross polars (horizontal field of view = 2.7mm).
Figure: 5.23  a) photomicrograph showing granulation of two strands from field, trending top to bottom (horizontal field of view = 8.7mm)  b) photomicrograph of two strands from laboratory, trending top right to bottom left (horizontal field of view = 4.3mm).
Figure: 5.24 Graph of mean grain diameter (µm) as a function of position in the sample for point count data derived from laboratory and field examples of faulting. Data is presented for laboratory samples deformed to axial strains of 4.2%, 6.34% and 11.22% respectively and field samples taken from varying proximity to a major compound fault zone.
difference. It is possible that granulation is less intense in the field due to the greater strength of the rock and cement with respect to the sandstone deformed in the laboratory. Alternatively, the deformation in the field is estimated to take place at a significantly lower confining pressure than that produced in the laboratory. Previous laboratory studies, e.g. Engelder (1974) and Sammis et al. (1986), have shown that the grain size of a gouge is sensitive to the confining pressure at which it forms and in fact grain size is decreased at higher confining pressures. This would be consistent with the smaller grain size I observe in the laboratory gouge strands formed at higher confining pressure.

Pod / Wall / Host Rocks
The wall rock and fault zone pods in the field samples show a slight reduction in mean grain size with respect to host rock, whereas laboratory faults do not (see Figure: 5.24 and also 4.34). There are several differences in the undeformed Hopeman (field) and Locharbriggs (laboratory) sandstones: grains tend to contact along straight edges in the Hopeman, as opposed to at points in the Locharbriggs, leading to lower porosity of approximately 14% compared to Locharbriggs which has porosity >20%; authigenic quartz overgrowths are prevalent in Hopeman and provide the cementing agent, but overgrowths are almost completely absent in Locharbriggs, which is cemented by hematite grain coatings. This results in Hopeman sandstone having a significantly greater strength than Locharbriggs. This greater strength may result in a smaller number of microfractures and hence less grain size reduction.

Microcracking
There is an apparent lack of microcracking in field samples which contrasts markedly to the extensive microfracture damage observed in laboratory deformed specimens (Figures: 5.11b and 4.38). Although microfractures cannot be seen in the field samples under x10 optical microscopy, there is evidence from other analytical techniques (e.g. cathode luminescence) recently applied to similar field samples (e.g. Fowles and Burley 1994) that they do exist in some cases. It is hoped that future work may repeat such analyses on the Hopeman sandstone in order to investigate whether this is indeed the case. The Hopeman sandstone has pervasive quartz overgrowths which are undoubtedly weaker than whole grains and hence will be easier to fracture. This may result in fracturing being concentrated in the quartz cement and grain edges rather than the intra-granular fracture common in the
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laboratory samples. Again this is a feature that is not visible under the x10 optical microscope but future work using other techniques may shed light on.

[5.7] Conclusions
A field example of fault structures occurring in the porous, quartz rich Hopeman sandstone of Lossiemouth, has been studied. Qualitative and quantitative analysis of deformation features at various scales reveals fault zones consisting of compound zones of multiple deformation bands. The individual pale, white, granulated gouge bands, anastamose in plan and lay parallel along the shearing direction. The reduction in grain size, porosity and sorting in the strands contrasts markedly with the pods separating them, which are relatively undamaged and notably, show no evidence of intra-granular microcracking. These pods tend to be wider than the strands, except in the case of an intensely deformed compound zone where several exceptionally wide strands are observed. Isolated strands are grain supported whereas strands with proximal neighbours are matrix supported, implying a more intense granulation process. Many of these characteristics have been duplicated in the fault structures I have created under laboratory triaxial compression e.g. the spatial complexity of compound strand structures, reduced grain size, sorting and porosity in the individual strands. These compound structures comprising zones of deformation bands have never before been produced in the laboratory. The main contrasts identified between field and laboratory fault zones are the apparent lack of microcracking in the field and the fact that grain size reduction in strands in the field is less efficient than in the laboratory. The similarities and differences between field and laboratory observations will be further compared and discussed in Chapter 6, where a model for the evolution of zones of deformation bands will be presented.
CHAPTER 6: DISCUSSION

[6.1] Introduction
The purpose of this chapter is to synthesise and interpret the field and laboratory results obtained in the preceding three chapters. The overall aim is to obtain a better understanding of the processes acting during the evolution of zones of deformation bands. Firstly, a brief review of laboratory results is presented with a view to correlating the mechanical data presented in Chapter 3 with the structural data presented in Chapter 4. This is followed by an interpretation of the possible mechanisms offering an explanation for the specific features observed. The main features important in a comparison between laboratory data (Chapter 4) and field data (Chapter 5) are then presented. Some systematic differences between field and laboratory results are presented and discussed with regard to possible reasons for the discord. The strain hardening model of deformation band evolution is presented along with the problems of applying such a model to my laboratory samples. I then suggest a new hypothesis, consistent with my observations, offering a mechanical explanation for the evolution of zones of deformation bands. Then finally, the potential influence that deformation structures would have on permeability is discussed.

[6.2] Review and synthesis of main laboratory results
This section presents a brief review and synthesis of the main laboratory results. Macroscopically brittle behaviour (in all but the highest confining pressure tests) can be inferred from the stress-strain curves and the formation of a localised fault plane, itself consisting of a set of gouge strands separated by pods of fractured host rock. The individual gouge strands anastamose in plan but lie parallel to each other in the plane of the slip direction. Strands have reduced grain size, reduced sorting, reduced porosity and some show evidence of shear movement in the form of linear striations on the fault zone surface. Fault zone pods have extensive microcracking but grains which are generally still intact. No deformation strands are observed in the test which was halted prior to dynamic failure, implying that the strands are formed only after dynamic failure. Strain hardening and softening, which occur immediately before failure, may be associated with the development of a population of microcracks observed adjacent to the macroscopic fault plane. The yield stress and ultimate stress are strongly correlated, in a way that appears to be independent of sample structure or applied confining pressure. Bulk sample compaction and dilatancy are
both observed in the volumetric strain curves. Compaction is seen in period before failure corresponding to linear elastic behaviour, and can be explained by elastic closure of pores (suggested by e.g. Paterson 1978). The onset of bulk dilatancy (or at least a reduction in compaction rate) corresponds closely to a departure from linearity at the yield point in the stress-strain curve, and is therefore interpreted as being due to non-elastic irrecoverable micro-mechanical processes. This phase begins prior to dynamic failure i.e. before any deformation strands have been formed, and hence must be explained by another mechanical process. Along with gouge strands, the other obvious microstructural feature in the fault zones are microcracks oriented predominantly in the axial direction which occur in the wall rock adjacent to strands. A population of microcracks with opening displacement perpendicular to maximum stress would each be responsible for local dilatancy, and together would be consistent with bulk sample dilatancy. Such dilatancy may also be expected just at the point where gouge strands are beginning to form due to particles rolling and riding over each other causing 'shear-induced dilatancy' Wong et al. (1997) (see section [1.4.8] and Figure: 1.10 this study).

Increasing applied axial strain following dynamic failure results in a linear increase in the number of distinct strands making up the fault zone (see Figure: 4.3). Axial strain can also be correlated to a degree with total fault zone width (see Figure: 4.4), however, the discrete integer values of number of strands have a much stronger correlation with strain than the continuous measurements of 'width'. This discrete growth of the gouge strands is associated with an essentially constant magnitude of stress, although small individual stress drops can be distinguished in some tests. Liakopoulou-Morris et al. (1994) observed relatively small acoustic emission events during the (constant stress) post-failure frictional sliding phase of laboratory compression tests. The small event rate and constant b-value implied smaller seismic sources and weaker material with respect to the main dynamic failure event and they therefore attributed these fluctuations to the shearing of asperities.

An increase in confining pressure leads to tests showing the traditional hallmarks of the transitional regime between macroscopically brittle and ductile behaviour. A systematic decrease in the magnitude of the stress drop at dynamic failure as a function of increasing confining pressure, correlates to observations of a progressive change from deformation along a single plane to deformation distributed more isotropically throughout sample. Although the macroscopic stress curve for the highest confining pressure test shows behaviour approaching macroscopic 'ductility', a post test analysis of samples reveals that microscopic mechanisms of deformation on a grain scale are brittle. These brittle processes
involve a combination of micro-fracture within grains and frictional sliding resulting in granulation along planes. The individual deformation strand structures produced are very similar to those seen in low confining pressure tests, but the main difference observed at high confining pressure is the spatial organisation of the gouge strands into a radially symmetric and more pervasive pattern. This contrasts markedly with the more localised, well oriented structure of damage at low confining pressure. An increase in confining pressure also appears to produce an increasing number of distinct gouge strands. Bulk sample dilatancy decreases as a function of increased confining pressure which also results in the occurrence of an increasing number of gouge strands. As confining pressure increases, lateral sample expansion is presumably inhibited by the confining fluid itself. ‘Shear enhanced compaction’ (see section [1.4.9] and Figure: 1.11 this study, Wong et al. 1997 and Menendez et al. 1996) may also contribute to the inhibition of overall dilatancy even in the presence of microcracking.

Samples with bedding parallel to length are clearly weaker than those with bedding perpendicular to length, but there is no significant difference identified in the structural features formed due to bedding orientation.

[6.3] Interpretation of results
In this section, the mechanical and structural laboratory results are interpreted. My laboratory tests show decisively a discrete, hierarchical development of more strands and a wider fault zone for increased amounts of axial strain. This agrees with observations of the offset on bedding across deformation band zones in the field (Aydin and Johnson 1983), and is the first time that such a positive comparison has been made with laboratory tests. Significantly, there is a linear relationship between number of strands and applied axial strain, and an absence of strands at the axial strain corresponding to dynamic failure (Figure: 4.3). This relation implies that there were no strands before the point of failure, and that the number of strands then increased linearly as a function of strain during the post-failure period. The implied absence of strands prior to failure was tested by examining a sample from a test halted just prior to failure: no strands had formed. This internal consistency is re-assuring.

A major stress drop is observed at the point of dynamic sample failure. It is suspected that this feature is associated with the development of the initial deformation band although this has not yet been proved. If the stress drop is indeed responsible for the first band, then one may expect the band to be structurally a little different from the subsequent bands
(thought to be associated with much smaller stress drops). No direct evidence has been found in the thin section studied to distinguish between the first and subsequent strands. Future work on thin sections from tests terminated at the point of failure may, however, reveal a subtle difference (not distinguishable above the noise in the signal at present) which could then be investigated for tests at higher strains. One reason suggested for the observed difference is the fact that the first band must break across the whole sample whereas this is not necessary for subsequent bands (Cowie, 1997, pers. comm.). I tentatively propose that the first band may different enough from the subsequent bands to be preferentially exploited and turned into a slip surface when deformation stops producing parallel deformation bands and instead becomes localised onto a single plane (see section [1.5.5] this study and Rudnicki and Rice 1975). I stress that no experimental evidence for this scenario exists at present (perhaps because the applied axial strains in my experiments have been too low) but that it is a very interesting avenue for future investigation.

Small stress drops (generally <5 MPa in magnitude) are observed in some stress-strain curves during the frictional sliding phase following dynamic failure. Stress does not normally recover to its former value after these stress drops through strength recovery as may be expected in stick-slip due to rate and state-dependent friction (e.g. Dieterich 1972). High confining pressure tests often show a small amount of strain hardening associated with the small stress drops in the post-failure curve, but this strain hardening is insufficient to recover all the lost stress. Although stick-slip can be suppressed at low confining pressure (e.g. Byerlee 1970), or when gouge has accumulated (e.g. Brace 1972), the difference between my observations and those of Dieterich (1972) may alternatively be due to the fact that my sample is not sliding repeatedly along one single plane, but by the production of several distinct planes each one locking up after each individual sliding / compaction event. It was suggested in Chapter 3 that stress drops in the macroscopic stress-strain curve, at and beyond dynamic failure, may be related to individual microstructural events e.g. strand formation. Figure: 6.1 is a graph of total number of stress drops versus number of gouge strands (the data is also tabulated in table: 6.1). Although identification of the stress drops is somewhat subjective, there is a strong positive correlation between the number of stress drops and number of strands. This correlation is not a 1:1 which may indicate that each stress drop is not necessarily correlated to the development of a new strand but instead implies that either more than one strand forms for each recognisable stress drop, or some strands form without a recognisable macroscopic stress drop. The number of stress drops is also correlated to the total amount of axial strain, and hence the length of the post-failure
Figure: 6.1 Graph showing the total number of stress drops discernible in the macroscopic stress-strain curves (Figures: 3.1, 3.2) as a function of number of gouge strands (Figures: 4.3, 4.26). A strong positive correlation is observed.
deformation period. Acoustic emission monitoring may help to understand these mechanisms more fully as in Liakopoulou-Morris *et al.* (1994).

Individual gouge strands show grain size reduction and an increase in the angularity of particles with respect to the host rock, implying that the brittle cataclastic mechanisms (grain crushing and grinding) are active in the fault zone. The occurrence of microcracks around the fault zone and in the pods is also evidence of brittle micromechanisms. Macroscopic brittle behaviour is also exhibited by the stress-strain curves with localised faulting in the low confining pressure tests. In contrast, the high confining pressure tests show much more ‘ductile’ deformation *e.g.* this study section [Figure: 3.3d, e], even though the micro-mechanisms remain essentially the same (Wong, 1990).

There is some evidence for movement along these gouge strands, shown by the form of structures (referred to as ‘stringers’, Figure: 4.20c) within the gouge strands, and also due to the occurrence of linear striation features on fault surfaces. The stringer structures do not trend in an appropriate direction, with respect to the shear gouge strands or the applied stress, to be defined as Riedel shears, unless significant rotation (~60 degrees) of local stress has taken place. The orientation and style of the structure within the strands resemble that of the ductile stringers created in the laboratory deformation of natural gouge layers by Logan *et al.* (1981) (see Figure: 6.2). These ductile stringers (see Figure: 4.20c) are undulating features, oriented at approximately 150-180 degrees with respect to the shear direction which are suggested to be a manifestation of displacements along both R1 and R2 shears (the shears themselves are not preserved). Differential amounts of displacement across the gouge strands are thought to account for the undulation. An alternative explanation is that these stringers were formed during unloading, in which case they have the correct orientation to be Riedel shear structures.

The laboratory shear band ‘stringer’ structures are very similar to the generic shear zone features based on field observations, and sketched in Figure: 6.3 after Antonellini *et al.* (1994). These were described as bedding planes which have been smears out and stretched along shear zones. The bedding planes in my samples can be identified through their concentration of dark brown material, thought to be hematite. The stringers observed within the laboratory strands are consistent with hematite which has been smeared out in a similar way to Figure: 6.3, with the greatest displacement occurring near the edges of the gouge strands. This shows a sense of slip along the gouge strands which is consistent with the shortening direction, and also the fact that differential amounts of slip may be occurring inside a single band.
Figure: 6.2 Schematic diagram of the development of a ductile stringer (Logan et al., 1981) showing orientations of shear along the edges of the band (large arrows). The orientations of Riedel shears (R1 and R2 both small arrows) which themselves are not preserved in the fabric of the gouge layer but are thought to deform the stringer as shown.

Figure: 6.3 Sketch of a sample from a shear zone in the Navajo member of the Entrada sandstone Utah (Antonellini et al., 1994). Sketched face is perpendicular to the slip vector and shows bedding planes being smeared out along a shear zone. The fabric created is very similar to that shown in Figure: 6.2 (this study).
[6.4] Discussion

In this section, possible explanations for interesting and puzzling features of the laboratory results presented in section [6.2] and [6.3] are discussed.

[6.4.1] Gouge strand formation

The mean grain size within the gouge strands is reduced to the same degree in a random strand regardless of the total bulk strain applied to a sample (Figure: 4.13a). If the extent of granulation was proportional to the amount of displacement along a strand, then each strand may be accommodating a similar and finite amount of displacement with further deformation being taken up on new strands. This certainly seems plausible from the relationship between number of strands and axial strain and could be interpreted by a strain hardening mechanism. However, Engelder (1974) observed, from studies of simulated gouge generation along a saw-cut sample, that granulation (and hence grain size reduction) is intense at first as a function of sliding but subsequently levels off asymptotically after a finite amount of displacement. Further displacement than has little effect on grain size reduction. If the same is true for granulation in my samples, then either the granulation is still intense, and therefore the grain size will be closely linked to the displacement, or the gouge strands have reached the point where granulation slows down, in which case displacement cannot necessarily be interpreted from the grain size. The latter case would invoke a grinding limit (steady state) where a particularly favourable grain size distribution has been reached by the material in these strands where further displacement along the gouge strands no longer causes intense comminution but may still cause rolling and reorganisation. The grain size may still be developing but mean grain size is essentially constant or changing very slowly.

[6.4.2] Gouge strand evolution

Sorting within the individual strands is reduced with respect to the host rock (section [4.3.2] Figure: 4.12a, b, c; and section [4.4.10], Figure: 4.32a, b, c). The gouge material in these strands has a much wider spread of grain sizes than the host rock and is best fit by a bi-fractal size distribution (section [4.3.2], Figures: 4.16a, b, c). This is consistent with Biegel et al.’s (1989) observations that the grain size distribution of fault gouge can evolve through granulation to a fractal size distribution. The change in grain size distribution has implications for the granulation process and also for changes in the petrophysical properties of the gouge strands with increased granulation. Scott et al. (1994) observed that fine gouge
(or gouge with a fractal size distribution) had a higher value of apparent friction than coarse gouge. If this is the case in my tests, it implies that a reduction in sorting and grain size with granulation may increase the friction within the strands and help them to lock up. The large original grains caught up in the gouge strands show very little microfracturing compared to those in the adjacent fault zone pods which are pervasively fractured (Figure: 4.38). Also, the host grains occurring in the gouge strands result in a matrix-supported fabric (Figures: 1.22c; and 4.20b), whereas fault zone pods are grain-supported (Figures: 1.21c and 4.38). This observation concurs with Sammis et al. (1987) and Biegel et al.'s (1989) theories that grains reach a stable condition when they are surrounded (cushioned) by smaller grains and do not need to fracture anymore hence the steady state grain size. It is also consistent with the laboratory observations of Marone and Scholz (1989) that when grains are reduced to a certain size, the grains actually appear stronger. Grains located in the pods of our fault zones are of a fairly uniform size, have very little pore space or matrix separating them and show extensive microcracking. This observation also corresponds with the ideas of Biegel et al. (1989), who suggested that grains of equivalent size lying next to each other will tend to result in one of the grains fracturing.

[6.4.3] Temporal relationship between microfractures and gouge strands

The orientation of microcracks observed in the fault zone pods, sub-parallel to maximum stress, is consistent, according to Coulomb failure, with the orientation of extensional fractures, which do not show a shear component and are thought to be formed during the dilatant phase of loading (i.e. phase III shown in Figure: 1.8). The Coulomb failure criterion also predicts shear fractures with no tensile component oriented at 030 to σ1, which is consistent with the orientation observed for the gouge strands produced in my tests. Microcracks are commonly observed in the laboratory associated with macroscopic shear fractures during brittle deformation (e.g. Engelder, 1974, Dunn et al., 1973). There has been some debate over the order of formation of microcracks and through-going shear fractures as to whether the cracks form a process zone before the shear fracture (Dunn et al., 1973), or a 'wake' of damage which forms after the shear fracture (Friedman and Logan, 1970). Lockner et al. (1991) showed evidence from the study of Acoustic Emissions that microcrack damage is initially distributed then microcracks cluster and coalesce to form macroscopic shear fractures, hence microcracks come first. Also Menendez et al. (1996) observe significant microcracking in a sample where macroscopic shear failure had not been reached. I observe, in some places near the edge of a gouge strand, evidence of shear
movement along a cracked grain which I interpret as a tensile crack which has subsequently been caught up in the shearing process. I investigated the rotation of such cracked grains occurring within the gouge strands for evidence of timing (i.e. if microcracks pre-dated the shear, they should be rotated). However, the majority of large grains caught up in the gouge strands showed no signs of fracture, most likely due to the cushioning effect of smaller particles mentioned above (section [6.4.2]), so there was insufficient data to come to a firm conclusion.

Microcrack damage in the wall rock extends 5-10 grain diameters out into the host in all samples studied here regardless of the total width of the fault zone. This measure is comparable to studies of the fracture of Coconino sandstone by Engelder (1974), who observed that many microcracks formed after the initiation of a shear plane and are progressively rotated movement along the shear. This implies that the microcracks form early on in the shear zone evolution. It is difficult to imagine that the halo of microcrack damage surrounding my laboratory fault zones forms prior to macroscopic failure, as the system does not yet know at this stage how much axial strain will be applied to the sample, and hence how wide the final fault zone will be. If instead, the gouge strands form after failure (as is concluded in section [6.3]), then the microcrack damage must be ongoing as the fault zone grows in width and therefore may be intimately associated with the formation of each new shear band. In summary, pervasive microcracking may occur during the brittle, inelastic process which causes the dilatancy observed during the strain hardening phase prior to dynamic failure, but the localised microcracking adjacent to the gouge strands is most likely to be an ongoing process closely associated with initial gouge strand formation.

[6.4.4] Evidence for shear along gouge strands
Linear striations on fault surfaces can only be identified in one or two strands in some tests. These strands with striations may be undergoing a different type of slip to the other strands in the same sample. Alternatively, their presence or absence may be due to the actual amount of fine grained material stuck to the fault zones. However, it is difficult to quantify style of slip or the amount of gouge in the fault zone. Strands with striations may have undergone a larger amount of displacement or displacement at a different rate. In this manner they could be compared to slip surfaces (see section [1.5.3]) observed in the field within zones of deformation bands (Antonellini et al., 1994) which accommodate significant amounts of displacement. However, the laboratory features can be easily distinguished from
field slip surfaces due to their different appearance *e.g.* the absence, in the laboratory, of the highly polished surface generally seen in the field.

[6.4.5] The influence of confining pressure

An increase in confining pressure simulates the influence of depth of formation on style of deformation. The main influences of increased confining pressure are the style of the macroscopic stress-strain curves (section [3.3.4], Figure: 3.3a-e) and the distribution of the deformation structures throughout the sample (section [4.2.3], Figure: 4.7a-f). The individual gouge strands are qualitatively similar at low and high confining pressure, having reduced grain size and sorting, but the spatial organisation of the structures is quite different. The structures appear to be much more controlled by the confining pressure than the heterogeneities in the sample itself. The number of strands increases as a function of confining pressure, as previously observed by Bernabé and Brace (1990). At high confining pressure, the volumetric strain curves indicate that samples are compacting throughout loading. This observation, taken together with that of distributed damage in the samples, invokes a comparison with a ‘shear enhanced compaction’ mode of damage presented in section [1.4.9], Figure: 1.11, and *e.g.* Menedez *et al.* (1996). In this damage mode, an initial reduction in porosity, due to high confining pressure, promotes strain hardening which inhibits shear localisation (Figure: 1.11). This interpretation would imply that deformation mode has changed to the cataclastic flow regime due to the influence of confining pressure.

The fact that lots of microcracks are observed near closely spaced gouge strands but few are seen near isolated strands (Figure: 4.22), may imply a critical or characteristic interaction distance for the process. This could explain why microcracks are less obvious in the field as the spacing of strands in the field area studied is generally much larger than that in experimentally formed fault zones. The observation that few microcracks are associated with isolated strands, may add weight to the theory that movement along a strand causes some microcracking. The observation that high intra-granular microcrack density occurs in regions of reduced pore space (*i.e.* fault zone pods), whereas, high porosity regions (*i.e.* the host rock) have very few microcracks implies that high pore space and high densities of microcrack populations may be mutually exclusive. This is intuitively obvious when one considers that upon compression, reduction of pore space is the easiest way to accommodate differential stress and it is only when the pore space has reduced significantly that tensile stress concentrations are high enough to cause grain fracture.
Wong et al. (1997), observed that the transition from brittle localisation to cataclastic flow occurs at an effective pressure which reduces with increasing porosity and increasing grain size. This implies that highly porous, coarse materials are likely to undergo cataclastic flow at relatively low confining pressures.

The most geologically reasonable deformation bands, i.e. those which are organised along a single fault zone as opposed to distributed throughout the sample, are produced in Locharbriggs sandstone at a confining pressure of 34.5 MPa (Figure: 4.2 a-j). Interestingly, this pressure is in the middle of the range (20 - 50 MPa) estimated to be the stress field in the Entrada formation in Arches National Park, under which the deformation bands observed by (Antonellini et al., 1994) were formed. Hence these experimental conditions are a good analogue for natural conditions where certain deformation bands form. It should be noted here that the deformation bands I have produced in the laboratory are identified by their cataclasis (grain crushing) as opposed to particulate flow (grain rolling - see section [1.4.8]) the distinguishing characteristic in some types of field deformation band e.g. Antonellini et al. (1994). Hence my deformation bands belong to a cataclastic granulation band ‘sub-set’ of the ‘global’ deformation band set described by Antonellini et al. (1994) (see section [1.5.3]). An important point is that the observations I make, on the influence of confining pressure on deformation band structures, may imply that any model for deformation band development will only be applicable for a finite depth range. A comparison between laboratory and field structures follows in the next section.

[6.5] Comparison of laboratory structures with field observations

The internal structure of the faults I have produced under laboratory triaxial compression are strikingly similar to that of field examples of deformation bands occurring in e.g. Lossiemouth, Scotland (see Chapter 5) and other field areas (see e.g. Aydin (1978), Underhill and Woodcock (1987), Antonellini et al. (1994). The laboratory test carried out at 5000 psi (34.5 MPa) confining pressure produce fault zones which resemble the field examples studied here most strongly. The characteristics of the single strands and the style of the compound deformation zones on a macroscopic scale are very similar. Both field and laboratory faults show macroscopic deformation in the form of anastamosing strands which lay parallel in the direction of shear (Figures: 5.21a, b; 5.22a, b; 5.23a, b). In general these strands are separated by pods of host rock which are wider than the strands themselves, the exception being where strand density is very high. The grain size reduction and reduced sorting in the individual gouge strands observed in the laboratory faults is also observed in
the field examples although the field strands appear to be slightly less intensely deformed (Figure: 5.24). A reduction in permeability is observed in the field (Figure: 5.10a) and laboratory (Figure: 4.36) gouge strands as well as a reduction in pore space in the fault zone pods separating the gouge strands and the wall rock adjacent to the compound fault zone.

It should be noted here that my samples have a finite size (grain: sample size ratio of 600: 1) and if scaled up to the earth would yield extremely large grains which are quite clearly not observed. This will most likely affect the deformation structures produced and how they interact, although in what manner we do not fully understand at present. However, these samples are larger than the majority of those used previously and hence may significantly improve our knowledge of grain size scaling by moving towards geological scales albeit by a small step.

[6.6] Contradictions between field and laboratory observations with possible explanations

My laboratory tests are carried out at a nominal strain rate of $5 \times 10^{-6} \text{ s}^{-1}$. Although this is at the lower end of the range of strain rates commonly used in dynamic faulting experiments it is several orders of magnitude faster than geological rates e.g. ocean ridge spreading rates $\sim 10^{-12} \text{ s}^{-1}$, typical basin formation rates $\sim 10^{-16} \text{ s}^{-1}$ (Ranalli, 1995). This will almost certainly have an effect but its investigation is beyond the scope of this study (see e.g. Scholz, 1990).

Cataclasis in the field examples of gouge strands is less intense than that observed in the laboratory (Figure: 5.24). One possible explanation for this is that the Hopeman sandstone is stronger than the Locharbriggs sandstone, therefore grain size will be reduced less rapidly. Another point is that the Hopeman sandstone has abundant quartz overgrowths, which are a potential source of gouge material since it is easier to shear off overgrowths than to create new fractures in whole grains. It is also possible that the field fault zones underwent less total displacement along the individual strands, although this is difficult to prove from direct observations.

In the field, individual strands tend to have widths between, 1-3 mm which are narrower than the fault zone pods of width 2-7 mm (Figure: 5.15). However, in the compound fault zone of concentrated deformation, several wide strands of width ~9 mm, hence wider than the pods, exist. These wide strands appear to be more ‘mature’ showing evidence of intense granulation and the larger grains in the strands are supported by a fine grained matrix material. This is similar to gouge strands formed in the laboratory. The isolated gouge strands formed in the field, where deformation is less concentrated, are less
intensely granulated and have their larger host grains in the strands supported by grains of a comparable or slightly smaller size i.e. 'grain supported'. This observation is in contrast to those of laboratory strands.

One of the major differences between field and laboratory observations is that microcracking is often not obvious in the field near deformation bands, whereas in the laboratory it is a pervasive feature. This may be due, in the field, to subsequent healing (i.e. precipitation of silica into the former microcracks), as observed by Lloyd and Knipe (1992), which makes the features harder to identify. Microcracking adjacent to deformation bands has been observed under optical microscope in field (Figure: 5b in Antonellini et al., 1994) in St. Peter sandstone and the Aztec sandstone of the Valley of Fire, Nevada. Fowles and Burley (1994) identified microcracks and evidence for healing in Penrith sandstone (not visible using optical microscopy) by using a cathodo-luminescence technique which highlights the slight difference in silica composition in the healed microcrack.

The fact that the tests are dry and of short duration may help to explain the differences between the micro-structures observed in the laboratory and the field samples. The lack of fluids and the absence of time dependent chemical effects preclude dissolution and precipitation of silica which are the important processes required for healing of microfractures, as seen in the field (e.g. Lloyd and Knipe, 1992).

An alternative explanation is that the abundance of microcracks in the laboratory and apparent absence in the field may also be a function of the differences in the loading conditions (Rick Sibson, 1997, pers.com.). In a dry triaxial experiment the system is in 3-D compression (comparable to a thrust system). In a standard laboratory test, a rock is failed by maintaining a constant confining pressure, and increasing the axial (maximum) stress. This causes an increase in mean stress ($\sigma =1/3(\sigma_1 +\sigma_2+\sigma_3)$), which induces microcrack formation parallel to the axis of maximum stress. In contrast, normal faults in the field (the geological environment where deformation bands are commonly observed) are formed by decreasing $\sigma_1$ and maintaining $\sigma_1$ constant i.e. the mean stress decreases. Microcracks are unlikely to form under these conditions as the stress is lower than that attained previously. If an increase in mean stress is the condition for microfracturing in the laboratory, then the reduction of mean stress in the field could account for the relative absence of microcracks. There is no independent evidence for this although it is an interesting scenario.
[6.7] Strain hardening model of deformation band formation

The majority of the structural evidence from my laboratory tests presented above agrees with field observations of zones of deformation bands (e.g. this study Chapter 5, Aydin and Johnson 1978). Aydin and Johnson (1983) invoked the theory of strain hardening materials, developed by Rudnicki and Rice (1975), to explain why the evolution of damage requires the formation of new strands from field observations. A summary of their ideas are presented in Chapter 1, section [1.5.5]. Although my structural data is consistent with this theory, one problem with applying such a theory to these laboratory experiments is that it is inconsistent with the form of the macroscopic load curves (Figures: 3.1, 3.2). Strain hardening occurs pre-failure then a stress drop occurs at macroscopic failure before essentially constant stress i.e. frictional sliding as further strain is accumulated. I believe that the gouge strands formed post failure due to the observation that no strands exist in a test halted just prior to failure (Figure: 3.1a). There is no clear evidence in the stress-strain curves for a macroscopic strain hardening episode (or episodes) after failure when the strands are thought to be forming. Therefore it would appear that invoking a simple strain hardening model is not sufficient to explain the data obtained during deformation band formation in the laboratory. In the following two sections, the outstanding problems in explaining fault zone evolution are listed then a new conceptual model is presented which addresses these problems.

[6.8] Several questions must be addressed before constructing a model:

*Does each strand accommodate a constant displacement?* The direct correlation between the number of strands and applied strain (Figures: 4.3, 4.26) would imply so. Grain size data does support this idea but caution is required when attempting to infer displacement from grain size. Certainly, random strands appear to have undergone a similar amount of granulation, and the resultant grain size distribution is comparable in each case. However, Engelder (1974) observed, in the laboratory, that initially grain size decreases rapidly due to shear displacement but subsequently this reduction levels off asymptotically towards a near constant grain size. Therefore in the initial phase of rapid change, the amount of displacement may be closely correlated to grain size reduction. However, when granulation slows down and grain size tends asymptotically to a mean value, a wide range of displacements may result in very similar mean grain sizes.
Could strands form prior to failure? There was no evidence of strands in the test stopped immediately prior to dynamic failure, so the answer here is clearly no, at least under the test conditions presented.

Is strain hardening / softening necessary for more strands to form? The answer here appears to be yes, since it is difficult otherwise to suggest a mechanism for deformation to swap to another area and fault previously intact host rock in preference to developing the existing fault strand. It is possible that strain hardening is not the mechanism by which deformation is forced to concentrate on a new strand and another factor is involved e.g. boundary effects of the sample stopping propagation when the strand reaches the edge of the sample.

Why is strain hardening not observed in the stress-strain curves? One explanation of the behaviour in my tests, is that the gouge strands may be forming due to local strain hardening and softening processes on a small scale, which is below the resolution of the macroscopic stress-strain curves. This would imply that the macroscopic stress required to continue slip along an existing strand and that needed to form another strand is comparable. Macroscopic stress-strain curves are the summation of the bulk properties of the sample and after failure, the localised stresses within the fault zone may be significantly different (e.g. much larger and much smaller) compared to the stress measured on the ends of the sample (e.g. Jaeger and Cook, 1979; Rice, 1992). The implication is that within a strand, the local stress concentrations could approach the failure stress as a new strand was forming, although the macroscopic stress measured would never necessarily reach dynamic failure stress. An alternative explanation is that more than one band is active at a given time, although they are at different stages of evolution i.e. one strand is locking up whilst another is forming. Assuming that the loading curve of each individual strand is similar but out of phase, this could result in local strain hardening (from the 'dying' strand) and strain softening (from the 'new' strand) which cancel each other out on the macroscopic load curve (see schematic Figure: 6.4).

A major stress drop is observed at failure. Why are the following stress drops much smaller in magnitude? The stress drop at dynamic failure is significantly larger in magnitude than the others because it is associated with macro sample failure. If the major stress drop at failure is related to the first gouge strand initiating, then it is surprising that the other strands which form post failure, and are indistinguishable from the initial strand, express their initiation so differently. However, the first strand simply needs to be less stiff than the intact matrix - which is highly likely - for this to be the case. This could be
Figure: 6.4 Sketch illustrating the idea local strain hardening and strain softening may cancel each other out and hence not be observed in the macroscopic load curve.
explained by supposing that when the initial strand forms, the macroscopic stress curve records all the information about this one strand forming. When the second strand forms, the sample is already broken, so the macroscopic stress curve is not truly representative of what is happening at the second strand due to the stress concentration / shielding caused by the presence of the first strand. Rather, it is either still measuring the frictional sliding of the primary strand, or a combination of the first strand moving / locking up and the second strand initiating, which either way results in a fairly constant macroscopic stress level. The major stress drop is almost certainly associated with the formation of a new slip surface which locks up immediately after failure due to a local hardening process (compaction and loss of porosity) then deformation must subsequently step off onto a new gouge strand and so on. However this contradicts all the ideas of a slip surface accommodating large amounts of displacement.

[6.9] A conceptual model for evolution of zones of deformation bands

A conceptual model for the evolution of zones of deformation bands is now presented, based on a synthesis of laboratory and field data discussed in the preceding sections. Figure: 6.5 is a series of schematic diagrams showing the structural and mechanical response to the proposed model of evolution. [1] Prior to the formation of the initial strand, microcracking is associated with the strain hardening observed in the macroscopic stress-strain curve. Failure occurs by an increase in differential stress and possibly also a reduction in sample cohesion due to the microfracturing. [2] The initial strand to form is associated with macroscopic failure of the sample, and hence is correlated with a significant stress drop. Microfracturing may be associated with movement on this new fault. New microcracking has been observed in previously unfractured wall rock due to frictional sliding along a simulated fault (Teufel, 1981). Teufel (1981), explains these "microscopic feather fractures" by invoking Johnson's (1975) theory that an increased stress concentration locally leads to tensile cracks forming adjacent to the shear band.

[3] This initial strand loads the surrounding area due to elastic forces. If the stress field induced in response to the first faulting event is consistent with elastic crack models, then it can be represented by the solution calculated by Kostrov and Das (1982) in Figure: 1.13b (after Das and Scholz (1981)). The elastic crack model (summarised in section [1.4.10]) predicts a reduction in stress on the fault itself and in the region immediately adjacent (Figure: 1.13b). However, also predicted is a significant increase in stress parallel to the original fault and a finite distance away (Figure: 1.13c). [4] The loading caused by
Explanation of the [6] stages of evolution is given in the main text (section [69]).

2 Regions X - the initial strain to form a lattice distance away from the initial strain; Y - the initial strain corresponding to the sample: (b) the macroscopic stress-strain curve: (c) and (d) the contour plots of failure envelope proposed model of deformation band evolution. Each stage shows a sketch of a) the structure inside the region. 6.5 Series of schematic diagrams showing the structural and mechanical response to the

Region X

STRESS CRIERION

AND FAILURE

Region Y

MACROSCOPIC

\[ \sigma \]
the initial strand may or may not enhance microcracking locally through mechanical softening but will certainly bring these regions closer to failure. New microcracks could be thought of as a combination of the 'wake' of damage related to the first strand, and the 'process zone' of the new strand.

[5] The initial gouge strand deforms through comminution involving granulation and therefore grain size reduction. The frictional properties within the gouge are likely to evolve to a different state from the host rock due to the change in grain size distribution and according to Scott et al. (1994) friction will likely increase. At this stage the stress on the initial strand has decreased (see Figure: 1.13b) and the frictional strength of the strand has increased, therefore continued displacement is difficult and the strand is likely to lock up. At the same time, stress has been transmitted into the surrounding area to produce very high but very localised stress concentrations in regions which have already been mechanically weakened and hence are closer to failure.

[6] At a certain point it becomes energetically favourable to initiate a new fault strand rather than slip on the existing one. On a local scale this will most likely involve a stress drop although the macroscopic stress-strain curve will not necessarily show a strain hardening or a re-loading to the maximum stress as the first strand does not need to harden by a large magnitude before a second strand is formed. The location of the new strand will be in the area near to the original strand which has experienced deformation / enhanced loading due to the initial strand formation but not immediately adjacent.

[7] The second strand, which will now be sliding frictionally, may then load the area around itself, weakening it in a similar way to the original strand before initiating a new strand. [8] An alternative scenario is that the 3rd strand is located in the complementary region to the second strand (on the other side of the fault) which was weakened by the initial strand. It is suggested this process is ongoing throughout the frictional sliding phase. The number of strands may not depend solely on the amount of bulk strain accommodated but also on the rate of this loading and the interplay between the increase in friction in the early strands and the reduction in strength of the area adjacent. The stress drops due to the formation of other strands are not always visible in the macroscopic stress curve.

In summary, this model can account for: the mechanism by which deformation shifts from one strand to another; why the stress drop at failure is much larger than any subsequent stress drops; why no strain hardening is observed in the macroscopic stress-strain curves; the observed separation of parallel strands by a finite distance; and microcracking which can be a combination of process zone and wake damage.
6.10 Influences on permeability

One of the main applications of this study is the effect of the structures observed on the flow of fluids and contaminants in such structures. Although permeability has not been directly measured in the rock deformation tests presented, the microstructural studies carried out highlight several features which may have a significant effect on fluid flow in a deformed sample.

Gouge strands are possibly the most significant feature which would be expected to inhibit fluid migration across the fault. The reduction in grain size, along with a reduction in sorting, will produce a much better packing with smaller pores - hence reducing total overall porosity - the smaller pore throats produced (i.e. smaller \( r \) term in equation: 1.8) will then reduce the connection between pores, and hence reduce permeability according to Poiseuille flow (equation:1.8, section [1.6.3] and Figure: 1.23). Due to the increased number of strands with increased strain, if each strand reduces permeability by a certain amount, then permeability reduction will be linked to applied strain. The laterally continuous pods of host sized grains caught up in the fault zone show a reduction in pore space with respect to the host rock. Fluid is still getting into the pods, but the permeability is still likely to be severely diminished with respect to the host.

A large number of microcracks, many of which extended across more than one grain are also prevalent in the host rock pods introducing a new secondary porosity. They are predominantly oriented in the axial direction and are observed, in the thin sections impregnated with blue epoxy, to be filled with blue resin. This implies that the fractures are open (although whether they were open while the sample was under pressure cannot be ascertained) and that they may in fact enhance permeability in the axial direction. These pods are lens like in cross section and appear to extend laterally throughout the fault zone. They are bounded by low permeability gouge strands so could be envisaged as a series of flow pipes with lens shaped cross sections. There would be little communication of fluid between neighbouring pipes and therefore fluids would have to enter the fault zone at its ends. This may or may not be reasonable in a geological environment but there remains the possibility of flow parallel to the fault zone whereas flow across the fault zone would most likely be significantly reduced.

6.11 Summary

In this Chapter I have synthesised and discussed the results obtained in the preceding three chapters. The micromechanical processes active under different test conditions have been
interpreted from mechanical and structural information. Several outstanding points which
cannot be pinned down at present have been discussed. I have demonstrated that laboratory
faults strongly resemble field faults and where the two differ significantly I have been able
to offer possible explanations. I highlighted the problems with the existing strain hardening
model, generally invoked to explain the evolution of zones of deformation bands, and
proposed a new conceptual model for structural evolution which is entirely consistent with
my observations. This model invokes the theory that the initial strand to form sets up a
stress field which loads the elements near but not adjacent to itself, weakening them and
bringing them nearer to failure whilst the initial strand gradually locks up due to increased
friction. This offers a mechanism by which deformation can swap to a new strand rather
than sliding on an existing one without requiring macroscopic strain hardening. An
interesting point is that a model using the same assumptions has been used to offer an
explanation to off-fault aftershocks associated with some earthquakes. Further work is
necessary to determine whether this model is robust for the evolution of deformation band
structures on a wide range of scales but it certainly accounts for the observations in my
experiments. The following chapter draws the main conclusions from this thesis and gives
recommendations for further work.
This thesis presents research into the deformation of a porous sandstone in the laboratory. The mechanical and structural results of a series of triaxial rock mechanics tests carried out on large scale samples have been presented, interpreted and discussed with reference to field studies. The main conclusions are as follows:

1) Zones of deformation bands strongly resembling natural examples of faulting in porous sandstone have been produced in the laboratory for the first time.

2) An increase in strain results in a linear increase in the number of individual strands which make up the fault zone. Hence the hierarchical development of deformation band structures, previously only inferred from field studies, has actually been observed under laboratory conditions.

3) The number of individual strands which form the fault zone is strongly correlated to number of stress drops observed in the post-failure period of the macroscopic stress-strain curve.

4) The style of deformation bands is highly sensitive to the confining pressure at which they are produced. An increase in both the complexity and the numbers of strands formed is closely associated with an increase in confining pressure.

5) Individual deformation bands are zones of intense comminution and dense packing hence have a high potential for reducing permeability. However, between adjacent deformation bands, there develop pervasively microfractured zones which may in fact enhance permeability.

Recommendations for future work

It is clear from this study, that the laboratory deformation of large scale samples yields a new and very interesting insight into the evolution of fault structures in a highly porous sandstone. Although this study by no means answers all the questions regarding deformation band formation, it does indicate that by using large rock samples it is possible to reproduce realistic deformation bands structures in the laboratory and hence opens the door for further investigation. A finite amount of experimental work is possible in the duration of a PhD and
therefore perhaps this study raises as many questions as answers. I feel that several interesting points are certainly worth pursuing to pin down particular aspects and mechanisms of deformation band formation which have until now remained elusive:

The temporal and spatial evolution of zones of deformation bands
The study of acoustic emissions may be useful in determining the spatial evolution of microcracks and shear bands. Due to the large sample size it should be possible to position six to eight sensors around the sample. Such coverage could potentially give good location information on individual events and hence distinguish different areas of activity. The spacing of individual deformation strands is close to the resolution limit of acoustic emission location, however, at the very least, a migration of activity away from the initial damage zone with time should be discernible. Correlation with post test structural analysis may determine which individual strand was the first to develop and whether this first strand has unique structural characteristics.

Which came first - the microcrack or the shear band?
A possible method for determining the relative timing of microcracks and shear bands, is the use of moment tensor analysis of acoustic emission waveforms. In a broadly similar way to the determination of fault plane solutions from earthquake moment tensors, it is possible, in theory, to distinguish the orientation and type (shear or tensile) of microcrack event. It may therefore be possible to distinguish axial microcracks occurring in fault zone pods, from inclined shear bands and hence determine whether the two main features formed in my laboratory tests are continuous or sporadic.

Is the co-seismic shear stress model invoked in Chapter 6 reasonable?
The model may be tested by determining the magnitude of acoustic events, in particular the stress drop associated with dynamic failure. This could be used to determine the magnitude of the induced stress field and the distance from the initial fault at which stress is maximised (Das and Scholz 1981). Also, the acoustic emission magnitude would yield an estimate of the size of the initial rupture ('length of the crack') and hence one could test whether the subsequent strand to form was located at a perpendicular distance of approximately one crack length as predicted by the elastic crack model (Stein and Lisowski, 1983).
Do zones of deformation bands form in 'wet tests'?  
All the deformation tests described in this study have been carried out under dry conditions. The presence of a pore fluid will undoubtedly affect deformation. Whether a fluid will simply change the stress conditions under which deformation bands develop (by means of the effective stress law) or transform the structures produced is an interesting question. It may be answered by repeating my tests under wet conditions at both ambient and enhanced pressure conditions.

A Final Comment  
Throughout my study I have inferred that the deformation features I produced have structural characteristics which may strongly influence permeability. Perhaps the most important future step, is therefore to actually monitor permeability in the laboratory during the production of these peculiar structures known as deformation bands.
REFERENCES:


References


Appendix A: Equipment Log

Experimental work was carried out in the periods 6/94-9/94, 11/94-12/94, 2/95-3/95, 11/96-2/97. Major repairs to deformation equipment with dates are presented below.

[3/95] Rupture of heat exchanger resulted in cooling water entering the hydraulic oil and causing an emulsion which circulated throughout the system.

[4/95-6/95] Clean up operation involved removing and cleaning servo valves, emptying and cleaning pump unit, flushing out fluid intensifiers by using alternative hydraulic source.

[1/95-2/95] Malfunction of chiller which cools water from heat exchanger required new chiller to be found, plumbed in and setup [6/95].

Motor in power pack malfunctioned and was replaced. Main O-ring seal in new motor extruded resulting in a significant drop in hydraulic pressure and subsequently ruptured. This was replaced three times [4/96, 10/96, 1/97] which involved completely dismantling power pack and motor.
APPENDIX B: GLOSSARY

Anastamose interweave
BDT brittle-ductile transition
B.N.C. cable connectors
C.P. confining pressure
Compaction volume reduction
Critical stress stress where behaviour first becomes non-linear
D.P.M. digital peak meter
Dilatancy volume increase with respect to behaviour expected for an elastic material
Disaggregate decompose a rock sample into individual grains
Endpiece steel circular faced pieces contacting with sample
E.S.H. Testing manufacturing company responsible for upgrade of electronic control system and test software.
Flocculate sticking together of very fine particles
Gouge fine grained granulated rock flour formed during
Host rock undeformed part of sample
Intergranular microcracking between grains
Intersonde suppliers of l.v.d.t.
Intragranular microcracking within grains
Keithley manufacturers of digital monitors
L.V.D.T. linear variable differential transducer
MOOG suppliers of servo valves
M.Pa. MegaPascals (S.I. unit of pressure)
P.A.C. Physical Acoustic Corporation
P.C. personal computer
P.F.P. pore fluid pressure
Platens steel circular faced pieces contacting with sample
P.S.I. pounds per square inch (imperial unit of fluid pressure)
R.D.P. manufacturers of transducers
R.I. Refractive Index
Run I.D. test identification
S.E.M. Scanning Electron Microscopy
Shear enhanced compaction reduction in sample volume promoted by shearing
Shear enhanced dilation increase in sample volume promoted by shearing
Slickensides smooth, polished, shiny, fault slip surfaces usually having linear striations
Sonicate shaking technique used to disaggregate samples
Step discontinuity in a fault surface perpendicular to orientation of striations
Strike the orientation, in the horizontal plane, perpendicular to the direction of maximum dip of a dipping plane.
Wall rock host material on the edge of a fault zone
APPENDIX C: BIG RIG OPERATION MANUAL

Karen Mair February 1995

CONTENTS:

[1] Bringing Rig Up Under Stroke Control
[2] Loading a Rock Sample
[3] Confining Pressure Oil Intensifier
[5] Creating /Editing a Block Programme
[6] Limit Detectors/ Overload Trips
[7] Running a Test
[8] Logging Data Whilst Running a Test
[9] Ending a Test
[10] Unloading a Rock Sample
[12] Conventions
[13] Operating conditions/ trouble shooting

Note: Each numbered section should be read and understood from the beginning to the end before any operation of big rig is attempted.
1) Turn on water recirculating pump in white cabinet outside the Press Room. Pump switch is on back wall at the left hand side directly behind the pump. SWITCH to 1 then PRESS GREEN BUTTON. Check pump is operating by touching pipe work to feel vibrations, also check the pressure gauge at the bottom right hand side beside the water recirculating pump. This gauge should read approximately 0.5 bars.

2) Turn on chiller (outside building) using switch at back right hand side. ON position is 1, OFF is 0.

3) Switch on the main power supply to the oil recirculating pump. This is the major switch on the right hand side wall by the flow meters in white pump cabinet. Move the lever UP and OVER to ON. At this point the 4 red lights (main line filter, return line filter, oil temperature and oil level) on the front of the control box (the grey box above the pump) should go on. DO NOT TOUCH any of the switches above this set of lights. Now the water should be running through the chiller and the pump should be ready to control from Main control cabinet.

4) NB: For computer control the relay panel at the back left of this control box has two switches - the bottom left Blue one is marked L and R (Local and Remote), the middle right hand side has a small black switch slider marked M and A. For Computer control these should be set at R (Remote) and M. DO NOT TOUCH these switches.

5) Before starting machine, check that the ZERO pots on the stroke amplifier, load amplifier 1, and load amplifier 2 on the 2nd rack down are all set at 5 (i.e. their central positions). Set all Amplifier ranges to maximum.

6) Check all hydraulic lines are secured in containers to prevent spillage when machine is activated.

7) Never change hardware range settings when machine pressure is ON. This may be dangerous. In the least case the limit detector will trip out and overload will follow.

8) In the case of an EMERGENCY, PRESS RED PRESSURE OFF then PUMP OFF BUTTONS (both located underneath the control switch on hydraulic control unit 4th rack down, left hand side).

9) Ensure that the position of the PRESSURE OFF and PUMP OFF buttons are noted and remembered.

10) TURN ON MAINS POWER to the Control Cabinet and computer (inside control room) at the back wall (two 3 pin plugs).

11) Inspect the position of the key at the left hand side of the Hydraulic Control Unit (Fourth rack down). SWITCH this KEY to MANUAL.

12) SWITCH ON POWER, top right hand side rocker switch on the Control Cabinet. At this point, various Red lights will light on the Hydraulic Control Unit and the Limit Detector next to it on the 4th rack down.

13) RESET the control cabinet by pressing the RED PUMP OFF BUTTON once. This acts as an OFF switch and also the RESET switch. All the Red lights, except possibly the Control Loop light will then go OFF. The Control Loop light is lit because you have not selected a control mode in which to run the machine.

14) Select the MODE OF OPERATION by going to the third rack down labelled "STROKE, STRAIN & LOAD SCAs".

15) If limit detector lights are lit these may be reset by pressing BLACK RESET BUTTON at top of limit detector (see section [6] "limit detectors/overloads" for further details ) then pressing the RED PUMP OFF button once more.

16) To select STROKE SCA, PRESS DOWN the TOP left hand TOGGLE switch, marked CANCEL or SELECT. If, when you attempt to select this servo controller, (i.e. the STROKE SCA, left hand side third rack down) you have a RED flashing light, this means that the error detector at the top of
this servo is out of range. If so, then dial the mean level pot (i.e. the top pot of the three) until the RED needle sits at the middle junction of the 2 red arrows. When it is in range the flashing RED light will go out and the GREEN light above the words "CONTROL MODE" will light up. This means that the STROKE SERVO Controller is now within range and locked into the machine.

17) When adjusting the Servo Controller, i.e. getting the GREEN CONTROL MODE light up, you should move the mean level pot very slightly CLOCKWISE from middle point. (This is to ensure the piston retracts as soon as the pump is engaged). Clockwise on the Servo Controller stroke SCA mean level pot i.e. the top of the three, means that the piston is retracting (going up).

18) Because it has already returned an error signal to the Hydraulics Control this must be RESET.

19) RESET by pressing the bottom RED PUMP OFF BUTTON. All the RED lights on the left hand side should now be OFF and a YELLOW "CLEAR" light should come up above the switch key.

20) To start the pump, PRESS AND HOLD IN GREEN PUMP BUTTON, (top left of the hydraulic controls), UNTIL hearing the pump ENGAGE outside. IMMEDIATELY press the GREEN PRESSURE ON BUTTON, top right hand side on the Hydraulic Control Panel. Pressing the Pump Button starts the pump, but the oil is being recirculated within the pump body only and not through the cooler unit attached to the back of the pump. Therefore you must press the Pressure Button as soon as possible to get the oil recirculating - a) through the machine and b) through the Cooler Unit at the back of the pump.

21) The piston can now be moved manually under stroke control by turning stroke s.c.a. MEAN LEVEL POT. Clockwise is for UP, anti-clockwise is for DOWN.

[2] LOADING A ROCK SAMPLE

1) Rig should be in three parts i.e. press, pressure vessel suspended by pulley system and base on trolley midway between. Blue confining pressure hose should be disconnected from pressure vessel. If pusher was used during last unloading it must be removed by lowering pressure vessel down onto wood board then replacing with cap piece (see section [10] "unloading a rock sample" for complete procedure). Cap piece must be carefully located and screwed into position. Pressure vessel can then be hoisted upwards using manual chain hoist until clearance is sufficient to allow trolley and sample to be located underneath.

2) Rock sample must be sleeved in rubber jacket then steel spacer(s) added if required. Check Appendix [a] for maximum and minimum sample lengths / piston throws / spacer thickness. If endpiece lubrication is required, bottom surface of steel (i.e. that not in contact with sample) should be greased then melinex disc added. If no extra spacer is used the clean side of the melinex will contact the sample directly. Space of approximately 15mm should remain at top and bottom of sleeve.

3) Grease outer rim of bottom endpiece to prevent oil leakage into sample then carefully lower stack assembly onto bottom endpiece (and also surface if endpiece lubrication is required). Add top platen endpiece greased round rim only. NB: Problem has been encountered of top anti extrusion ring turning over and jamming sample in vessel top piece. Solution may be to leave out upper anti-extrusion ring and fit only lower ptfc ring. (Care should be taken to avoid scratching platens or breaking up sandstone surface which would mobilise sand dust).

4) Check large pressure vessel O-ring is is clean, undamaged and located well.

5) Top bleed valve should be closed and finger tight before pressure vessel is lowered onto sample.

6) Measure and note the distance between cap piece and top of furnace.
7) Trolley supporting sample assembly should be rolled into position under hanging pressure vessel ensuring that pressure vessel is high enough to allow clearance. (NB: take care not to pinch white control lead). Drop locating pins to secure trolley when in position.

8) Lower suspended pressure vessel over sample. Ensure locating holes are directly above pins (slight guidance/rotation of vessel by hand will normally be required). NB: Bolts must also locate well in holes and should protrude approximately 1" when vessel is correctly located. Cap piece must be observed during lowering of pressure vessel. If cap piece moves up farther than expected (expected dimensions are noted in section [11]), sample may not be located properly. If level of cap piece is thought to be anomalous when tension is off lifting links, DO NOT ATTEMPT TO TIGHTEN NUTS instead follow section [10] "unloading a sample" from #11.

9) If cap piece moves as expected, pressure vessel should be located correctly and lowered until lifting links are loose. Underside nuts should be tightened, in opposite pairs, firstly by hand, then using torque wrench (to 20kg). All nuts should be checked then lifting arms may now be removed and pulley hoisted clear.

10) Rig must be brought up under stroke control (follow section [1] "bringing rig up under stroke control"). Use mean level pot on stroke s.c.a. to raise ram if required to ensure clearance as pressure vessel rolled under on trolley. Roll trolley under piston (ensure that white control lead and blue hose/blue hose are not pinched during this manoeuvre).

11) Unbolt screws attaching pressure vessel to trolley.

12) Attach lifting arms to press, 2nd top holes should be attached first (link then washer, then nut finger tight), then ram should be moved up or down using mean level control until bottom holes are aligned. Attach bottom holes as before, checking lifting links are flush to sides of the cell.

13) Carefully lift whole cell by turning mean level stroke s.c.a. control clockwise until cell is clear of trolley. Roll out trolley towards front of room, then bring down whole cell onto base. Manual guidance/rotation of cell will normally be required to ensure good location in base guide. While lowering cell watch lifting links - as soon as top nut moves relative to hole, system is in compression i.e. has located in base plate. REMOVE BOTTOM LINK BOLT A.S.A.P. and SWING LINKS CLEAR before removing top bolts. (This prevents lifting bars being crushed and risk of injury to operator).

14) Rig can be switched OFF by pressing red PRESSURE OFF then PUMP OFF.

[3] CONFINING PRESSURE OIL INTENSIFIER

[a] Observations

Observations below were made with "pump" and "pressure" ON but confining pressure ramp controller not engaged.

1) Displacement zero pot on confining pressure adjusts the range over which the piston moves. Total range can be selected +/-50mm, +/-20mm etc. using the top pot on the displacement stroke amplifier. Turning the zero pot moves the oil intensifier piston up and down the range selected. Dialling the zero pot on confining pressure displacement does not change the pressure measured by the DPM, therefore we conclude that the zero pot does not in fact drive the piston but instead merely measures its position.

2) Confining pressure stroke amplifier zero pot can be used to dial the pressure up and down prior to the start of a test. DPM pressure reading changes when pot is dialled, but displacement does not change. We have assumed that this zero pot was a drive control for pressure but as the piston does not appear to move when we dial up the pressure, either the zero pot is not be a drive control or one of the transducers is not acting.
3) "Pump" and "pressure" must be ON before trying to monitor displacement of the piston on confining pressure intensifier e.g. when filling it up. Although red "reset" light is lit on ramp controller, pressure can still be controlled using zero pot on stroke amplifier.

4) Drive control is the leftmost set of controls marked RAMP CONTROL. This only works on a forward direction at low pressure i.e. to back off the intensifier you must have enough back pressure in the oil to push the piston down against the internal seal friction. To empty the oil intensifier you must set a positive pressure reading on the ramp controller and this will force the intensifier to drift upwards in an attempt to raise the oil pressure.

[b] OPERATION PROCEDURE

1) Require at least half a tank of "Shell Tellus 68 oil". Oil must be manually pumped from tank into electric pump (watch fill level on front of pump as this is being done). Electric pump is connected to oil intensifier via stop valve. Blue connecting lead to pressure vessel should be disconnected.

2) Switch ON electric pump then a few seconds later OPEN blue handled stop valve at top of the intensifier and oil will be pumped into the intensifier. (must allow a short amount of time for pressure to build up in pipe work). Electric pump should be filled up with oil using manual pump, if required, during this operation. Keep pumping oil until it flows freely and does not spurt out of the blue connector hose. This should mean that all air is out of the system. CLOSE blue valve then STOP electric pump.

3) Fit connector pipe onto underside of pressure vessel and tighten using adjustable spanner, then open bleed valve on top of pressure vessel. White bleed pipe must be connected to this valve via a tapered connector and washer to avoid spillage (if washer is not fitted, air may escape into valve and cause a flow of bubbles in white pipe which wrongly imply air is still in pressure vessel).

4) Switch ON electric pump then OPEN blue intake valve. Pump oil into pressure vessel using the electric pump until it flows freely from top bleed valve. Close blue valve, stop pump.

5) NB: Bleed pipe should be held up vertically to ensure oil settles naturally - bending it over as done in the past creates a siphoning effect and removes oil form the pressure vessel. Allow oil to settle and a few minutes for any air bubbles to flow along bleed pipe. Bleed pipe should be removed and tapered fitting left in place for a few more minutes to allow oil in vessel to settle. Now tapered fitting can be removed - level of oil should be near top of hole and no airlocks should remain (if this is not the case, procedure above should be repeated).

6) With top bleed valve open, pressure in vessel should be approximately atmospheric - therefore dial confining pressure zero pot to show 0.00 (on DPM), then screw on bleed nut and remembering washer. Tighten bleed valve using adjustable spanner.

7) Before bringing rig up under stroke control it may be necessary to put a small amount of oil into the confining pressure intensifier. If the piston is at the top of its range then the electric pump does not supply enough pressure to overcome the hydraulic pressure, therefore a small amount of oil should be pumped into the intensifier by switching electric pump ON then opening up blue valve. Whilst doing this DPM should be monitored on cp pressure and displacement. Blue valve should be CLOSED then electric pump switched OFF when the piston has moved a little.

8) Now bring rig up under stroke control (follow section [1] "bringing rig up under stroke control").

9) Ensure bottom and top bleed valves are well tight. Turn right hand side DPM to displacement - this monitors the position of the oil intensifier piston.

10) To fill up oil intensifier and run piston back, switch ON electric pump then a few seconds later OPEN blue valve. When blue valve is opened, the pressure indicated on the electric pump dial pressure gauge should drop. DPM should reduce in value, keep valve open until DPM value levels off to a constant value then CLOSE blue valve and STOP electric pump. (Constant value implies that piston has run back to its limit). Note this value and compare to previous runs. Check also the reading on DPM cp pressure - this should relate to the value on dial pressure gauge.
11) Confining Pressure controller can now be engaged.
12) PUMP and PRESSURE should still be ON.
13) To engage the confining pressure controller, first, zero the error meter (2 red arrows and bar) on the confining servo panel using the zero pot on the confining pressure amplifier panel.
14) Next, press black pressure button on ramp control. You should get a green light underneath showing that confining pressure controller is now locked in.
15) Remember at start of experiment, we want confining pressure zeroed i.e. level pot on ramp controller should read zero and DPM pressure reading should read zero. Check also that DPM confining displacement display is within 10 engineering units. If not, adjust using zero pot on confining displacement panel.
16) Confining Pressure can now be applied by turning leftmost level pot clockwise to increase, anti clockwise to decrease. V/MIN dial = ram rate of intensifier i.e. high rate implies almost immediate response to manual increase/ decrease in confinement. At this stage confining pressure is engaged and ready for use.
17) At start of test, level pot on ramp controller should read 0 on dial - if this is not the case adjust it to 0. DPM confining pressure should read 0 psi and error meter should show needle in centre of two arrows. Adjust pressure down to 0 using zero pot on confining pressure amplifier. DPM piston displacement should read some value between 0 - 10. It can be dialled into this range by using zero pot on confining pressure displacement.
18) Error trips (i.e. limit detectors) should always be set and switched ON before starting a test. see section[6] Limit Detector/ Overloads.

[4] COMPUTER CONTROL

The safety interlock on the pressure circuitry is governed by the RED & GREEN push buttons on the hydraulic controller (fourth rack down). This pressure interlock prevents the computer taking hold of the machine. When the computer is booted up it will send out all ZEROS to the equipment so that effectively it has no effect on the equipment, but be aware of the safety interlock built into the original hardware. Before engaging computer check that all error pots are within error limits set on limit detector. Also check confining pressure control pots, reset pressure if required.

In the case of an EMERGENCY, PRESS RED PRESSURE OFF then PUMP OFF BUTTONS (both underneath the control switch on hydraulic control unit 4th rack down, left hand side).

1) To bring up the computer, switch it ON at the back (right hand side) and switch the monitor ON. The software will boot up until you get to the Main Menu. Now insert a 5\textsuperscript{1/4} " disk into the disk drive.
2) First query is any change of date: Press [return] unless the computer clock has been down. New time prompt: press [return]. The system will then initialise.
3) The three menu choices are: [1]Create and Edit - To make a programme. [2]Block Programme run - to run the programme. [3]Exit to DOS - to get back into the operating system. Here, the computer is not connected to the hardware, it has no drive function because of the pressure interlock (explained above).
4) To bring the computer into synch with the hardware, turn the pressure OFF using the bottom RED PRESSURE OFF BUTTON (hydraulic control unit left hand side 4th rack down), turn the KEY
switch upright to COMPUTER, then turn pressure back ON using GREEN PRESSURE ON BUTTON. You should then have a Green Manual light on and a Red Computer light on, on the hydraulic control (left hand side 4th rack down). Now the computer can address the hardware.

NB: In the case of problems, manual control can be regained by turning the KEY switch to MANUAL. This should be done with pressure off if possible.

[5] CREATING / EDITING A PROGRAMME

1) Go to the Main Menu on the Computer and press [1] i.e. Create Edit. The first Menu is File Maintenance.

2) To drive the machine you must create a programme on number [2]. Options are ramping pore pressure, confining pressure or neither. For a simple programme Press [N] for neither. The control is a series of blocks which are timed ram sequences. The machine will run through block after block until it gets to an end when it will either repeat the block (if you requested this later in the programme), or will terminate the test.

NB: You are warned at the end that at the termination of test, the machine may go to the HARDWARE DEFAULT positions i.e. if you have left the HARDWARE in a ram down condition and the software has taken it to a ram up position, when the programme ends the ram will move to its HARDWARE SETTING.

WARNING: This can be dangerous. Before running a programme remember to check that the SERVO Controller Stroke SCA, (i.e. the left hand side, 3rd rack down) is set at its mid-point, mean level 5 on the Mean Level pot. This means that the ram is halfway retracted. The 5 on the mean level pot is on the range of the Stroke Amplifier of +/- 100 - (i.e. the stroke amplifier 2nd rack down left hand side).

3) First block of any programme must be a HOLD block to allow the computer to lock into the hardware i.e. stroke, 30 sec, mean level =0, cp =0

4) Mean level is equivalent to absolute position to which ram will move relative to ZERO, NOT the distance the ram will travel i.e. 0mm will move the ram to 0 position if it is anywhere else.

5) Number of blocks appears to be limited to 11 - if more blocks are written, the operations in blocks 12, 13, ...etc will be superimposed onto block 11 this could obviously be dangerous so must be avoided.

BLOCK 1 - To move ram 10mm in one direction over 10 seconds
Select ? (stroke lvdt, load, wait or end) [stroke] [return]
Duration ? i.e. How long do you want it to take to move from present position to position requested under mean level [10s] [return]
Mean level ? machine is zeroed by dialling 5 on the zero pot for stroke amplifier and on mean level pot. We wish to go to +10mm so [10mm] [return]
Acquire data ? in this case [NO] note : a disc is still required even if data is not acquired.

BLOCK 2 - To displace ram 20 mm over 20 seconds through zero to -10 mm
Select ? (stroke lvdt, load, wait or end)  [stroke] [return]
Duration ? [20s] [return]
Mean Level ? To move ramp from +10mm to -10mm position  [-10mm] [return]

BLOCK 3 - To return ram to its original position
Select ? [stroke] [return]
Duration ? [10s] [return]
Mean Level ? [0mm] [return]

BLOCK 4 - To end block programme
Select ? [end]

Do you wish to save programme ? [yes]
Filename ? [KAREN1]  NB: do not use file extension .blk
[return] takes you back to file maintenance programme

To Edit a Programme
On File Maintenance Menu select [3]

To Run a Programme
On File Maintenance Menu select [8] Main Menu
Select [2] Block Programme Run then [8] to load a programme
Title ? name of file without .blk extension  e.g. [KAREN1] [return]
(Cursor should skip across to next field)
Number of Times ? Number of times you want programme to run  e.g. [2] [return]
(Cursor should now skip down to left - if not, it cannot find the file so try again)
[end] [return]
Run the test ? [yes] [return]
Floppy disc in drive A ? [return] (programme will not run without a disc)
OK to use disc ? i.e. is it part empty ? [yes]
Name for output test ? software only checks first 4 letters of name  e.g. [tst2]

Hardware Ranges of the Computer will control
These are the ranges set on the servo amplifier control hardware. When a valid choice is selected the choice turns from yellow to black.

Stroke Range ? [100] [return]
LVDT Range  [100] [return]

Load 1 Range  kN dial setting on load amplifier 2nd rack down  [200kN]

Confining Displacement Range  Stroke displacement of piston in tank supplying fluid for confining and pore pressure, measured in mm. Dial settings on 6th and 7th racks down (ramp control, confining pressure, pore pressure) Here both are set at 5 so [5mm]

Move ram to mean level =5 by turning mean level pot slowly.

NOW SET ERROR TRIPS on hydraulic control unit and confining pressure servo according to section [6] "limit detectors / overloads".

NB: WARNING - the test starts immediately you press [return] on RUN (also note that it warns you TOO LATE that the actuator may move at the end of the test). Before pressing return you should check to make sure that you have not changed any of the settings from the beginning of the test such that the machine will now go into an overload condition.

To get out of the next screen, instead of putting in another programme, just type [END] [return]. This should return you to the main menu screen.


[a] Stroke/Load and Strain

1) These error trips are located on the limit detector on the HCU.

2) The six point limit detector unit monitors output voltages of stroke strain and load amplifiers and compares with pre-set "limits". Upper and lower limits can be set independently for stroke, load and strain. On exceeding a limit a red indicator will light up on the appropriate panel and hydraulic power will automatically be shut off.

3) The "upper limit" is the most positive (or least negative) voltage. The "lower limit" conversely is the least positive (or most negative value) and therefore must be algebraically smaller than the upper limit. e.g. upper may be set at +8V and lower at +2V or upper at +3V and lower at -1V.

4) Limit levels are set using the 0-10V pots on the left of each limit detector (on 4th rack down). Error limit should always be set at a level a little above the highest VOLTAGE expected during the test on each of the amplifiers. REMEMBER the voltage corresponding to the expected value will depend on the hardware settings selected on the amplifiers so should be checked for each test.

5) When rig is being run under STROKE control, stroke upper and lower limits can be set with good confidence i.e. relatively close to the displacement requested in the block programme. Load upper and lower can be estimated from previous tests.

6) Trip toggle switches should be down i.e. ON and alarm switches will normally be up i.e. OFF. +/- toggle switches will depend on the range of positive and negative voltages expected during a given test (see above #3 and also [12] for conventions).

7) After setting limit levels, "reset limits" black button should be depressed. All the red indicators on the limit detectors should be out.
[b] Confining Pressure

1) If using confining pressure always set and switch ON error trips before running a test. Error detector is situated on confining servo rack.

2) Limit level is set by turning 0 - 10 LEVEL pot to required value. 0-10 describes voltage i.e. equivalent to 0-10,000 psi with ramp controller set at x1 rate or 0-1000 psi if ramp controller is set at x0.1 rate.

3) Level should be set a little higher than the highest pressure expected during a test (therefore a new setting will be required for each test). TRIP toggle switch should be down i.e. ON and ALARM toggle will normally be up i.e. OFF. After level has been set, black RESET button should be pressed. Error light will then go out.

4) If an error limit you set is exceeded, RED ERROR light on Confining servo will show, RED RESET light will show and AUXIALIARY light on HCU will show. Error trip will automatically switch OFF pump and pressure.

5) To reset error detector, error meter must be in range. Dial down an error using confining pressure amplifier zero pot until needle is centred on meter. Check limits are reasonable for test conditions. Press reset on Confining pressure servo- red error light should go out. Before starting machine ensure cause of error has been recognised.

6) RESET HCU using RED PUMP OFF button - Auxiliary light should go out. If all error lights are now out (except ramp controller light which must stay on until confining pressure has been engaged which can only be done when pressure and pump are on), check all zero pots and mean levels - adjusting if necessary then "bring up rig under stroke control" according to section [1]. Confining pressure can be engaged by #10 section [3].

[c] Hydraulic Control Unit Interlock Lights

1) Red "overload" light on usually means that one of the pre-set limits on the six point limit detector has been exceeded.

2) Red "auxiliary" light on means an auxiliary unit error limit has been exceeded e.g. the confining pressure trip.

3) In both the above cases a red indicator will light up next to the error limit being exceeded. This may need to be dialled down into range using the appropriate pot before the limit detector can be "reset". Once the error has been reset, red "pump off" button on the HCU must be pressed to reset machine. A clear condition should resume.

4) Red "power pack" light refers to the pump unit (in white cabinet outside). Main line filter, return line filter, oil temperature, oil level red light should indicate where the fault originates.

NB: If you trip one of the overloads on the Load amplifiers 1 or 2, you must use the SERVO CONTROLLER STROKE SCA, 3rd rack down, left hand side, to bring it back to a safe condition before attempting to put it back on again. Failure to do this may result in severe damage to the load cells (and the operator!).

NB: The first limit with trip enabled which shuts off hydraulic power will inhibit subsequent operations of any further limits.
[7] RUNNING A TEST

1) Write a computer programme for test according to [5] "creating / editing a programme" and note blocks in lab book.

2) Measure dimensions and describe characteristics of rock sample. Ensure just prior to test that sample ends are truly at right angles to the sides using a Teesquare. (If ends are not right DO NOT LOAD SAMPLE - arrange to have sample lathed).

3) Load the sample following section [2] "loading a rock sample".

4) Fill up pressure vessel and oil intensifier and run piston back using section [3] "confining pressure oil intensifier".

5) Position protective shield in front of pressure vessel.

6) Get rig under computer control -see section [4] "computer control".

7) Dial down mean level pot on stroke s.c.a. to a value of 5. TAKE CARE - RAM WILL MOVE. Displacement will be measured from this position i.e. this is 0 mm for a value of 5 on mean level pot.

8) Engage confining pressure servo

9) Set error trips - see section [6] "limit detectors / overloads".

10) Bring ram down to 5mm above top of cap piece by turning zero pot on stroke amplifier SLOWLY. (If pot is turned too rapidly, displacement stroke DPM on left hand side will read a value which may exceed the error limits and trip out pump and pressure. Reset Error trips if they have been set "high" to allow ram positioning on coarse scale.

11) Select RUN A TEST.

12) Input hardware settings to computer and note on run sheet.

13) Check all pots, zeros, error detectors are ON. Before pressing RUN only one light should be showing at computer / manual key on HCU. If any other red lights are on DO NOT START TEST.

14) WARNING: Be aware that test will start immediately RUN [return] is pressed.

15) If all is clear RUN and [return] can now be selected.

[8] CHECKS WHILST RUNNING A TEST

1) The time at the start and end of each block should be recorded on log sheets.

2) S.C.A. error detector of controlling servo will show a needle deflection during run - this is the feedback error used to accurately position and re-adjust the actuator or intensifier piston and should cause no worry.

3) Periodically (approximately every 50 seconds) load, stroke, cp, cp piston displacement should be recorded.

4) Cp displ should stay essentially constant once loading block has started. Cp should stay constant throughout load block.
[9] ENDING A TEST

On ending a test you should ensure that the limits set on the six point error detector are "high" enough to allow movement of the ram clear without tripping out. It may be necessary to change these limit settings.

The ram should be returned to its median level. i.e. it should be at 5 at mean level on the SERVO CONTROLLER STROKE S.C.A. on the 100mm STROKE AMPLIFIER range. To do that you must run the STROKE Amplifier zero back up to 5. i.e. its pre-set median position.

1) At the End of a test, press [return] then type END [return].

2) First return the zero control of the stroke amplifier to 5. The mean level pot (3rd rack down) is only applicable to the +/- 100mm range so take care when adjusting this. If it is turned the wrong way you will overload the load cells. When the mean level pot and the STROKE Amplifier, (left hand side, 2nd rack down), is set at 5 you should be able to switch the range on that Amplifier between 10 and 100 without the piston moving as long as the mean level pot below it is also set at 5. In other words if the STROKE SERVO CONTROL AMPLIFIER is set at 5 for its mean level pot and the STROKE AMPLIFIER ZERO above it is set to 5, i.e. its median position and you can switch the STROKE AMPLIFIER range up and down without actually moving the piston.

3) NB: Don't forget that you are still under STROKE control. You can tell this by looking at the SERVO controllers on the 3rd rack down and seeing which one has a GREEN light on showing CONTROL mode.

4) Assuming that everything else is now set, press the RED PRESSURE OFF button, then press the RED PUMP OFF button, switch the CONTROL KEY to MANUAL and turn OFF RACK POWER SWITCH top right hand side.

5) Exit to DOS, [3] on the computer. At C prompt, switch off computer at back right hand side. Switch off the monitor. Switch off power supplies to the cabinet and computer at the back wall. Finally, remove the disk and leave beside computer.

6) Check the chiller has stopped i.e. oil is re-circulating is cool enough.

7) Turn OFF main power switch (major lever switch right wall beside flow switches).

8) Turn OFF chiller pump i.e. PRESS RED BUTTON then turn 1 to 0.

9) Turn OFF chiller (outside turn switch from 1 to 0).

10) Following the end of a test hydraulic oil must be drained from the equipment. CARE REQUIRED as some residual pressure may cause spray of oil as bleed valve is opened. WEAR SAFETY GLASSES. First open lower bleed valve and remove blue connector hose then attach 3" long transparent pipe. Open top bleed valve then drain into clean container for an afternoon or overnight. This process can be speeded up by pumping air into the top air bleed valve. Used oil should be filtered for further use.

[10] UNLOADING A ROCK SAMPLE

1) Bring Rig up under stroke control as described above.

2) Move top platen UP if it has slid down since last use by turning mean level pot stroke s.c.a. CLOCKWISE.
3) Connect two lifting links to either side of the top section of the cell by the 2nd top holes. Attach link then washer then nut and finger tighten. Ensure the link is resting on the block NOT the screw. Do not attempt to attach the bottom holes.

4) Move the cell UP (or DOWN) until the bottom holes are aligned with the bottom bolts. Loosen the top bolts if necessary then connect the bottom (and top ) parts of the link as described above. Check that the links lie flush to the sides of the cell and are not inclined at all.

5) Disconnect the pore fluid and confining pressure pipes from the cell. (Beware of leaking fluid). Disconnect the furnace leads. Check that remaining leads will not be pinched by movement of the cell.

6) NB: Do NOT disconnect white control leads as this will disconnect the control unit from the rig. The cell may move back to it's mean stroke level unexpectedly and may cause injury.

7) Lift whole cell (now connected together with the linking arms) UP by SLOWLY turning the MEAN LEVEL pot (stroke s.c.a) CLOCKWISE. Continue to check that both the linking arms are parallel to the cell sides and the white control lead is not in danger of being pinched. Continue to lift Up the cell until enough clearance exists between its base and the rail to roll in the support trolley.

8) Check that the trolley is the right way round by trying to bolt it into position below pulley. Then roll in TROLLEY beneath base of cell until bolt holes match up.

9) Lower cell DOWN onto TROLLEY by SLOWLY turning MEAN LEVEL pot ANTI-CLOCKWISE. The trolley position may need adjusting to match the holes with those on the cell base. Once the weight of the cell is taken up by the trolley loosely bolt the two together. (Look at the holes on the linking arms to determine if stress is acting down i.e. small gap at the top of the hole or up i.e. small gap at the bottom of the hole - here we wish a small gap at the top).

10) REMOVE the linking arms, disconnecting bottom link nuts first then upper nuts. TIGHTEN the BOLTS on the trolley. Move the ram UP (by turning MEAN LEVEL pot CLOCKWISE) until clear of lower cell.

11) Switch OFF "pump" and "pressure".

12) MOVE trolley plus load cell along tracks to the winch pulley system taking care not to pinch any remaining leads. LOCATE the trolley into position beneath the pulley. Replace top bleed valve and washer.

13) CONNECT load cell to pulley using linking arms (used previously). LOOSEN OFF large bolts on underside of load cell using TORQUE WRENCH in opposite pairs (to prevent stresses being unevenly distributed). Take care for oil and fluid leakages. Gradually work round and round the cell (loosening each bolt a little at a time) until bolts can be removed.

14) Apply pusher to prevent sample lifting when outer vessel is hoisted up. Slacken pulley system slightly and push arms to side to allow access to cap piece. Undo 4x cap screws in pusher head on top of pressure vessel using Allen keys and take out. Twist and lift cap piece off -NB: Care is required as cap is heavy. Be extremely careful of green wills ring inside cap piece. Put pusher down hole. Screw in 2x stub bolts as far as they will go , then lower pusher by winding gold dial up slightly. Place nuts on stub bolts and tighten.

15) Compress end spring by turning gold dial upwards. Screw until spring is tightly compressed (1/3 of way up should suffice).

16) Gently tighten up hoist then slowly hoist up pressure vessel. Watch that gold dial on pusher goes down as you are hoisting. Dial up gold dial again once it has reached the bottom of its range. Take care as nut on top of pusher will stop it pushing anymore.

17) Once pusher drops down easily (i.e. sample is loose from the vessel) pull up pressure vessel until clear of rock sample.
18) Undo positioning bolts then CAREFULLY ROLL trolley and lower part of cell along tracks until clear of the overhanging upper part.


**sample length**
- for good tests sample length: diameter ratio should be at least 2:1.
- maximum sample length allowable according to scale drawings is 245mm (9 5/8") this is with no spacers at top and bottom only upper and lower endpieces.
- shorter samples can be made up to correct length using combinations of steel spacers. Spacers available with central holes 1/2", 1", 1 1/2".

**piston throw**
- maximum throw (measured from scale drawings) is 45mm.

**cap piece clearance**
- distance between cap piece and collar should be approximately 25mm when no sample is loaded and 35mm with sample loaded and lifting links loose.

[12] Conventions

**stroke s.c.a.**
- mean level pots and zero pots
  clockwise = retraction of piston i.e. ram moves UP
  = positive displacement
  = positive load
  anticlockwise = ram extension i.e. ram moves DOWN
  = negative displacement
  = negative load

**load**
positive = tension
negative = compression

**confining pressure**
positive = increasing pressure
NB: PANIC BUTTONS - In the case of an EMERGENCY, PRESS RED PRESSURE OFF then PUMP OFF BUTTONS (both underneath the control switch on hydraulic control unit 4th rack down, left hand side).

Oil temperature in the pump: the temperature of oil in the pump should not rise above 55°C. To control this, there is a thermostat on the back bottom left side of the Oil Tank which is currently set at 55°C. If you get an unknown trip out anywhere in the system, one of the things to check for is that you have not exceeded 55°C on the oil pump.

Safety interlock on the digital panel meter, top left hand side, on the 1st rack. i.e. the red numbers left hand side. This panel meter is arranged from -10 to +10 Volts. It has a safety cut-out at around 10 Volts but this can be triggered early by e.g. pump surges. So, in any control mode, if you can't get the machine up, check this reading for the mode that you are trying to bring up the machine in, and make sure that you are within the 10V range. Otherwise pump will not even run.

If at any time during operation you hear the pump disengage PRESS RED PUMP OFF BUTTON.

Never disconnect white control lead whilst rig is ON.
APPENDIX D: LASER SIZE ANALYSIS

[1] The Coulter LS100 Laser Particle Sizer
The range of particle sizes detectable by the LS100 is constrained, on the lower side, by the wavelength of the incident laser light, which must be significantly smaller than the particle size being measured to avoid Mie scattering which leads to complex diffraction patterns. In this system, laser light at a wavelength of 750nm permits sizing in the range 0.4µm to 800µm diameter. The sample in a suspension fluid is passed into a 3mm wide sheet (diffraction sample cell) perpendicular to the laser beam to try to prevent particles interfering with each other. Two Fourier lenses are used to collect light diffracted up to 15 degrees and between 15 and 35 degrees respectively. Three sets of photodiode detectors, for low angle, mid angle and high angle scattering (126 detectors altogether) receive the light focused from the appropriate Fourier lens. One advantage of the Coulter LS100 over other sizers is that it can simultaneously measure the whole range of sizes. The computer uses a set of proprietary complex algorithms to calculate the particle size distributions from the composite diffraction patterns measured at the sensors. These routines presumably compare the diffraction patterns measured at the sensors to a theoretical model of the scattering patterns predicted for different sized quartz spheres. The error between the measured results and the model would then be reduced by e.g. a least squares regression or some alternative inverse method, to deduce the best size distribution to fit the data.

[2] Sample Preparation
Deformed cylinders of rock were removed from their rubber sleeves then opened up along the fault zone. Fault zone material (gouge or fractured pods) was then carefully scraped from fault zones and collected in a clean beaker of known weight. The white powdered material occurring in the fault zones as strands was defined as gouge, whereas the lenticular areas of red material were defined as fault zone pods.

In order to disaggregate specimens, they were mixed with distilled de-ionised water then agitated gently with a glass rod. Although any agitation may alter the particle size distribution by breaking grains along incipient microfractures, it is believed that using a downward pounding motion (rather than a twisting grinding motion) is likely to separate aggregate grains rather than to crush single grains into smaller fragments (Krumbein & Pettijohn chapter 3, 1938), hence this method was used. Specimens were then shaken in a sonic bath for a few minutes at room temperature prior to measurement. This is a standard technique used to prevent the flocculation of smaller particles and should not damage larger grains. Any aggregate surviving and still held together at this point is considered as a single particle. At this stage, the separate specimens of fault zone pods and host rock were wet sieved using an 800µm sieve to ensure no large pieces, which could damage the sizer, remained. Although this appears to be biasing the sample measured, individual pieces greater than...
800\(\mu m\) are very unlikely to be single grains as the undeformed host rock does not even have grains this large. The maximum number of pieces removed from any one specimen during this procedure was less than 5 and hence not significant for the distribution. Gouge strand specimens did not need to be sieved to 800\(\mu m\) as they had visibly smaller sized particles. It was difficult to completely purge the fault zone pods of fine grained gouge material so the measured grain size distributions may actually reflect a combination of both the pods and gouge material.

The disaggregated specimen was placed into the sample vessel with distilled de-ionised water then rapidly pumped round the closed system. Obscuration and flux were measured and noted during this procedure. For optimum analysis, an obscuration (relative amount of particles) of 10\% was required, which correlated to approximately 0.5g of fine gouge sample. These measurements took approximately 10 minutes and were necessary to check the background levels of the equipment were all normal.

[4] Calculating the data
The data presented from the system following calculation of the best fit size distributions was in the form of a binheight (intensity) and bin diameter (size range). The binheight number could be converted into volume or number using the following equations:

\[
Volume = \frac{\text{binheight}}{(\text{bindiam}/2)^3} \quad \text{equation: E.1}
\]

\[
Number\% = 100 \times \frac{Volume}{\sum Volume} \quad \text{equation: E.2}
\]

Channels (or bins) had a finite width which increases with increasing grain diameter so that the upper bound of each bin was 12\% greater than the lower bound. This is in accordance with the traditional and standardised method of using the phi scale to analyse grain size distributions. The equation relating the Phi scale to millimetres is shown below where \(d\) is diameter in mm.

\[
\phi = -\log_2 d \quad \text{equation: E.3}
\]

This log base 2 scale is convenient to use on geological data, as it commonly plots up as gaussian distributions. Gaussian distributions are simple to characterise hence comparison between different sets of data is straightforward. Here the data is plotted in microns (\(\mu m\)) and the Phi scale is used for
the relation of increasing widths of bins with grain diameter. 72 channels or bins cover the whole range of particle diameters from 0.4µm to 800µm, and are listed in below.

[5] The effects of sonicating

Figures:E.1-E.4 Graphs of volume (%) versus diameter (µm) for laser particle size data, each showing one sample which has been subjected to different amounts of sonicating, in order to assess any artefacts introduced by carrying out this process immediately prior to sizing.

Figure:E.1 shows volume (%) as a function of particle diameter (µm) for sample 2d. Curve 2dfs01 is for a gouge strand specimen which has undergone no extra sonicating. The curve peaks at a value of 400µm which is actually in excess of the peak of undeformed Locharbriggs sandstone curve. This is obviously unexpected and implies that small gouge particles are flocculating (coagulating) together to form aggregates. After 60s and 180 s of sonicating, shown by curves 2dfs02 and 2dfs03 respectively, a significant shift left gives a more reasonable mean grain size of e.g. 9.97µm for sample 2dfs02. There are clearly no grains larger than 200µm and there is little change observed between the sample sonicated for 60s and 180s. The dramatic change when no extra sonicating is carried out indicates that the sample will coagulate rapidly and requires extra sonicating immediately prior to grain size analysis.

Figure:E.2 shows data for sample 5d and 5dfs (fault zone pods and fault gouge strands respectively). Little change with is observed with increased amounts of sonicating except to accentuate the main features of the curves more. The mean grain sizes in 5d01and 5d03 are 154µm and 145µm respectively, indicating that sonicating does not significantly alter the grain size distributions although there is a very slight shift of the mean peaks to the left with increased amounts of sonicating. Similarly, samples 5dfs01 and 5dfs02 both show their most significant peaks at 9.97µm.

Figure:E.3 shows volume (%) versus grain diameter (µm) data for the fault zone pods of sample 1d which has been deformed to 6% axial strain. The graph shows curves with approximately 4 peaks, the largest volume occurring at 31µm. The effects of increasing amounts of sonicating from 0s through 60s to 180 s are presented in curves 1d01, 1d02 and 1d03 respectively. The effects of the increased sonicating are to isolate the individual peaks slightly and shift the peaks to the left by a very small amount, the main peak occurring at 29µm, 26µm and 24µm respectively. None of the peaks exactly correlate with the undeformed Locharbriggs peaks but they are of a similar order.

Figure:E.4 showing data from sample 6, highlights the largest shift of mean grain size (from 26.171µm to 11.1µm) between the curves for 0s and 60s sonicating for the gouge sample. Material from fault zone pods shows much less change in grain size with sonicating although there is an absence in larger particles with more sonicating. The shift left is of higher magnitude than in samples 5d and 1d.
### Bin sizes

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<td>3.8</td>
<td>44.732</td>
<td>526.57</td>
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<tr>
<td>4.23</td>
<td>49.794</td>
<td>586.15</td>
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Figure: E.1 shows gouge strands (2dfs01, 02, 03 - correlating to 0s, 60s, 180s sonicating) and fault zone pods (2d01 - correlating to 0s sonicating) for run2.

Figure: E.2 shows gouge strand (5dfs01, 02 - correlating to 0s, 60s sonicating) and fault zone pod (5d01, 02, 03 - correlating to 0s, 60s, 180s sonicating) data for run5.
Figure: E.3 presents fault zone pod data (1d01, 02, 03 - correlating to 0s, 60s, 180s sonicating) for run1.

Figure: E.4 presents run6 data for both gouge strands (6d01, 02, 03 - correlating to 0s, 60s, 180s sonicating) and fault zone pods (6dfs01, 02, 03 - correlating to 0s, 60s, 180s sonicating).
APPENDIX E: IMPREGNATED THIN SECTIONS OF 4 INCH FRACTURED SANDSTONE CORES

The cores must be impregnated with coloured resin before removal from the rubber sleeves. Araldite MY778 with hardener HY951 was used, along with a 20% by mass Araldite DW0135 blue dye.

The lower end of the core was sealed with PVC tape to produce an air tight seal and the core placed in a vacuum chamber. The resin mixture was poured onto the top of the core to a depth of 5 to 10mm. A vacuum of around 150 mbar was applied and left for around 40 minutes. The vacuum was then slowly released and the cores placed in a hot cupboard (approximately 60 degrees centigrade) to cure the resin. Once the resin was cured the top edge of the rubber sleeve could be rolled back and a 5 to 10 mm slice was cut of the end of the core to give a rock slab suitable for preparing a thin section from.

In most cases it was found that the resin had not penetrated the full depth of the rock slice over the entire area of the slice. This was partly due to the blue dye increasing the viscosity of the resin and partly to the preference of resin to follow the easiest path through the rock - along the largest fractures and between the rock and rubber sleeve.

During cutting of the slice from the core significant damage was often done to the cut face of the slice by the cooling water washing out the fine grained crushed material in the fracture zones which had not fully absorbed resin during impregnation.
### THIN SECTION POINT COUNT ANALYSIS

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<th>Test identifier</th>
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<td>Rocktype</td>
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<td>Transect number</td>
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<tr>
<td>Number of Counts</td>
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<tr>
<td>Magnification (x4, x10, x40)</td>
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<tr>
<td>Anglelity Index</td>
<td>Very Ang.</td>
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<td>Petijohn (1973)</td>
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#### Mean Diameter

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<tr>
<th>Volume %</th>
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<tr>
<td>Strain</td>
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<td>182.24</td>
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#### Neighbouring Grains

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<td>N = Grain</td>
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<tr>
<td>N &gt; Grain</td>
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#### Orientation of Fractures in degrees 000-000 stress

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<tr>
<td>Subtotal</td>
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### Appendix F

#### Visual basic template for processing raw point count data.

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<tr>
<td>Minor Fractures / Grain</td>
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<tr>
<td>Mean Diameter</td>
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</table>

#### Mean Diameter

<table>
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<tbody>
<tr>
<td>Minor Fractures / Grain</td>
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<tr>
<td>Total Fractures / Grain</td>
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#### Minor Fractures / Grain

<table>
<thead>
<tr>
<th>Minor Fractures / Grain</th>
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<tbody>
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</table>

### Micrometre ticks

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#### Area G

<table>
<thead>
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#### Area P

<table>
<thead>
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<tbody>
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